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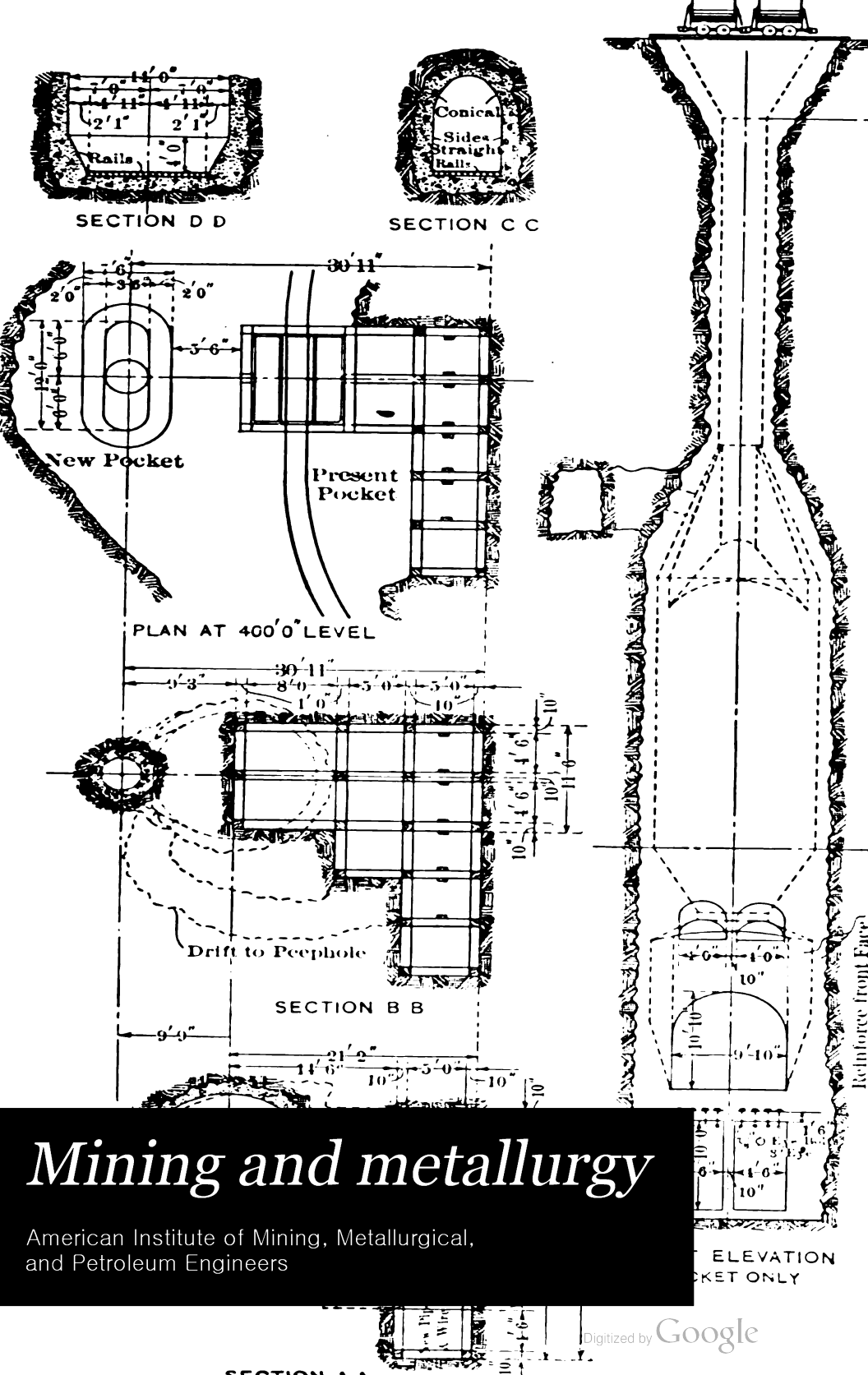


# *Mining and metallurgy*

American Institute of Mining, Metallurgical, and Petroleum Engineers

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AUGUST, 1914

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# Bulletin of the American Institute of Mining Engineers

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# Melting of Cathode Copper in the Electric Furnace*

BY DORSEY A. LYON AND ROBERT M. KEENEY, PITTSBURG, PA.

(Salt Lake Meeting, August, 1914)

## INTRODUCTION

THE electric furnace has always been found to be especially adapted to melting, refining, and finishing processes throughout its gradual acceptance by metallurgists as a practical apparatus for conducting metallurgical operations. In the steel industry, the electric furnace is firmly established in the manufacture of steel of the highest grade, equal to crucible steel. For the production of the cheaper grades of steel in large tonnage, the electric furnace gives a higher-grade product than the open-hearth or converter, but due to the high cost of power prevailing in steel centers, considered from the electric furnace standpoint, it is not economical to produce tonnage steel in the electric furnace. As a refining and finishing agent for open-hearth or Bessemer steel, the electric furnace has had some degree of success in the production of tonnage steel.

After a study of the use of the electric furnace in the steel industry, it appears that there may be a possibility of its use to advantage for the melting of cathode copper. In the case of copper the problem is not one of actual refining, because the copper if refined properly by the electrolytic method needs no further refining; it needs simply to be cast into a marketable shape. The name "refining" is applied to the present finishing process, because in the operation oxygen and other impurities are removed, which the electrolytic copper absorbs in being melted in the reverberatory furnace. Considering that processes may be positive, negative, and neutral, the electrolytic method would be positive; reverberatory melting of cathode copper, negative; and electric furnace melting, neutral. The electrolytic method produces about as pure a copper as is possible, if operated properly and the anodes are not too impure. The final reverberatory melting of this cathode copper lowers the grade of the final product, due to the absorption of gases and impurities.

The reverberatory melting of cathode copper is one of the weak parts

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of the metallurgy of copper to-day. A refining process is not required, for before the copper is melted in the reverberatory furnace it is pure. The ideal process would be one in which the cathode copper could be melted and cast into marketable shapes without absorbing impurities; *i.e.*, a neutral process.

### Reverberatory-Furnace Refining

As cathode copper is practically pure copper, it would be ready for use if in a marketable shape, but as it is not, the cathodes must be melted. Common practice is to melt cathodes in the reverberatory furnace. The practice at Great Falls, Mont., has been described by W. T. Burns[1] as follows:

Two coal-fired reverberatory furnaces of 100,000 lb. capacity each per charge are used for melting cathodes. The coal used in firing them contains about 3.5 per cent. sulphur. On account of this high sulphur content in the coal, it is necessary to protect the copper as it melts from the sulphur dioxide gases resulting from combustion of the fuel. In order to do so about 30 per cent. of the cathodes going to make up a wire-bar charge are dipped in milk of lime before charging. As the copper melts the lime forms a protective coating for the metal, thus hindering in a large measure the absorption of sulphur by the molten copper. Just before rabbling (or oxidizing) is started the slag is skimmed from the furnace. The slag obtained is equivalent in weight to about 3.5 per cent. of the copper charged, varying with the amount of lime added. The slag contains about 60 per cent. copper.

After the charge is completely melted, the usual practice is followed of rabbling and poling the copper for oxidation of the impurities and bringing the metal to the proper pitch for casting. Rabbling is effected by introducing compressed air at a pressure of 16 lb. per square inch into the molten bath. The poling and reducing action is obtained by forcing green pine poles into the bath and by covering the surface of the bath with charcoal.

The condition of the bath, as regards oxygen content, is noted from time to time by the refiner as he examines the successive button samples taken from the bath in a small ladle. When the physical appearance of the button indicates that the "tough pitch" stage has been reached, the poles are removed and the dipping operation is begun.

### Why Reverberatory Refining is Superfluous

The process of reverberatory melting of cathode copper as above outlined is not a refining process. That it is so called is due to the fact

[1] Burns, W. T.: Notes on the Great Falls Electrolytic Plant, *Bulletin* No. 80, Aug., 1913, p. 2011.

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that it is necessary to remove the oxygen and other impurities which the electrolytic copper absorbs during melting in the reverberatory furnace. That the furnace refining of copper is superfluous in so far as it improves the grade of marketable copper has been shown by the experiments of E. Keller.[2]

As a result of these experiments Keller states that "electrolytic copper which has not been melted and refined is superior in conductivity to the refined and cast copper by over 2 per cent. There is, therefore, a field for improvement in the refining or making of wire from electrolytic copper without previous melting." Peters states[3] that "ordinary electrically deposited copper produced on a large scale at a plant using the Emerson system of depositing sheets direct without subsequent furnace refining showed a conductivity of from 102 to 103 per cent."

Another test[4] showing the furnace-refining operation to be superfluous, or even harmful, was made by rolling wire from a mass of Lake Superior native copper and, after annealing it, comparing it with cathode copper and furnace-refined native copper. The mass copper and cathode copper each had a conductivity of 102.5 per cent., while furnace-refined native copper had a conductivity of 99.5 to 100 per cent.

## Effect of Cuprous Oxide on Copper

As before stated, in the reverberatory process, while the copper is being melted, it absorbs oxygen, sulphur and other impurities from the gas passing over it from the fire box of the furnace, and when air is blown into the molten copper, until about 5 to 6 per cent. cuprous oxide is in the charge. This cuprous oxide gives up its oxygen to the metallic impurities in the copper, forming oxides, which pass into the slag. After the removal of all the sulphur dioxide gas the cuprous oxide is reduced to metal by poling until the button sample shows the proper structure and the copper is tough pitch, when it is cast. The cathode copper contains less than 0.1 per cent. cuprous oxide. After furnace refining it contains 0.4 to 1.2 per cent. cuprous oxide, with an average of about 0.7 per cent.

It seems that under the present conditions of refining, cuprous oxide is essential to the quality of the copper produced, as is shown by the fact that if the poling is continued below certain limits the quality of the metal becomes impaired.

It does not appear, however, that the presence of cuprous oxide itself has the direct effect of improving the quality of the metal, as mass Lake copper without previous melting shows a higher all around standard than

[2] Keller, E.: *Mineral Industry*, vol. vii (1898); Peters, E. D.: *Principles of Copper Smelting*, p. 484.

[3] *Principles of Copper Smelting*, p. 484.

[4] *Idem*, p. 485.

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furnace-refined Lake copper. Hampe[5] found that the addition of cuprous oxide produced no preceptible effect upon the strength or malleability of pure copper until 0.45 per cent. was added, when a slight diminution of tenacity was observable. With increasing proportions of cuprous oxide, the quality of the copper suffered more perceptibly.

Addicks[6] found that the addition of 0.05 per cent. oxygen (0.44 per cent. $Cu_2O$) to pure copper increases slightly its conductivity, which drops back again to about normal when the addition of oxygen reaches 0.1 per cent. (0.9 per cent. $Cu_2O$) and decreases considerably by further addition.

The theory has been advanced that the presence of cuprous oxide improves the quality of the copper by preventing the reduction of the dissolved foreign oxides, harmless while oxidized but injurious when reduced to the metallic state. These impurities may also be reduced during over-poling to reduce the cuprous oxide.

Peters states, "We may, I think, say with safety that, while the proportion of cuprous oxide found in ordinary good refined copper does not appear to diminish its electrical conductivity (it may even increase it slightly), the very highest conductivity tests are yielded by the copper which contains no determinable oxygen and that cuprous oxide, in the proportion usually found in good refined copper, appears to have but little effect, one way or the other, upon the malleability, ductility, or electrical conductivity of the metal."

## Conditions Necessary for the Electric-Furnace Melting of Cathode Copper

From the preceding discussion of present practice in melting cathode copper, it can be seen that the ideal process for melting cathodes is a method by which it is possible to produce cheaply in marketable form as pure a copper as is charged into the melting furnace. Such a process to be successful both metallurgically and commercially should conform to the following conditions:

1. In order to keep the percentage of impurities as low in the final product as in the cathode copper charged into the furnace, there should be a neutral atmosphere in the melting chamber with a minimum possibility of introduction of air, gases, or impurities which might contaminate the copper.

2. Melting should be performed without excessive losses of copper, either by volatilization or in any slag which cannot be returned to the blast furnace or reverberatory furnace for resmelting.

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[5] *Zeitschrift für Berg-Hütten und Salinenwesen* (1873, 1874, 1876); Peters, E. D.: Principles of Copper Smelting, p. 491.

[6] *Trans.*, xxxvi, 18 (1905); Peters, E. D.: *Principles of Copper Smelting*, p. 491.

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3. The process should be susceptible to ready mechanical manipulation for charging and tapping.

4. The cost of production should be at least as low as by existing methods, which means that the furnace must handle a large quantity of copper per day with an efficient use of labor, fuel, and power.

The reverberatory process possesses all of these qualifications with the exception of the first, for, as has been shown, air, gases, and impurities are absorbed by the copper and must be removed as far as possible later in the operation. It is not simply a neutral melting furnace. It melts copper cheaply, is easily charged and tapped, and has no apparent loss by volatilization, but produces a poorer grade of copper than it receives.

### *Factors Governing Use of the Electric Furnace*

The use of the electric furnace for melting cathode copper would depend largely upon the cost of operation and the loss of copper by volatilization. The cost of operation would be influenced chiefly by the cost of hydro-electric power in comparison with the cost of coal, with the higher efficiency of the electric furnace in its favor. A loss by volatilization has been found to occur by experimenters using the direct-arc type of furnace with the copper covered with slag. There has been no loss in this manner with the indirect-arc type of furnace, heating by radiation of the arc, or in the resistance furnace.

This is a question requiring further experimenting. A large electric furnace is easily manipulated both for charging or discharging, as has been found to be the case with electric steel furnaces, and lends itself readily to the use of mechanical devices for charging cathodes and casting the copper.

### *Advantages Offered by the Electric Furnace*

Disregarding the commercial economy of an electric furnace for melting copper, the electric furnace provides an absolutely neutral melting chamber with no possible introduction of air or impure gases of combustion during the melting of the copper. Hence, since electrolytically deposited copper is practically pure, the copper melted in the electric furnace should be as pure as cathode copper, and subsequent oxidation and poling are not necessary to remove impurities acquired in melting, as none are acquired. That a product entirely free from oxygen can be produced by melting copper in the electric furnace has been shown, as stated in a previous paper by the writers.[7]

Native copper concentrates which contained about 35 per cent. copper were melted in an arc furnace, the slag being formed by the gangue

[7] The Smelting of Copper Ores in the Electric Furnace, *Bulletin* No. 80, Aug., 1913, p. 2117.

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of the concentrate with a suitable flux. As a result of 17 tappings of the furnace, copper was produced in which no trace of oxygen could be found after etching with picric acid and studying the results with the microscope. To confirm these results, the same samples were submitted to E. S. Bardwell, of Great Falls, Mont., for etching with hydrogen and determination of oxygen by his method.[8] After an examination of 12 samples Mr. Bardwell confirms the results of the writers by reporting that he can discover no oxygen in the samples after etching with hydrogen.

## Types of Electric Furnaces that Might be used for Melting of Cathode Copper

There are four types of furnaces which might be used for the melting of cathode copper: (1) the induction furnace; (2) the resistance furnace; (3) the direct-arc furnace; and (4) the indirect-arc furnace.

Of these furnaces the *induction furnace* is not practical, for the following reasons: (1) The resistivity of the copper is so low that the current necessary to produce the required heat must be so great that the pinch effect severs the column of molten metal; (2) the induction furnace is not adapted to melting cold metal; and (3) the size and shape of the hearth would make it difficult to charge large quantities of cathodes. It is stated by one authority that when copper is melted in the induction furnace, the column breaks off so often that it is necessary to have a crucible of molten metal ready to pour into the break.

The *crucible resistance furnace* with a solid resistor is too small for melting large quantities of copper, and as now constructed is not practical in large sizes. A resistance furnace of the type used by FitzGerald,[9] with a conducting roof which acts as a resistor and from which heat is reflected to the metal beneath, would not be practical in very large sizes because of the weakness of the roof. Also this type of furnace was found to have a low thermal efficiency.

Electric furnaces of the direct resistance type in which the molten bath is the resistor have not been successful in the steel industry, and attempts to use this principle have been abandoned. The recent use of the "*pinch phenomenon*" in resistance furnaces with a molten resistor, however, has been successful in small installations. Even in melting steel in such a furnace it is necessary to use a very low voltage to get the high amperes necessary for heating. The transformers are attached to the bottom of the furnace, which, while all right in small sizes, might cause

[8] *Bulletin* No. 79, July, 1913, p. 1429.

[9] FitzGerald, F. A. J.: *Transactions of the American Electrochemical Society*, vol. xix, p. 273 (1911); vol. xx, p. 281 (1911).

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complications in a large furnace. With copper in the furnace an even lower voltage would be necessary, adding to the difficulties of design. For molten steel a voltage of from 6 to 10 volts has been necessary.

The pinch furnace does not seem adapted to a large capacity per charge, and as yet has not been built in sizes approaching arc steel furnaces. While it might be used for melting about 500 lb. of cathode copper per charge, with short successive runs, this scheme does not seem practical for a plant melting 100 tons of copper per day. When copper was tapped at intervals of about 10 min., it would be difficult to use large-scale charging and casting machinery, which reduces the labor cost very much. The wear on the lining of the furnace would be greater than when a large furnace is used and discharged at intervals of several hours.

Another disadvantage of such a furnace is that in reverberatory melting large amounts of copper are absorbed by the lining, and this would doubtless be the case also in electric-furnace melting. This would prove a great disadvantage to the use of an electric furnace with a conducting hearth of any sort. While the hearth of the pinch furnace is not conducting over its entire cross-section, it is conducting at the points where the injectors or electrodes are placed in the hearth. After absorption of copper by the hearth short circuits might occur between the electrodes, and if the furnace were relined before affairs could reach this point, it would be a case of relining the furnace very often, at great expense. In the case of a furnace with a non-conducting hearth, the copper could be permitted to accumulate on the hearth without injuring the operation of the furnace.

In regard to the use of the pinch furnace for melting cathode copper, there is no question that it gives the best metallurgical conditions for carrying on the operation in a neutral atmosphere, but, in the opinion of the writers, it does not seem, from present developments, to be certain of success for this purpose in a practical way on a large scale. There are too many problems to be worked out in regard to the practical application of this type of furnace to warrant its use for tonnage melting at the present time. It provides a neutral atmosphere, with no chance of the introduction of impurities, and has one advantage over the direct-arc type of furnace in that there is not the necessity of keeping a bath of slag over the copper to prevent loss by volatilization. It is practically impossible for any volatilization of copper to occur in a pinch furnace.

Another possible difficulty in the use of the pinch furnace on a large scale is the necessity of keeping a molten bath of copper in the furnace in order to operate it. In charging, cold solid copper would be placed into hot molten copper. If there was any splashing of the copper, there might be a tendency for copper to collect on the roof, as the roof is the cold part of the furnace, since all the heat is generated in the metal charged.

The *indirect-arc furnace* in which an arc is drawn between electrodes,

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and in which the passage of the current is entirely independent of the molten bath, has the advantage of having the heating independent of the slag or metal. In steel melting the very basic slag used tends to promote the steadiness of the arc. Hansen[10] states that the arc in a zinc furnace or brass-melting furnace of this type is rather snappy and unsteady. This is apparently not fatal, for in his experiments with the Weeks furnace the arc was operated for 42 hr. continuously in an atmosphere of zinc vapor. The electrodes were not adjusted once during this time.

For copper melting on a large scale a furnace with from three to six electrodes probably would be used. Under such circumstances it would be necessary to have the electrodes controlled hydraulically by hand, as in the indirect-arc steel furnace, or by automatic electric control. From experience with the indirect-arc steel furnace it is evident that with furnaces of this type difficulty occurs in the regulation of electrodes and the regulation of the arc for a capacity of over 2 tons per charge. The larger size seems to present complications too great to permit of satisfactory operation. The 5-ton furnace built at Turin failed largely for this reason, and in steel melting it has been evident that the indirect-arc furnace is not adaptable in sizes of over 2 tons, or 300 kw. power load. In his first trials with the electric furnace, Stassano built a rectangular hearth furnace with six horizontal electrodes arranged in three pairs, lengthwise of the furnace. This furnace failed largely because of the difficulty in maintaining the arc and the lining.

One of the greatest objections to the indirect-arc furnace for steel manufacture has been the high repair cost for linings resulting from the direct radiation of the heat of the arc. Magnesite linings, which increase the expense considerably, are necessary. The heat losses in water-cooling the electrodes are also larger than in some other types of electric steel furnaces.

As a result of experience with the indirect-arc steel furnace, while there would be less chance of loss of copper by volatilization than in the direct-arc type, it is believed that the furnace would be too complicated for practical work; the up-keep cost too expensive; and the heat losses high. This furnace, while giving the proper atmosphere for carrying on melting of copper, does not seem to be a practical, efficient furnace for large-scale work.

The *direct-arc furnace*, in which the arc is formed directly between the electrode and the slag or metal, may be of the conducting or non-conducting hearth type. The conducting hearth type is not feasible for copper melting, as before stated, because it has been found that the combustion reverberatory furnace used in copper melting absorbs large quantities of metallic copper. Also there is always the danger of metal

[10] Hansen, C. A.: *Transactions of the American Institute of Metals*, vol. vi, p. 110 (1912).

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breaking through the bottom of a conducting hearth, which would be more apt to occur in the case of copper than steel, on account of the lower melting point and greater fluidity of the metal.

After a careful study of the existing types of electric furnace, it is believed that the most practical furnace for copper melting would be the direct-arc type, having a non-conducting hearth, as for example the Heroult steel furnace. As an efficient, practical furnace, this furnace has demonstrated its superiority in the metallurgy of steel. The objections to the direct-arc type as not affording theoretically the best possible conditions for copper melting do not, in the opinion of the writers, overweigh the practical points in its favor.

There are three chief disadvantages to the direct-arc electric furnace for the melting of cathode copper:

1. In order to prevent volatilization it is necessary to keep slag on the bath at all times when current is passing through the furnace.

2. From experimental work to date, it appears that there is a slight volatilization of copper with a furnace of this type even when a slag is used.

3. The presence of slag makes it difficult to discharge the furnace and keep the copper at the proper temperature for pouring.

With the exception of loss by volatilization, these objections do not seem serious, and the volatilization loss should be very small if the furnace is operated largely as a resistance furnace with a thick enough bed of slag (2 to 3 in.) to prevent direct arcing with the metal, and a low current density in the electrode to reduce excessive local heating at the points where the electrodes dip into the slag. Hansen states that in a newly lined furnace not designed for copper melting he tapped 99 per cent. of the metal charged. The actual loss could be determined only after a campaign of several months. If loss occurred, it would probably be at the start of a run when the slag is cold and not fused.

The question of discharging the furnace is a problem of operation. The copper could be cast by hand from a ladle, as is done at Great Falls. In this case it would be necessary to preheat the ladle and skim the slag from it before pouring, if the slot tap hole usually used on reverberatory melting furnaces is employed. Or a preheated bottom-casting ladle could be used, as in making steel castings. A siphon tap hole might be designed so that the copper drawn came from the bottom of the bath, leaving the slag on top of the metal. This tap hole would be lowered gradually, as in present melting practice, taking the metal from the bottom, but with only the pressure of the surface of the bath behind it. Only the last of the copper drawn would contain slag, and the current could be maintained on the furnace enough to keep the copper at the proper casting temperature until the slag was lowered to the bottom of the furnace.

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Thus bottom casting, if preferable to skimming in the ladle, would not be necessary except at the end of the operation.

For machine casting, with a siphon tap hole the operation would be performed as in reverberatory practice, except that it would be necessary to skim the last ladles or use a bottom-casting ladle on account of the slag. It is simply a question of manipulation.

In the electric process of melting cathodes with the direct-arc furnace, the charge would be placed in the furnace mechanically with a charging machine, as in reverberatory practice. The electrodes would be drawn up during charging. After the cathodes were all added to the furnace, lime and silica would be shoveled in to form a protective covering of slag. The slag should be of as low a melting point as possible. The electrodes would then be lowered, the furnace shut up and the circuit closed. The electrodes would be regulated by hand at first and later by Thury regulators, as in steel furnaces. Under these circumstances, as the slag is practically pure and no air is admitted into the furnace, poling is not necessary and the furnace can be tapped when the charge is all melted and at the proper casting temperature. During tapping the electrodes would be gradually lowered and the current kept on, so as to maintain the proper temperature. When operated with a slag 2 to 3 in. thick, the furnace is really a resistance furnace rather than a direct-arc furnace.

Because of the small amount of work done with melting of copper in the electric furnace, accurate estimates of the cost of the process are difficult to obtain. It is probable that the power consumption would not be over 300 kw-hr. per ton of copper melted. Using this figure and basing other expenses on electric-furnace steel practice, the cost of melting cathode copper in a furnace of 25 tons capacity per charge should not exceed $4.75 per ton, or 0.238c. per pound, of copper melted, with power at ½c. per kilowatt-hour.

## Conclusion

This short discussion of the subject is not offered with the idea of attempting to convert those interested in the matter to the belief that the only thing to do is to melt their copper in an electric furnace, but rather to suggest that the electric furnace might be used for that purpose.

We also hope that the paper may be the means of starting a discussion of the subject, by melters of copper and electric furnace men alike, for, as is well known, quite often it is not the paper itself, but the discussion of it, that makes it valuable, and so we hope it will be in this case.

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# Lead Smelting at East Helena

BY EDGAR L. NEWHOUSE, SALT LAKE CITY, UTAH

(Salt Lake Meeting, August, 1914)

THE lead smeltery at East Helena, Mont., controlled by the American Smelting & Refining Co. since 1899, has been in continuous operation for the past 25 years. Most of the old smelting and roasting practices have been superseded by newer and more modern methods, but on account of the difficulties involved in changing the original construction, some of the methods in use of handling materials are not the most modern. This plant, however, is particularly interesting as being one of the few essentially lead smelteries in the United States. At Federal, Ill., Collinsville, Herculaneum, and possibly other places, similar work is being done, but it is doubtful whether any of them produces as much lead as the East Helena plant, whose monthly production averages 6,500 tons.

The situation, 5 miles east of Helena, is convenient to both the Great Northern and Northern Pacific railroads, while the electric power is obtained from Canyon Ferry, 12 miles east on the Missouri river.

Local quarries supply the requisite amount of lime rock, which the contracts require to be crushed to 5 in.

Of the 450 men on the monthly pay roll, a census showed 200 Austrians, 110 Americans, and 30 Italians, the balance being made up of miscellaneous nationalities.

The ore supply is mainly derived from the Cœur d'Alêne mines in Idaho, and consists of crude ore, coarse and fine concentrates, middlings, and slimes. About one-half of the total output of the Cœur d'Alêne camp is sent to East Helena. An average analysis of these ores gives approximately 40 per cent. lead and 11 per cent. sulphur.

Local shipments, varying from month to month, are mostly highly siliceous gold and silver ores carrying little lead. Small shippers, leasers of old mines, and prospectors are very numerous throughout Montana, and it is the policy of the company to encourage these men by every possible means, including the free sampling and analysis of their ores and aid in the development of their mines and prospects whenever their results show promise.

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### *Sampling, Crushing, and Bedding*

In order to obtain a desirable roasted and sintered product for the blast furnaces, and as complete an elimination of sulphur as possible, practically all the ore is carefully bedded in bins of from 1,200 to 1,500 tons capacity. The accurate bedding of the ores is considered as essential to successful metallurgical practice and hence the utmost stress is laid upon this point, the importance of which is clearly demonstrated particularly in the winter months, when during days of 5° to 20° below zero all the ore freezes both in the cars and in the bins, the bedding is rendered inaccurate, and the result is evident in the subsequent erratic behavior of the furnaces. The slimes and fine concentrates are bedded direct and the crude ore and coarse concentrates are unloaded from the box cars into a pan conveyor, which discharges into the sampling and crushing mill. This mill has a capacity of 45 tons per hour, crushing to 0.41 in., and taking a mechanical sample of $\frac{1}{625}$ of the original volume by means of five Vezin samplers. The sample is then cut down by coning and quartering and treated in the usual way.

Rich gold and silver ores are put through a smaller sampling and crushing mill, a $\frac{1}{25}$ part being taken for the sample. These ores are binned separately and go direct to the blast furnaces on account of the high metal losses that would result in roasting. Single car lots are put through a third mill, which crushes to $\frac{3}{8}$ in., taking $\frac{1}{25}$ out for the sample. Twenty-five per cent. of the incoming ore is handled in the two small mills.

Montana mining laws require that $\frac{1}{10}$ of all ore shipped from Montana mines be held by the smelting companies for 30 days pending settlement.

The bins are divided into four divisions, three of the divisions containing the charges for the Dwight-Lloyd sintering machines, the Godfrey preroasting furnaces, and the Huntington-Heberlein converter pots, and the fourth being miscellaneous bins for flux, sintered product, and oxidized ores.

### *Roasting*

In 1907 a small straight-line Dwight-Lloyd machine, 30 by 150 in., was installed for experimental purposes. The results were so satisfactory that in 1908 four additional machines, 42 by 264 in., were ordered. These machines have given great satisfaction and have been a decided success in every way. Until 1912 crushed siliceous ore or barren lime rock was used as a grate dressing, mainly to prevent the charge from sticking to the grates. This practice was discontinued, however, as it was found that by using a straight-slot grate in place of the original herringbone pattern, by regulating the physical make up of the bin, and by keeping the grates

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open, the grate dressing could be dispensed with. The change from the herringbone to the straight-slot grate made possible the installation of a grate-cleaning device on each machine, consisting of a steel disk roller running parallel to the direction of the pallet. The charge mixture is trammed from the bins and conveyed to a Robins automatic tripper by means of a belt conveyor and bucket elevator. The tripper feeds the four hoppers above the machine uniformly. A thorough mixture of the charge is made in loading the bedded ore from the bins into the tram cars and also in distributing to the machine hoppers. The moisture, approximately 5 per cent., is regulated to suit the physical and chemical condition of the charge. Ignition is secured by means of a gasoline burner under 80 lb. pressure and the suction varies between 5 and 8 in. of water, depending on the tightness of the charge. The charge which can be put upon the Dwight-Lloyd machine is extremely flexible; varying percentages of matte, flue dust, slimes, and very fine concentrates, mixed with milled ore, crushed to 0.41 in., are daily sintered successfully. The sinter produced is hard and porous, with very little fines. Tonnages vary from 90 to 120 tons per machine day of 24 hr. The speed of the machine is from 16 to 20 in. per minute, and the sulphur is reduced from 12 per cent. to 3½ to 5 per cent. The following is a typical charge made up of approximately 75 per cent. milled ore, crushed to ¼-in. mesh, 20 per cent. slimes, and 5 per cent. flue dust. This charge ran at the rate of 110 tons to the machine producing an excellent product for the blast furnaces:

| | $SiO_2$ | Fe | S | Zn | CaO | Pb | Cu |
|---|---|---|---|---|---|---|---|
| Before roasting..... | 15.0 | 15.2 | 11.6 | 4.1 | 2.6 | 39.0 | 0.1 |
| After roasting..... | 16.8 | 17.2 | 4.5 | 4.0 | 2.9 | 41.7 | 0.4 |

In September, 1912, experiments were begun to determine the advisability of filtering the gases from the Dwight-Lloyd machine. A small experimental bag house was erected containing both cotton and woolen bags. At the end of six months, tensile-strength tests showed the bags to be uninjured by the gases. These tests were continued, with the result that in September, 1913, the gases were turned into the main bag house along with the blast-furnace gases. On account of the high lead content of the charge and the heat developed, sufficient lead oxide is formed, together with a small amount of zinc oxide, to neutralize the $SO_3$ in the gases.

The blast-furnace matte is preroasted in the four Godfrey furnaces, each having a capacity of 27 tons per day, converting the sulphur from 20 to 10 per cent. This preroasted matte is then mixed with the direct converting mixture containing not over 10 per cent. sulphur. The combined mixture is then blown at a pressure of from 8 to 12 oz. in the Huntington-Heberlein pots having a capacity of 12 tons each. At the end of 12 hr. a pot is raised by means of a 20-ton crane, the non-agglomer-

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ated fines are raked off, and the pot is carried to the breaking floor, where the button is dropped. The large lumps are picked up by the crane and dropped again, while the smaller lumps are fed into a Blake crusher set to 3 in. and hoisted into bins of 200 tons capacity, from which the product is taken to the furnaces. The sintered product is harder and less porous than the Dwight-Lloyd sinter, with perhaps 60 per cent. of it slagged. Taken as a whole the Dwight-Lloyd sinter is more satisfactory for the blast furnaces than the Huntington-Heberlein sinter. The standard tonnage for the department, consisting of the four Godfrey furnaces and 12 Huntington-Heberlein pots, is 300 tons per day. Following are analyses of the matte before roasting, the direct converting mixture, and the final Huntington-Heberlein sinter.

| | $SiO_2$ | Fe | CaO | S | Pb | Zn | Cu |
|---|---|---|---|---|---|---|---|
| Matte..... | 2.0 | 42.8 | 0.4 | 21.0 | 14.0 | 5.4 | 6.6 |
| D. C. mixture..... | 11.4 | 14.7 | 2.5 | 9.4 | 42.0 | 3.5 | Tr. |
| H.-H. product..... | 9.1 | 27.9 | 1.9 | 5.0 | 35.0 | 4.7 | 2.7 |

### *Smelting Operations*

There are four blast furnaces, 48 by 136 in. at the tuyères, with an effective height of charge column of 18 ft. Each furnace is provided with 16 tuyères 3½ in. in diameter supplied with blast at 33 oz. pressure. Three of the furnaces are generally in operation with the fourth held in reserve. At present, however, the four furnaces are running. This necessitated starting up three old hand reverberatories pending the installation of two new Dwight-Lloyd machines under construction. The standard tonnage for each furnace is 220 tons per furnace day, but this has been exceeded by 90 tons. The 4-ton charge car is raised to the feed floor by means of an incline, the hoistmen raising and dumping the car. The lead from the furnaces is tapped into 1-ton pots and sent to the dressing plant, and the slag and matte are run into a brick-lined floor hearth, from which the matte is tapped into Kilker mold cars, from which it is dumped into a railroad car and sent to the mill to be crushed and put into the Godfrey roasting bins. The slag overflows from the settler into a settling pot, where any matte that may have come over settles out, and from this pot it overflows again into a second slag pot. When the settling pot is full it contains a large percentage of matte and hence is kept separated from the slag shells that are returned from the slag dump. These shells are loaded from the dump into railroad cars by a locomotive crane and go direct to the furnaces. The matte from the settler pots is returned to the mill and crushed for the roasting bins. The following is an extract from a daily report:

Slag Analysis, Per Cent.

| Pb. | Ag. | SiO2 | FeO | MnO | CaO | MgO | Zn. |
|---|---|---|---|---|---|---|---|
| 0.90 | 0.30 | 33.2 | 31.9 | 2.6 | 18.4 | 2.1 | 5.0 |

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| Charge, Per Cent. | | | | | | |
|---|---|---|---|---|---|---|
| Pb. | S. | Coke | F.C. | H.-H | D.-L. | Slag Resmelted |
| 31.9 | 3.8 | 12.92 | 10.1 | 36.5 | 52.6 | 15.4 |

Matte fall, 14.1 Average blast, 33 oz. Average tonnage of three furnaces, 260

The matte on this day contained 18.7 oz. silver, 13.4 per cent. lead, and 9.7 per cent. copper. The specific gravity of the slag was determined to be 3.3 and that of the matte 4.2.

The report speaks for itself, but there are several points of special interest: (1) The high percentage of lead on charge, with the accompanying small amount in the slag; (2) the high speed combined with good reduction; and (3) the fact that there was 89 per cent. roast on the charge. Such a large percentage of roast is handled at few plants.

The Eilers-type slags are carefully adhered to, experience at this plant having taught that in so doing the best results on the furnaces are obtained.

The so-called lead smelters of Colorado and Utah run with from 8 to 15 per cent. lead on charge. With them the lead is simply a means of collecting the precious metals and saving the small amount of lead in their ores, whereas at East Helena, on the other hand, with from 30 to 40 per cent. lead on charge, this order is reversed on account of the high percentage of lead in the ores, and their low silver and gold content. As the percentage of lead on the charge increases, the problems of reduction and good blast-furnace work become greater and the furnaces become extremely sensitive to changes that would hardly affect them when running on a lower lead charge.

Whereas 5 to 6 per cent. Zn in the slag, produced from a low-lead charge, gives little trouble, its importance is greatly increased when we have from 30 to 40 per cent. lead on charge. Smelting a high-lead charge is a distinct problem, occupying a field of its own, whose difficulties offer no comparison to blast furnaces running on a low-lead charge.

Three brick bag houses, 43 by 133 ft. and 50 ft. high, each containing 1,260 cotton bags 18 in. in diameter and 30 ft. long, filter the blast-furnace and Dwight-Lloyd machine fumes. The bag houses are connected to the furnaces and Dwight-Lloyd machines by flues 1,570 ft. long. The suction in the flue is obtained by means of a fan making 136 rev. per minute and driven by a 100-h.p. motor. The temperature of the gases entering the houses varies from 36° in the winter to 92° in the summer. The average fan pressure on the bags is 0.40 in. water and the suction in the flue 0.70 in. The gases entering these bag houses contain a very small amount of $SO_3$ and in consequence the life of the bags is approximately four and a half years.

The collected dust is burned, for the main purpose of getting it into such a state that it can be easily handled on the roasting bins. Formerly, when high arsenical ores were treated at this plant, the raw dust contained 40

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to 50 per cent. arsenic, which on burning was reduced to 15 per cent. The following is an analysis of the bag-house dust before and after burning.

| | $SiO_2$ | Fe | Mn | Bi | Zn | CaO | MgO | S | As | Sb | Cu | Pb |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Raw dust..... | 2.6 | 0.7 | Tr | Tr | 2.4 | 0 | 0 | 10.2 | 1.2 | Tr | 0 | 68.0 |
| Burned dust.... | 3.2 | 1.3 | Tr | Tr | 1.8 | 0 | 0 | 7.8 | 1.6 | 0.3 | 0 | 62.8 |

The lead bullion is drawn from the furnaces into 1-ton pots. These are trammed into the drossing plant, adjacent to the furnace building, hoisted to the level of the drossing pots and dumped by means of a small auxiliary hoist. There are four large pots having a capacity of 54 tons each. While being filled the pot is heated, so that when full it is only necessary to allow it to stand a short time before removing the dross by skimming. This is done by hand, and generally requires only a short time. The lead is tapped from the bottom of the pot into bars weighing 90 lb. each. These are loaded by hand and shipped to the refineries at Omaha and Chicago, while the dross is put directly back into the blast furnaces. The monthly production of lead bullion is approximately 6,000 tons on a three-furnace basis.

The greatest stress is laid upon the safety and welfare of the men at the East Helena plant. Every possible precaution is taken to insure the men against injury and to promote their general welfare. A safety inspector is in charge of this work and every effort is being made to obtain the co-operation of the men and their foremen with the staff in charge to secure a high degree of efficiency in this department.

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# Tests of Rock Drills at North Star Mine, California

BY ROBERT H. BEDFORD AND WILLIAM HAGUE, GRASS VALLEY, CAL.
(Salt Lake Meeting, August, 1914)

THE objects of this paper are:

1. To give cost data on drilling obtained at the North Star mine in California;

2. To describe the methods of testing drills employed there, giving the results of shop tests, and the correlated data obtained underground;

3. To demonstrate that these tests form a satisfactory basis upon which to judge rock drills.

The rock at this mine is either close-grained diabase in the upper levels, or tough grano-diorite in the lower levels. The vein, which has an average dip of 23°, consists of about 5 ft. of "formation" lying between walls of unaltered country rock. Of the 5 ft., solid quartz and stringers make up 18 in. of pay ore. The formation, which consists of but slightly altered country rock, is almost as hard as the unaltered walls. From 4 to 4½ ft. is sent to the mill. One-third of the holes are drilled in the quartz; two-thirds in the formation. The holes are about 4½ ft. deep and break on the average of 1.15 tons per hole. The average number of holes per stope drill shift is at present 5.65. The number of drill shifts throughout the mine in 1913, was 18,679, and the cost of labor for drilling, power, supplies, upkeep of machines and air lines, tool sharpening, and distribution, amounted to 33 per cent. of total mining expense. To do this work there are in commission 43 No. 12A, three No. 17V and eight No. 16V Waugh stopers, three Jackhamers, two Waugh pluggers, and 17 No. 8 and two No. 7 water Leyners.

For the past three years a record has been kept of the repairs on each machine and number of shifts operated by it. The figures given in Tables I and II for the cost of repairs per drill shift have been taken from this record. The underground air pressure is about 90 lb. The air consumption per drill shift was measured by aerometers. The leakage of pipe lines between compressor and meter is not included. From the number of pieces of steel sharpened for each type of drill during the year the average used per drill shift has been ascertained. For other supplies the figures given represent a year's average.

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TABLE I.—*Cost per drill shift for Water Leyner No. 8*

| Item | Description of Unit | No. of Units Used per Drill Shift | Price per Unit | Cost for Item per Drill Shift |
|---|---|---|---|---|
| Labor of drilling | 8-hr. shift | 1 | $3.25 | $3.25 |
| Maintenance, | | | | |
| Labor | | | | 0.10 |
| Supplies | | | | 0.62 |
| Power | 1,000 cu. ft. of free air compressed to 100 lb. | 15 | 0.0275 | 0.41 |
| Supplies, | | | | |
| Lubricants | Quarts of "Red Engine" | 0.66 | 0.07 | 0.05 |
| Hose | Feet of 1 in. 5-ply wire-wound | 0.12 | 0.33 | 0.04 |
| | ½-in. 5-ply wire-wound | 0.12 | 0.19 | 0.025 |
| Drill steel | Pieces used | 16.75 | | |
| Labor sharpening and repairs | | | | 1.12 |
| Labor distribution | | | | 0.35 |
| Steel consumed | Pounds | 2.66 | 0.125 | 0.33 |
| Power for sharpener | 1,000 cu. ft. free air compressed to 100 lb. | 5.00 | 0.0275 | 0.15 |
| Oil for forge | Bbl. of 42 gal | 0.12 | 1.80 | 0.21 |
| Upkeep of air pipe | | | | 0.09 |
| Total | | | | $6.75 |

TABLE II.—*Cost per Drill Shift for Waugh 12A*

| Item | Description of Unit | No. of Units Used per Drill Shift | Price per Unit | Cost for Item per Drill Shift |
|---|---|---|---|---|
| Labor of drilling | 8-hr. shift | 1 | $3.00 | $3.00 |
| Maintenance, | | | | |
| Labor | | | | 0.10 |
| Supplies | | | | 0.37 |
| Power | 1,000 cu. ft. of free air compressed to 100 lb. | 17 | 0.0275 | 0.47 |
| Supplies, | | | | |
| Lubricants | Quarts "Red Engine" oil | 0.33 | 0.07 | 0.02 |
| Hose | Feet ¾-in. 5-ply wire-wound. | 0.12 | 0.27 | 0.03 |
| Drill steel | Pieces used | 10 | | |
| Labor sharpening and repairs | | | | 0.30 |
| Labor distribution | | | | 0.21 |
| Steel consumed | Pounds | 2.2 | 0.065 | 0.14 |
| Power for sharpener | 1,000 cu. ft. free air compressed to 100 lb. | 1.5 | 0.0275 | 0.04 |
| Oil for sharpener | Bbl. of 42 gal | 0.034 | 1.80 | 0.06 |
| Upkeep of air pipe | | | | 0.09 |
| Total | | | | $4.83 |

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The 16V and the 17V Waugh drills cost 10c. less for maintenance supplies, but require 15 pieces of steel per shift instead of 10, which leaves the cost about equal.

As already stated, the drilling at this mine is one-third of the cost of delivering ore at the mill. The importance of selecting a suitable drill for the work and keeping it in its highest state of efficiency is obvious. At first, time was wasted and trouble experienced in testing underground drills which were unsuitable. Drills which sounded all right when run in the repair shop failed to work satisfactorily underground.

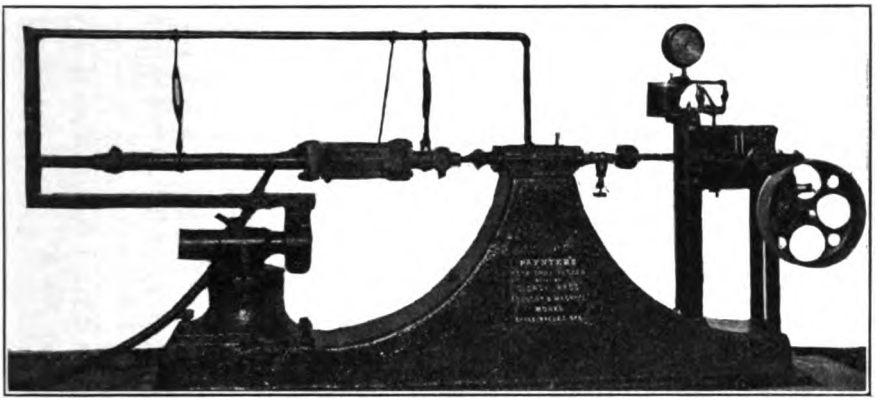

FIG. 1.—PAYNTER'S ROCK-DRILL TESTING MACHINE.

There was need for some reliable testing machine to overcome the difficulties. W. D. Paynter, who was responsible for keeping drills in repair, after experimenting for some time, perfected a testing machine, which he has patented. This machine has been described in detail elsewhere.[1] Briefly it consists of a device whereby the blow of the drill delivered against a plunger is measured by the distension of a diaphragm, oil being the medium of transmission. By means of a lever arm, the movement of the diaphragm is amplified. A pencil on the end of the lever, marking a piece of paper on a revolving drum, gives a graph of the work done by the drill over a given period of time (Fig. 1).

In Figs. 2 and 3 are given cards taken from different types of drills. *A* is the base line, line *B* represents the feed-barrel pressure in the case of an air-feed stoper. The length of the line *AC* is a measure of the energy transmitted through the drill steel to the plunger. The graph covers a period of 5 sec. The number of peaks, *C*, multiplied by 12, therefore, gives the blows per minute. To express the energy of the blow in foot-pounds, the tester was calibrated by allowing a sphere of known

[1] *Mining and Scientific Press*, vol. cvii, No. 5, p. 179 (Aug. 2, 1913); *Engineering and Mining Journal*, vol. xcvi, No. 18, p. 829 (Nov. 1, 1913).

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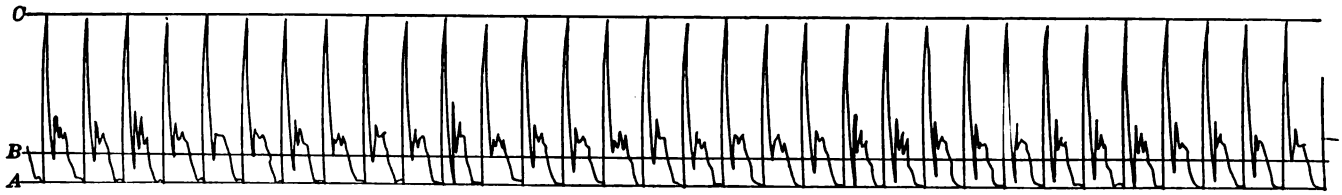
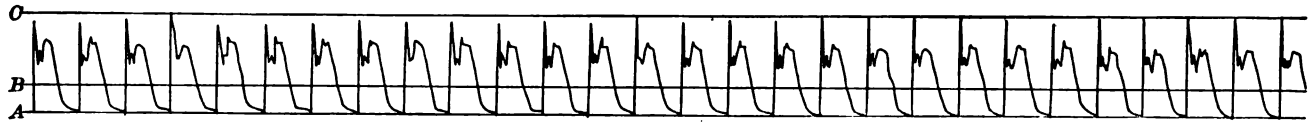
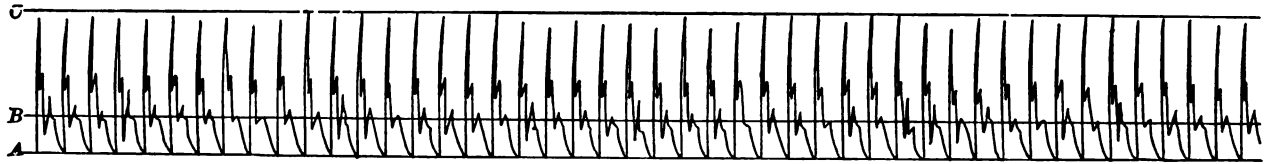



Air pressure, 95 lb.; blows per minute, 1,584; foot-pounds per blow, 55. Card No. 1.—In good condition.




Air pressure, 95 lb.; blows per minute, 1,368; foot-pounds per blow, 25. Card No. 2.—Not in good condition.

FIG. 2.—CARDS NOS. 1 AND 2.




Air pressure, 97 lb.; blows per minute, 2,352; foot-pounds per blow, 40.

FIG. 3.—CARD NO. 3.

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weight, suspended as a pendulum, to fall through measured distances against the plunger. The distensions of the diaphragm corresponding to these blows were noted. By forcing oil into the system with an attached pump, the diaphragm was distended to each of these points in turn, and the static pressure corresponding to each was read on the gauge. A curve was then constructed with foot-pounds as abscissæ and pounds static pressure as ordinates. The strength of any blow, recorded on

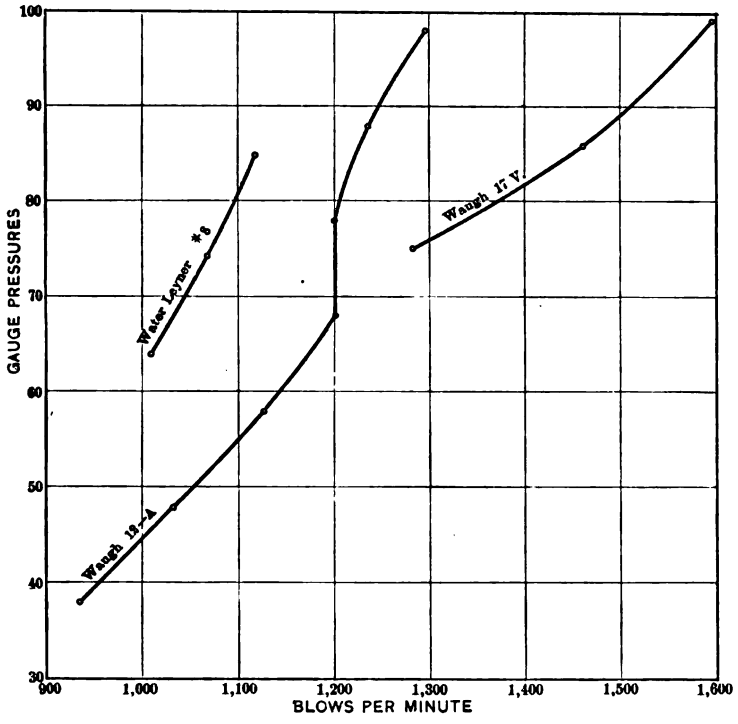


FIG. 4.—CURVE SHOWING INFLUENCE OF GAUGE PRESSURE UPON BLOWS PER MINUTE.

the graph as the line $AC$, may be measured by pumping oil into the system until the distension of the diaphragm moves the pencil out to point $C$. The pressure in the system at this point is read on the gauge. The foot-pounds corresponding to this can then be read directly from the curve. Since the areas of the plunger and of the diaphragm do not change, equal blows will produce the same pressure, even though permanent straining of the diaphragm may result in their producing unequal distensions. Errors which might be caused by these inequalities of distensions are avoided by thus expressing strength of blow in terms of equivalent static pressure instead of magnitude of distension.

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The general method of testing a drill consists in first obtaining graphs in the shop and then the drilling speed underground. Having passed the first test, the drill is used in the mine, a record being kept for several months of the footage drilled. Graphs taken from time to time show whether or not the drill is deteriorating, while the card record of repairs gives the cost of its upkeep. Drills sent to the shop for repairs are not returned to the mine until they show on the tester a satisfactory graph.

From the graphs taken by testing a drill at different gauge pressures, characteristic curves may be constructed, as shown in Figs. 4 and 5.

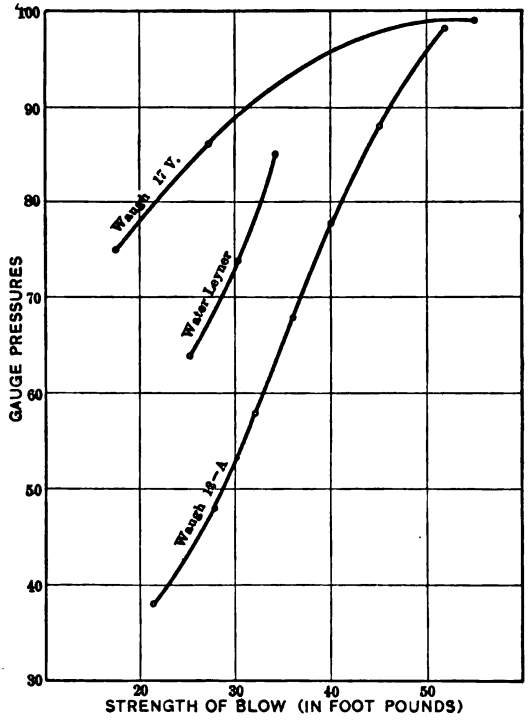


FIG. 5.—CURVE SHOWING INFLUENCE OF GAUGE PRESSURE UPON STRENGTH OF BLOW.

Below are given the results of tests made to determine, if possible, the relation existing between drilling speed, blows per minute, and foot-pounds per blow.

Table III compares (*a*) three drills striking a hard blow, (*b*) two drills striking a medium blow. These were tested in very hard ground with holes at an inclination of 25°, 1¼ in. cross steel with 2¼ and 2 in. bits being used; conditions, however, were kept the same for each test, which lasted 5 min.

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TABLE III

| | Foot-Pounds | Blows per Minute | Drilling Speed Feet per Minute |
|---|---|---|---|
| (*a*) | 53 | 936 | 0.126 |
| | 52 | 1,320 | 0.165 |
| | 55 | 1,584 | 0.250 |
| (*b*) | 40 | 1,260 | 0.108 |
| | 42 | 2,352 | 0.195 |

These results indicate that drilling speed varies approximately with the blows per minute, the strength of the blow remaining constant.

Table IV compares drilling speed with varying strengths of blows. The conditions of test were: Ground only medium hard; length of each test 5 min.; 1¼-in. cross steel with 2-in. bit; holes at an inclination of 45°; drill used, air-feed stoper.

TABLE IV

| | Blows per Minute | Foot-Pounds per Blow | Feet Drilled per Minute |
|---|---|---|---|
| (*c*) | 1,272 | 48½ | 0.378 |
| | 1,222 | 43 | 0.447 |
| | 1,200 | 38½ | 0.308 |
| | 1,170 | 34½ | 0.250 |
| | 1,090 | 30½ | 0.188 |

A test under the same conditions as in (*c*), except that the inclination of the holes was 20°, gave the results shown in Table V.

TABLE V

| | Blows per Minute | Foot-Pounds per Blow | Feet Drilled per Minute |
|---|---|---|---|
| (*d*) | 1,272 | 48½ | 0.251 |
| | 1,222 | 43 | 0.206 |
| | 1,200 | 38½ | 0.198 |
| | 1,170 | 34½ | 0.135 |

A test (*e*) was made under the following conditions: Ground hard; length of each test 5 min.; drill used, No. 8 water Leyner; 1⅛-in. hollow steel with 2¼-in. bit; holes nearly horizontal. In this test varying strengths of blow were obtained by means of stops of different lengths screwed into the ends of the valve chest.

TABLE VI

| | Blows per Minute | Foot Pounds per Blow | Feet Drilled per Minute |
|---|---|---|---|
| (*e*) (1) | 1,368 | 40 | 0.162 |
| | 1,272 | 45 | 0.234 |
| | 1,260 | 52 | 0.119 |
| | 1,212 | 65 | 0.129 |
| Repetition of test (1) for confirmation: | | | |
| (2) | 1,368 | 40 | 0.135 |
| | 1,272 | 45 | 0.224 |
| | 1,260 | 52 | 0.183 |
| | 1,212 | 65 | 0.195 |

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As already stated, Table III indicates that for blows of equal strength the drilling speed is approximately proportional to the number of blows, even when these differ as much as 40 per cent. In constructing the following curves from Tables IV, V and VI, where the maximum variation in the number of blows is 15 per cent., the drilling speed has been arbitrarily adjusted according to the results obtained from Table III, so that the effects of the strengths of blow may be comparable.

These curves all show the same tendency to flatten at above a certain strength of blow, which in this ground happens to be about 45 ft-lb. It will be noticed that the curves for the Leyner and for the stoper at 45° inclination are almost exactly similar. This is thought due to the fact

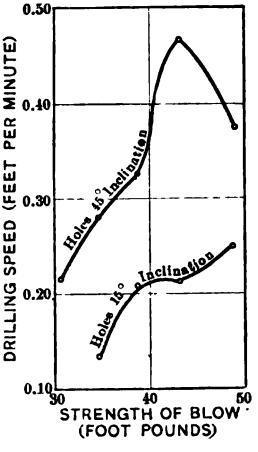


FIG. 6.—CURVES SHOWING INFLUENCE OF STRENGTH OF BLOW ON DRILLING SPEED (AIR-FEED STOPER).

that in each case the face of the hole is clear of cuttings. In the cases of the flat holes, which do not clear themselves, the drilling speed is not only lower, but does not show the same peak. This suggests that where holes can be cleared of cuttings a 45 ft-lb. blow is sufficient to obtain the fastest drilling, whereas in holes where the deadening effect of cuttings exists the limit of effective blow is higher. This also emphasizes the desirability of a stoper with water feed to the face of the hole.

From these tests, and others giving similar results, it has been decided that for conditions existing at this mine, stoping drills should strike a minimum of 40 ft-lb. As none of these drills has yet been found that strikes over 55, it remains solely a question of maintenance to keep them at this point. The Leyner, with blows as high as 65 ft-lb., presents a different problem. In this type the valve was so adjusted, by means of standard plugs screwed into the ends of the valve chest, as to give at average gauge pressure blows in the neighborhood of 45 ft-lb. It is

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hoped that this reduction in strength of blow will result in lessened breakage of steel, decreased repair costs, and maximum drilling speed. The adjustment has not been in use long enough to give any figures on the first two points.

The effect of setting the minimum strength of blow in the stoping drills at 40 ft-lb. has been to increase the breakage of steel by 1 lb. per

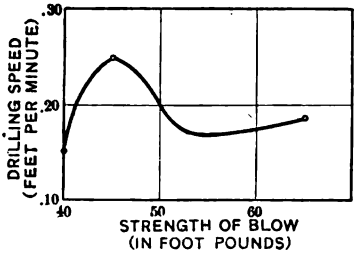


FIG. 7.—CURVE SHOWING INFLUENCE OF STRENGTH OF BLOW UPON DRILLING SPEED OF WATER LEYNER NO. 8.

drill shift, and the cost of repairs by 27c. per drill shift. The footage drilled, however, has been increased 15 per cent., reducing the cost per foot of hole drilled from 20.3 to 18½c. The output per drill shift has increased 10 per cent. during the same period, but this figure is complicated by the width of stope broken, and is therefore not quite comparable.

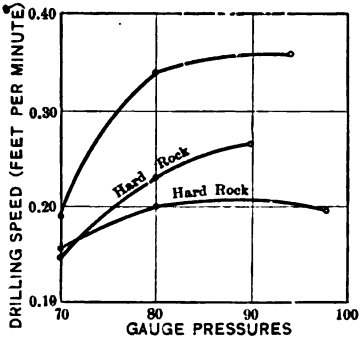


FIG. 8.—CURVE SHOWING INFLUENCE OF GAUGE PRESSURE ON DRILLING SPEED.

In the case of the Leyner drill, the cost for sharpening, breakage, and repairs makes a high cost per drill shift. Comparison with a type of machine in which these items are low is valuable, as it proves that the expense is justifiable. A 5 by 7 ft. drift in this mine requires 11 holes, aggregating 43 ft., to break a round. A 2½-in. reciprocating drill, striking 40 to 50 ft-lb. blows, 600 to 800 times a minute, used to require two drill shifts for drilling the round, at a cost of $4.45 per drill shift, or 20½c.

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per foot of hole. The Leyner does this in one drill shift for 15½c. per foot of hole.

The use of the tester in the shop has greatly facilitated repairs, as any abnormal action in the drill is disclosed at once. Experiments have demonstrated several points. Among these may be mentioned the effect of feed-barrel pressure. If the feed-barrel packing is slightly defective, the drill does not hold against the ground hard enough, causing as much as 30 per cent. decrease in drilling speed. In the 16V stopers, wear in the barrel bushing after about 250 drill shifts causes cushioning due to leakage of air, with reduction in the strength of blow, amounting to as much as 20 per cent. In the 17V type, at pressures below 85 lb. per square inch, the strength of blow drops very rapidly. Plugging some of the exhaust ports causes the strength of blow to decrease less rapidly. This makes the drill more effective at low pressures.

Examination of Table IV shows the decrease in drilling speed that may be expected with a stoper striking 30 ft-lb., as compared with 45 ft-lb. On this point it may be stated that many confirmatory tests have been made. Unless the drills are maintained at their best, the strength of blow drops rapidly (sometimes after 100 drill shifts), with resulting decrease in drilling speed. The cost per drill shift, however, remains practically the same. It is obvious what the result will be. Weston has stated in his book, *Rock Drills*, "If the whole system from compressor to drill is not maintained at the highest point of efficiency, the neglect will surely reveal itself in high mining cost."

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Treatment of Copper Ore by Leaching Methods

BY W. L. AUSTIN, RIVERSIDE, CAL.

(Salt Lake Meeting, August, 1914)

THE advance made in recent times in this branch of metallurgy is indicated by the attention the subject is receiving from important American copper-producing companies. Reference to the files of publications devoted to the mining industry discloses that some 20 American companies are actively investigating the amenability of their ore, or other material, to leaching methods, and that plants of varying capacities up to 10,000 tons per day are under construction or are projected. Several leaching works are in commercial operation.

It has been frequently pointed out that no method of leaching is universally applicable, for the reason that each ore differs in some particular from apparently similar ore found elsewhere, and also because local conditions are rarely the same even where the ore closely approximates in character that being worked at some other point. For this reason the older companies are approaching the matter in a conservative manner, experimentally determining for themselves the salient features in each instance.

In considering a leaching proposition three factors at once fix the attention: (1) disengaging the metal from mineralized forms in which it is found in nature; (2) recovering the metal in a commercial state after it has been dissolved; (3) the apparatus best adapted to carrying out the several operations.

## *Bringing the Metal into Solution*

This feature has been found to present no serious difficulties. Numerous solvents for copper are known, and it has been repeatedly demonstrated on a commercial scale that from 80 to 90 per cent. of the metal contained in an ore can be rapidly brought into solution. It is interesting to note, however, that all of the companies above referred to have selected sulphuric acid as the basis for a lixiviant. Where oxidized ore practically free from precious metals is treated simple leaching with

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sulphuric acid has been adopted in every case reported, less than an hour sufficing for the operation under favorable conditions.

When the copper to be extracted is found mineralized as a sulphide the use of sulphuric acid as lixiviant necessitates breaking up the sulphur combinations by roasting in order that the metal can be brought into soluble form. This operation, however, instead of being a drawback, presents advantages. The expense of roasting is light, for it is very effectively and cheaply carried out in modern mechanical furnaces, and the lixiviant itself is thereby obtained with which the copper is subsequently removed from associated gangue. In some cases it has been found that the addition of a little salt at certain stages in the roasting assists in bringing copper into solution and facilitates the extraction of any silver present.

The degree of comminution necessary in order to obtain a high extraction depends upon the mineralogical and physical characteristics of the ore. In the case of a sulphide requiring a preliminary roast, reduction to 16 mesh has usually been found sufficient. Where an oxidized ore is concerned, as the copper minerals for the most part lie along fracture planes in the matrix, comminution to ¼ in., or even ½ in., often exposes the cupriferous portion sufficiently to permit satisfactory extraction.

If time is an element of importance, as in the case of small mines where it is not desirable to have considerable capital tied up in ore undergoing treatment, or where a high percentage of extraction is essential, fine grinding accompanied by agitation may sometimes be advantageous; but the benefits accruing from fine comminution disappear where large deposits of low-grade ore are handled. Naturally, the finer the grain, the more readily the acid solution attacks copper minerals, and the less time is required to bring the metal into solution. Then the question arises whether or not it is more economical to crush to ¼ in. and leach in large vats by percolation, or whether the ore should be reduced to 16 mesh, or finer, and subjected to agitation.

For instance, a certain ore when crushed to ¼ in. was found to yield 70 per cent. of its contained copper in three days by the use of weak acid solution. The same ore crushed to 16 mesh permitted 85 per cent. extraction in 4 hr., using a much stronger acid lixiviant. With a 2 per cent. ore this means a saving of 6 lb. copper per ton of ore treated, together with economy in time. Offsetting these advantages are: greater first cost of crushing and agitating machinery; higher maintenance charges on plant; and difficulty in washing the finer tailings. Almost any ore may be treated by percolation when properly prepared. It has been frequently noted by metallurgical writers that solutions will percolate more freely through a roasted ore than through one treated in its natural state.

Agitation of course brings copper more quickly into solution than percolation, and one advantage of this rapid action is that sulphuric acid

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solutions manifest a selective action, attacking oxidized copper minerals before the iron and alumina content of an ore. Hence, in leaching by agitation the resultant lixivium is apt to be less contaminated by these elements than where percolation is applied. This is of importance when copper is to be subsequently removed from leach liquors by electrolysis, using cells based upon copper-refinery practice. It is not so important when depolarizers and moving electrodes are employed.

Many oxidized copper ore bodies occur in what are known as contact-metamorphic deposits, the gangue of which is largely composed of garnet and associated minerals, wollastonite, vesuvianite, epidote, etc. Analyses of such ore disclose large amounts of calcium oxide, which may lead to erroneous assumptions and possible rejection of leaching processes as unsuitable in such cases. The facts are, however, that these calcareous minerals are not attacked by weak acid lixiviants in the length of time necessary to extract the associated copper. It is not always safe to base a verdict upon the evidence of analyses alone when considering the appropriateness of this or that method of reduction.

When leaching with sulphuric acid lixiviants, and if the ore contains no sulphides or sulphates, the acid cost may constitute a preponderating proportion of the expense, and a source of cheap acid becomes a vital factor. On low-grade ore treated in the raw state the consumption of acid varies between 2 and 4 lb. per pound of copper produced, and when this reagent costs $0.014 per pound the outlay for this item alone may equal one-third of the total cost of the operation. This expense may be greatly reduced, or even wholly removed, when sulphides are available, for by roasting the latter and passing the gases containing sulphur dioxide into electrolyzing cells the amount of current required to deposit copper is reduced and an excess of acid accumulates for use in leaching new batches of ore. With the exercise of some ingenuity sulphur dioxide may be applied in this manner without delaying the operation and without causing annoyance to those working around electrolytic cells.

In leaching roasted mill tailings the consumption of acid appears to be small—about 50 lb. acid per ton of tailings handled—even with very fine material. This illustrates the benefit derived from roasting substances containing colloids, such as mill slimes, when the copper is to be subsequently extracted by lixiviation. Roasted porphyritic ore also requires very little acid, partly because aluminous colloid compounds are dehydrated in the process, and partly because such ore contains few ingredients soluble in the strength of solutions employed.

Referring again to the fouling of liquors made use of in cyclic leaching, experimental data recently published, as well as unpublished, indicate that the matter is not as serious as was formerly thought. In one instance where considerable quantities of ore containing large amounts of lime and soluble iron were treated experimentally with copper sulphate solutions,

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ferrous sulphate was present in such quantities that it crystallized in the pumps and pipe lines, at times completely stopping the flow, and yet the results were considered satisfactory; that is, in a series of six tests 1,082 lb. of copper are said to have been deposited, under conditions mentioned, with an expenditure of 1.5 kw-hr. per pound of copper. All of these tests except one were made without use of a diaphragm. The experiments were carried out at the Greenawalt ore-testing plant in Denver.

The height of ore column permissible in percolation has been the subject of much investigation. Garnetiferous ore crushed to ¼ in. percolates freely, even when the coarser material is mixed with much fine. An ore which contains such a proportion of fines that solutions will not pass through it when crushed to ¼ in. and thrown into a vat, can be rendered permeable by dampening and thoroughly mixing before charging. By this operation the finer particles are made to adhere to the coarser, and do not collect in impervious layers, which is apt to occur when the material is charged in a dry condition.

It has been found when leaching an ore carrying large amounts of lime and iron that when the lixiviant becomes weak in acid, basic iron salts and gypsum separate and clog the interstices between the ore grains, causing the flow to cease. This does not take place when sufficient acid is present to retain iron salts in solution. With proper precautions an ore column of average oxidized copper ore 10 ft. or more in height can be percolated without difficulty; in one recently described ore-leaching plant the percolation vats are being built 16 ft. high.

Heating the lixiviant, of course, always renders it more active, but this is not always desirable in copper-ore leaching using sulphuric acid, because under such conditions combinations of elements other than those of copper are attacked and their bases are brought into solution. It is seldom found necessary to heat the liquors in leaching proper, the strength of acid lixiviant used being sufficient to effect solution of the copper at ordinary temperatures.

Most ores amenable to the leaching process carry such small quantities of the precious metals that they can be neglected. Where gold and silver are present in amounts sufficient to warrant their extraction, they are either converted into chlorides in the roasting furnace and afterward dissolved in brine, or other solvents, from which solutions they may be removed by any one of several well-known methods, or else they may be recovered from the residues in separate operations after the copper has been taken out by acid lixiviation.

### *Recovering the Metal from Leach Liquors*

There have been many processes brought out for removing copper from solutions containing that metal, and some of these may find application in

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special cases, but there are only two which have met with extended commercial use—precipitation by means of metallic iron, and deposition by the electric current. The first mentioned still has its advocates,. but electrolysis is meeting with more and more favor as that method becomes better understood by men in the field, although the apparatus now in use is still poorly adapted to the purpose.

At the present time the practice so long in use at electrolytic copper refineries is being closely followed, but it is doubtful whether these methods of procedure are best suited to the requirements of ore leaching. The necessity for improvement becomes obvious when the conditions in an electrolytic cell employed in removing copper from an ore lixivium are closely studied. In copper refining the bath is kept fairly constant with regard to copper content and free acid present, fresh metal being taken up from the anodes as fast as that in the electrolyte is deposited upon the cathodes. In ore leaching, on the other hand, copper is being constantly removed from the electrolyte, and the latter has to be returned to the ore for a fresh supply. It is manifest, therefore, that conditions differ essentially in the two operations and the premises seem to call for modifications of the apparatus used.

In treating an ore lixivium, at the electrode where current enters the bath an acid radical ($SO_4$) is being constantly disengaged which decomposes water and combines with the hydrogen set free to produce $H_2SO_4$. This reaction is made apparent by the formation of oxygen bubbles on the anode; if these are not in evidence it is because the gas is entering into combination with some salt or element present in the electrolyte, a feature usually, though not always, detrimental to the process. Nascent oxygen produced in this way may attack the material of which the anodes are composed, wasting them, or it may oxidize ferrous salts in the electrolyte to ferric, thereby bringing into action a solvent which corrodes the copper being deposited on the cathodes. Ferric salts are then being formed at the anodes and reduced to ferrous at the cathodes, wastefully consuming electric current. Or anions and cathions liberated at their respective poles may set up counter currents working in opposition to the main current. To overcome complications introduced by formation of ferric salts attempts have been made to keep the anolyte separated from the catholyte by means of diaphragms, but this recourse has not met with much success in commercial operations.

It is therefore evident that in electrolysis of copper lixiviums there is urgent need of a cell in which oxygen forming at the anodes may either be removed as fast as produced, or rendered innocuous by chemical combination, operations to which the type of electrolytic bath found to meet the requirements of copper refiners does not readily lend itself. Ingenious forms of apparatus have been devised by means of which the liberated gases are shaken from the electrodes as fast as set free, and corrosion of

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lead anodes is said to be thereby reduced to about 10c. per ton of copper deposited. This is an advance in the right direction, but in such cells impoverishment of the electrolyte at the cathode still assumes adverse proportions, and solutions have to be returned to the leaching department with copper content only slightly reduced.

In electrolysis, economy is effected by facilitating the supply of cathions to the negative electrode so as to avoid as far as possible waste of current in doing work other than that desired. If left to the magnetic attraction of the electrodes alone the movement is too slow to prevent impoverishment and its attendant evils, at least in commercial electrolysis of ore lixiviums.

Circulation of the electrolyte is the means usually taken for supplying necessary copper ions to the cathodes; the more rapid the movement, the more efficient the supply. Also, adhesion of oxygen bubbles to the anodes is lessened by a current of electrolyte impinging against them. Circulation alone, however, does not go far enough, as is apparent from the fact that only comparatively rich copper lixiviums can be treated in the common type of electric refining cells, and that the liquors must be returned to the ore-leaching department in some instances when only about one-fifth of the copper content has been removed. This method of operation calls for continuous pumping of liquors, which serves no useful purpose other than taking up a small amount of copper in one department and depositing it in another, both operations being incomplete. More of the copper than was just stated may under favorable circumstances be removed from a rich lixivium in a refiner's cell, but only by applying a low current density, which means a large plant for a given production. For example, an electrolyte may be reduced from 5 to 1½ per cent. or less, but the first cost of such a plant will be large. The desideratum is a cell which will rapidly deplete a 1½ per cent. electrolyte to about 0.3 to 0.4 per cent. with the same energy efficiency as in the former case, turning out at the same time a high-grade metal.

Experimental data on a fairly large scale have been obtained through application of the two factors—moving electrodes and depolarizing agents—which give a hopeful aspect to the solution of the problem involved. It is practical to get rapid deposition of metal of excellent quality from impure lixivium, but at the expense of power. However, considered from the commercial standpoint, results ranging from 1.2 to 2.3 kw-hr. per pound of copper deposited are encouraging. Energy efficiency must, of course, always be taken into account in addition to current efficiency when the respective merits of different cell types are considered, but first cost and speed are also commercial factors; the greater the current density, the less the size of electrolytic plant.

Deposition of metal on cathodes is affected by current density perhaps as much as by any other factor. This is the same as saying that

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deposition is in a large measure dependent upon the amount of copper carried by the electrolyte, or which can be brought into contact with the cathodes in a given unit of time, for current density must be progressively reduced as the electrolyte is depleted of its copper if pulverulent precipitation is to be avoided. A firm deposit of copper may be obtained at current densities running into thousands of amperes per square foot, provided impoverishment of the liquors immediately adjacent to the cathodes is prevented. Arborescent accretions also are markedly absent when circulation is properly adjusted to current density.

There are other well-known causes which bring about disturbances during electrolysis, but if copper ions are supplied to the cathode as fast as needed, thereby focusing the current on the work desired of it, irregularities disappear. Whatever the origin of pulverulent deposition may be thought to be, the fact remains, that a firm bright deposit of copper can be obtained from an impure (1.5 per cent.) electrolyte when copper ions are properly supplied to the cathode, and provision is made for counteracting the injurious effects produced by liberated gases, whereas the same liquor in the electrolytic refining type of cell gives only a loose, dark-colored spongy mass which bridges the electrodes and causes short circuiting.

Heating the electrolyte is a questionable procedure, because it renders the objectionable components of the liquor more active in corroding the deposited copper. It has long been known that a greater quantity of copper can be deposited by the same amount of current from a neutral copper sulphate solution than from one of the same metallic content but carrying in addition free acid. The cause of this is that the acid redissolves the deposited copper. If ferric salts are present corrosion is still more marked. These reactions are furthered by elevating the temperature of the bath. Heating an electrolyte always facilitates passage of the current; but ore lixivium contains a variety of salts, and as all ions may serve the current as a means of transport, a large amount of electricity may find passage without depositing copper. A good conducting electrolyte is not necessarily economical in deposition of metal.

### *Apparatus*

Concrete as a material for construction of vats, launders, etc., in leaching works has found application a number of times, but nothing is on record as to its durability under working conditions. As the cement binder is alkaline in its nature, and as leaching solutions are of varying acidity, it is to be expected that where the two come into contact chemical reactions must necessarily follow. In a measure the strength of the acid liquors will have an influence, as will also the salts contained in the solutions, but where much free acid is present ordinary concrete will crumble.



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Concrete has manifest advantages over wooden construction, especially in hot, dry countries, and, of course, concrete vats are less apt to deteriorate than are wooden ones in case of temporary suspension of activities. To make the good qualities of concrete available the problem is to devise means for preventing penetration of acid liquors into the body of the concrete, which sooner or later must result in its destruction.

There are several ways in which this object may be accomplished on a small scale, but their adaptability to commercial purposes has yet to be demonstrated. The inner layer of the concrete may be made of some acid-resisting composition, or a concrete vat may be lined with brick which are not affected by the solutions, or after the vat is completed a non-corroding coating may be applied which sinks into the finished concrete and protects the binder. Heavy mineral oils, and paraffin applied hot, have been used in this manner. Coatings of pitch or asphaltum do not seem to answer the purpose well, for sulphuric acid solutions penetrate them through minute holes, forming gypsum, with consequent swelling, and the asphalt coating peels off as the concrete disintegrates.

Recently it has been stated that reinforced-concrete vats may be lined with mastic asphalt and satisfactory results obtained. The asphalt mixture had to be specially prepared and the work was done by workmen skilled in handling such material. It remains to be seen how this construction will endure under working conditions on an extended scale. It would seem to be subject to the defect common to all linings designed to protect corrodible material from the action of strong acids: if a leak starts anywhere the alkaline concrete binder will be attacked as a matter of course and this will only become known when the structure collapses.

Mixing heavy mineral oils with the concrete as it is made, affords a hard liquid-repelling mass which behaves well with weak acid solutions. The oil is incorporated with the cement-sand mortar before the broken rock is added, as in the L. W. Page water-proofing process. This method renders the whole mass to some extent acid-resistant in that the oil repels liquors seeking to permeate the structure. It has the advantage over the lining system that leaks are self-stopping and the lining is not pushed off leaving the concrete backing unprotected. Still, even oil concrete slowly disintegrates when strong (10 per cent.) acid liquors are permitted to act on it for weeks at a time.

Certain kinds of wood—notably the Western fir—have been found to resist acid solutions to a remarkable degree, even without lining. Storage vats holding cupriferous solutions containing 10 per cent. or more free sulphuric acid showed no signs of failure after being in use for several months.

With the help of reinforced concrete, leaching vats of large dimensions may be constructed. In a plant now being built the vats are designed to hold each 10,000 tons of ore. It is expected that two days' percolation

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will suffice to render soluble 90 per cent. of the copper contained in this ore.

*Costs*

Extraction of copper from suitable ore by leaching methods is generally assumed to cost less than the combined expense of concentration, smelting, and refining, and experimental data tend to confirm this opinion. At the Butte-Duluth plant, which is handling in excess of 100 tons of oxidized ore daily, the copper is said to cost $0.085 per pound. This figure probably refers simply to working expenses and does not include freight East, marketing, etc. After the contemplated increase in the capacity of the plant it may be expected that cost of the metal will be materially reduced.

Experimental work carried out at Morenci, Ariz., indicated the probable cost of leaching tailings slime at $0.0743 per pound copper delivered in New York. This includes expenses after the copper leaves the mine, with due allowance for interest and depreciation.

From extended tests made at the Anaconda works, carried out in great detail upon a working scale, the conclusion was reached that copper could be produced from concentrating-mill tailings for about $0.07 per pound copper.

At the Steptoe mill, in Nevada, a small leaching annex is treating oxidized copper ore in conjunction with flue dust at a cost of $0.08 per pound copper recovered.

The actual cost of copper produced by leaching methods is, however, not of so much importance as the fact that ore may be handled in this manner which is not amenable to reduction by any other process. A siliceous oxidized copper ore carrying from 1.5 to 2.0 per cent. metal may be profitably leached under favorable conditions; there are no other metallurgical means known for commercially handling ore of that character and grade.

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# Mining Methods at the Copper Queen Mines

BY JOSEPH P. HODGSON, BISBEE, ARIZ.

(Salt Lake Meeting, August, 1914)

IN 1880, mining operations were commenced at the Copper Queen mine. The famous Queen orebody, which extended to the surface, was first quarried from a large open cut in the outcrop. The orebody was followed down to the 300 level with the Queen incline, and stoped. This, we believe, marks the introduction of square setting in the Bisbee district, which is still the method most generally used.

## *Orebodies*

The orebodies in the Copper Queen mine occur in the limestones, and the main portion of the ore has been mined from the Abrigo, Martin, and Escabrosa limestones. While the orebodies outcrop in the extreme western end of the mine, the general dip is to the east and south, at an angle of about 20°, although this dip is not by any means regular, being, in fact, very irregular locally. The ore is remarkable for its variableness in character, some of it being very soft, requiring a large amount of timber, and other portions consisting of extremely hard sulphides. The orebodies, in general, while remarkable for their continuity, are very irregular, both as to shape and size. The mine having been in operation so many years, and mining having been prosecuted over such a large area, the operations have caused the shrinkage of the overburden, and there is a constant movement, which is shown by large cracks and interstices in the surface rocks, which largely contribute to the very high deadwork or maintenance cost of the mine. It is surprising that for year 1913 it cost almost $400,000 just for the one item of repairs and deadwork in this property.

## *Haulage and Hoisting*

The underground openings have been extended in the neighborhood of 200 miles. The ore hoisting is performed at a centrally located shaft, the Sacramento, to which all the ores from different parts of the mine are conveyed by electric haulage, of which there was slightly over 9 miles in operation in 1913. The hoisting levels at the Sacramento shaft are placed 200 ft. apart, commencing at the 400 level and continuing down to the 1,600 level, the average hoisting distance being 1,000 ft. The

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method of hoisting is by Kimberly skips, which are loaded from pockets, and as high as 400 skips have been hoisted through this shaft in a 7½-hr. shift.

Intermediate tramming to haulage chutes is done, in general, by mules and hand. Waste material is used for filling the square-set and cut-and-fill stopes, and when there is a surplus, it is sent to the surface at the subsidiary shafts, of which there are seven in operation, and which are also used for hoisting and lowering men, timber and supplies.

### *Lighting*

All main haulageways, powder houses, etc., are lighted by electricity. The lights used by workmen have been candles, but carbide lamps are being substituted, as it is thought that by using them there is less liability of fire, and it has also been demonstrated that carbide is more economical. It is a better light to work by, and, as much sorting of ore is done, it enables closer sorting.

### *Compressed Air*

All hoists, with the exception of the Sacramento, are operated by compressed air, generated at the central power plant, by compressors having a total capacity of 21,000 cu. ft. of free air per minute. Electric power for underground and surface lights and haulage is supplied by three Curtis turbo-generators, which are connected with seven 407-h.p. water-tube boilers.

### *Square Setting*

Up to about 1½ years ago, square setting was used exclusively in this property. The system, as a whole, has been very successful. It is quite elastic, and permits the following of stringers from any point in the stope, and also permits of efficient prospecting. Perhaps the greatest objection to it in very soft ground, is the liability to loss of the stopes by excessive weight. There is also a high timber cost, and, as a whole, perhaps it is not as economical as other systems, although it must be said that a large proportion of the mine will always be worked upon this plan, by reason of the unequal and changing character of the ground.

The general custom in the square-setting system practiced here is to block the ore out in sections, numbering the sections consecutively, the object being to mine, if possible, four sections around one central raise. This can often be done, but frequently the ground is of such heavy character, and so much weight is thrown upon the timbers, that it is impossible to take out more than two or three sections to any one raise. These sections are laid out according to the local character of the ground, and are from two to four sets in width, and from six to ten in length.

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Great care must necessarily be taken in laying out the work to avoid making sections too large and of too great width, so as to risk the possibility of caving.

As the stoping progresses from the sill upward, the raise is usually

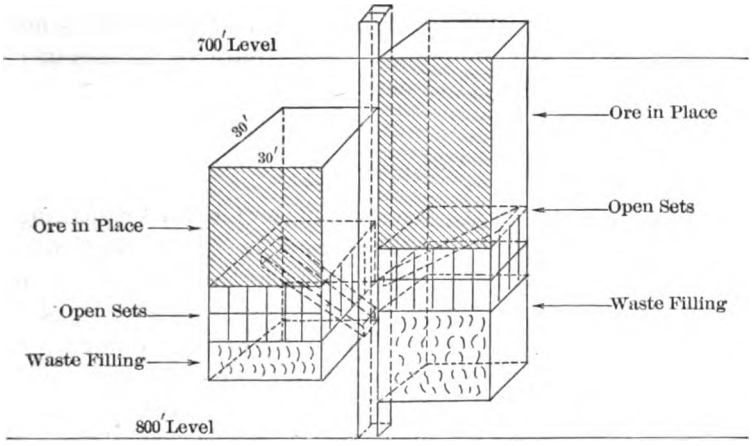


FIG. 1.—SQUARE SETTING.

sent through to the next level, to admit of the lowering of timber, the proper ventilation of the stope, and the dumping of filling; as it has been found necessary to carry the filling within about two floors of the back of

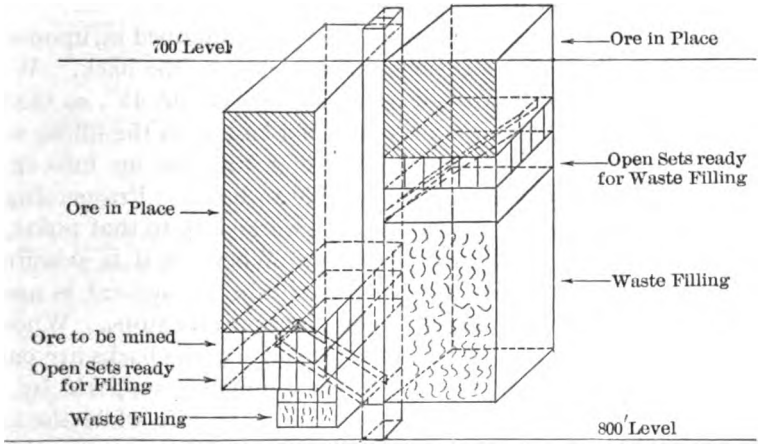


FIG. 2.—SQUARE SETTING.

the stope and immediately below where men are working. In many of the stopes, particularly in the oxide ores, a good deal of hand sorting is necessary, as it has been demonstrated that it is cheaper to eliminate the waste in the stope than to pay for smelting it. While this in many

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cases increases the mining cost materially, yet the company believes it is good business to leave the waste in the gob rather than put it into chutes, tram, hoist, and pay transportation and smelting charges upon it. In order that nothing is mined but what shows a margin of profit, a system of minima is in effect, based upon the selling price of copper.

Figs. 1 and 2 illustrate different stages of extracting the ore by means of sections under the square-set system. The chutes are so arranged as to require the least amount of mucking.

### *Cut and Fill*

Within the last two years, the management has been making some experiments in other mining methods, and, in certain portions of the mine, notably in the Holbrook, Spray, and Gardner divisions, some cut-and-fill stopes have been opened. This system of mining is, of course, applicable only in hard ground, and these stopes are exclusively in sulphide ores. The experiments have to date proved successful, having materially decreased the mining cost, as compared to the square-set method, and it is believed that this system should be worked wherever the conditions are suitable. The method in vogue is somewhat as follows:

The orebody is prospected as far as possible in advance, and the side and vertical dimensions of the ore are determined. Drifts are driven where possible under the bottom of the ore and raises put through the ore to the level above to permit the dumping of filling. Chambers are then cut out, drifts formed either by cribs or sets of timber, the back blasted down, raises cribbed up at convenient points, and filling dumped in, upon which the men may work and at all times be kept close to the back. Wherever possible, the stope is worked on an angle of about 45°, so that the broken ore may slide down upon a plank bed laid upon the filling to the chutes. This materially reduces the cost of getting the ore into chutes, and is very desirable wherever it is possible to use it. Prospecting can be done from any elevation, as the stope is worked up to that point, and the filling easily and cheaply disposed of. Wherever it is possible to work them upon an angle of 45°, very little timber, in general, is needed, as the slope of the ground materially helps to support the stope. Wherever a stope cannot be worked upon the slope and where the backs are carried horizontal, it is often necessary to put up temporary supports by cribbing and blocking up the back to make the stope safe while the ore is being extracted.

This system is also elastic, inasmuch as small blocks can be worked out wherever it is deemed necessary, provided the back is heavy and will not admit of being opened up in a fairly large chamber. One absolute necessity upon working this system is that the men be watched closely and taught to bar down the backs, taking care before they commence

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drilling operations that all loose or unsafe ground is taken down. Perhaps a disadvantage is where ore is intersected by stringers or bunches of waste,

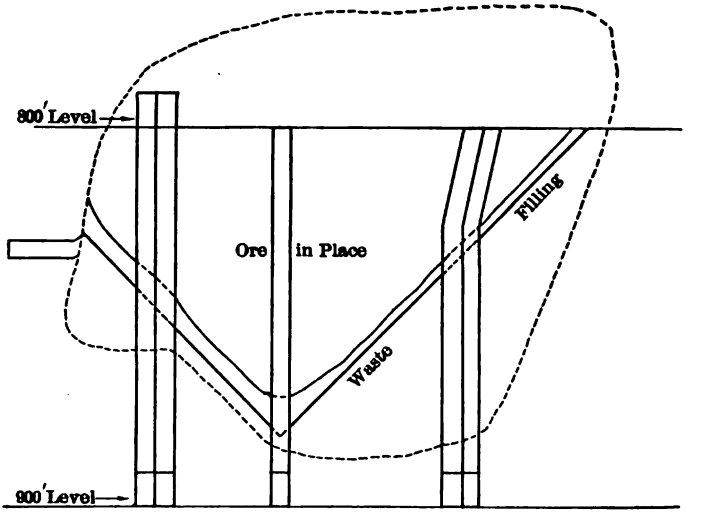


FIG. 3.—CUT AND FILL.

but, if care is taken, this waste can always be blasted down or put in the gob and the ore mined clean. The cut-and-fill method, up to the present time, has worked fairly successfully in the Copper Queen mines, and the

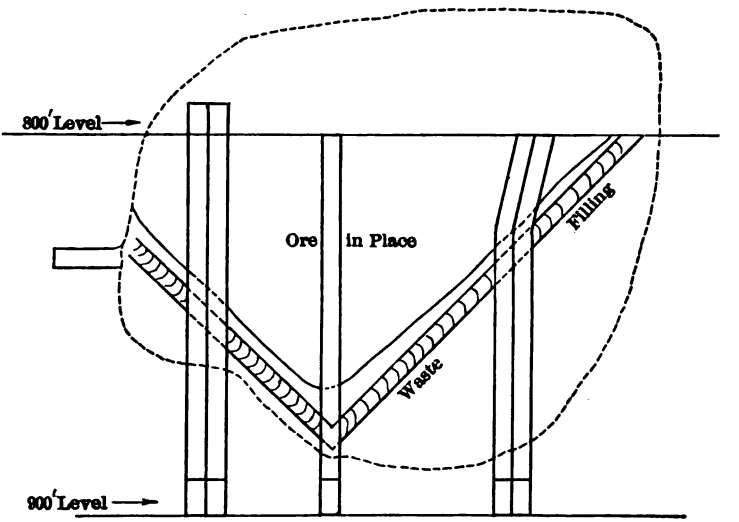


FIG. 4.—CUT AND FILL.

writer knows of many mines in Michigan and other places that have been worked successfully by similar means.

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Referring to Fig. 3, starting at the top of the slope, water holes are drilled and a section of ground about 8 ft. thick and 20 ft. wide is blasted down. The work is done underhand wherever it is possible in order to keep the back solid.

Fig. 4 shows that the ore has been removed, waste filling has been run in and is floored over, consequently, the stope is again in the condition as shown in Fig. 3.

### *Shrinkage*

Up to this time, only one place in the Copper Queen mine has been found where, in our judgment, a shrinkage stope could be developed. This stope is on the 1,100 level of the Lowell mine, and stoping is in progress there upon an orebody approximately 100 ft. in length and about 50 ft. in width.

In order to use the shrinkage system successfully, the surrounding character of the walls must first be ascertained, and it must be demonstrated beyond a doubt that they are strong enough to permit the removal of the ore after the stope has been carried to the top of the orebody or to the level above. This work is in progress at present in the Lowell mine and promises to show very material reduction in cost, as compared to square setting in the same character of ground. Practically no timber is necessary, and as the ore is kept close to the back the workmen are at all times close to the working face. As in the cut-and-fill system, care must always be taken that the workmen bar down and make safe the backs before commencing drilling operations. One disadvantage in the shrinkage system is that should bars of waste occur in the orebody, it must necessarily be mined and there is a danger of it becoming mixed with the ore.

### *Top Slicing*

Another system that has also been receiving attention in these properties is that of the top slice. The top-slicing system probably originated in the iron-ore mines of the northern part of England, and, we believe, was first introduced in this country in the iron mines in northern Michigan. This system consists of first driving in the main-level drifts, crosscutting and finding the extent of the orebody, putting up raises through the orebody to the top of the ore, and commencing operations at the extreme top of the orebody. It must, of course, be demonstrated to the satisfaction of the management that there is no possibility of other orebodies being over the country that is to be mined, as the system, when properly used, does not necessitate filling.

The method simply consists of driving lateral drifts and taking out the ore in small blocks, making sure to clean the top of the orebody, placing either plank or split lagging upon the sill of every individual slice as the

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operations are continued downward, thereby forming a mat, upon which the overburden and débris will rest. It is usually found in the preliminary operations of a top slice that the heaviest weight from the overburden is attained when the first three or four slices are being extracted. After this, the mat, old timber, and overburden become intermixed, and in a measure self-sustaining. This system is applicable and desirable in very soft or wet ground, and it is the writer's opinion that in this character of ground it will prove successful and profitable where square setting and other methods would fail.

Top slicing has been commenced in what is known as the Dividend slice of the Czar mine. This orebody contains perhaps 750,000 to 1,000,000 tons of very soft, wet, aluminous ore, and wherever square setting had been tried, it was found to be very expensive and almost impossible to complete a section successfully. As a preliminary to starting this slice, a drift has been driven in the foot wall on the 400, the orebody lying on the foot wall and extending about 50 ft. above the 200 level. Raises were then put up to the 200 level in the foot wall, as it has been found that raises in the orebody will not stand the terrific pressure which is brought to bear upon them.

On account of the ore being very wet and aluminous, it is very hard to handle in the chutes, and concrete pockets have been designed which have the shape of a funnel, with the large portion of the funnel downward. About 30 ft. above the 400 sill, or in the top of this funnel, an offset or baffle has been put in, and from this point the raise is continued to the 200 level. This raise is circular and lined with concrete, and, while we have not yet proved that this type of pocket will be successful, we are confident that it will materially lower the cost of handling the ore.

The work in this orebody has not progressed to a point where a comparison of costs can be made, but we are sure that the operation will be successful.

One advantage of top slicing is that it is very elastic. Drifts for prospecting may be driven in any direction from any floor, and the waste disposed of in the workings. Another advantage is that in mining the orebody from the top down, the ore is mined clean, and still another is that, wherever it is desirable, some incline raises may be put up at any point from main raises to the mining floor to lower the cost of tramming. Several orebodies in the Czar, Holbrook, Gardner, and Sacramento mines are being developed upon this plan, and the management is of the opinion that they will show a material lowering of costs, as compared to square setting. It must be understood, however, that top slicing can only be used where it will not damage any portion of the mine, and, particularly, it must be demonstrated, as before noted, that there are no orebodies above the territory worked upon this system.

Fig. 5 shows a top slice started and mat formed. Fig. 6 shows ex-

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tremities of orebody being taken out in advance of central portion. This is done where a main extraction tunnel is immediately below the slice, in order to obviate repair costs in main levels.

It is evident to the writer that in future developments in these prop-

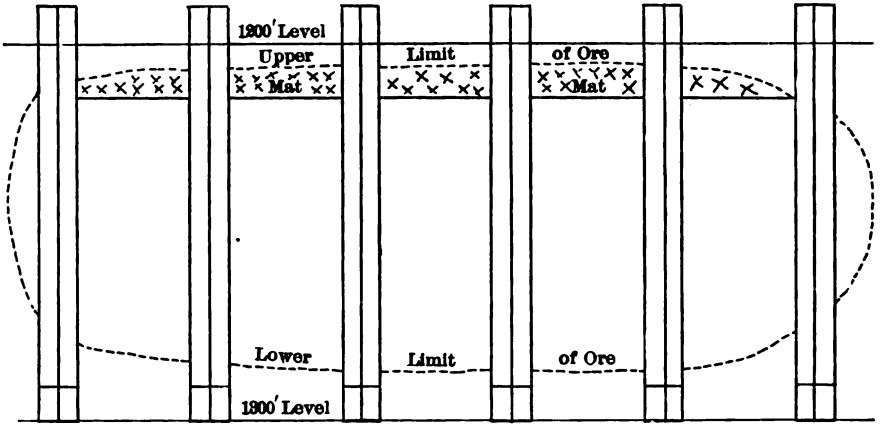


FIG. 5.—TOP SLICING.

erties, wherever orebodies are developed which are adapted either to top slicing, cut-and-fill, or shrinkage methods, these methods will be found to be much more economical than square setting, which has been

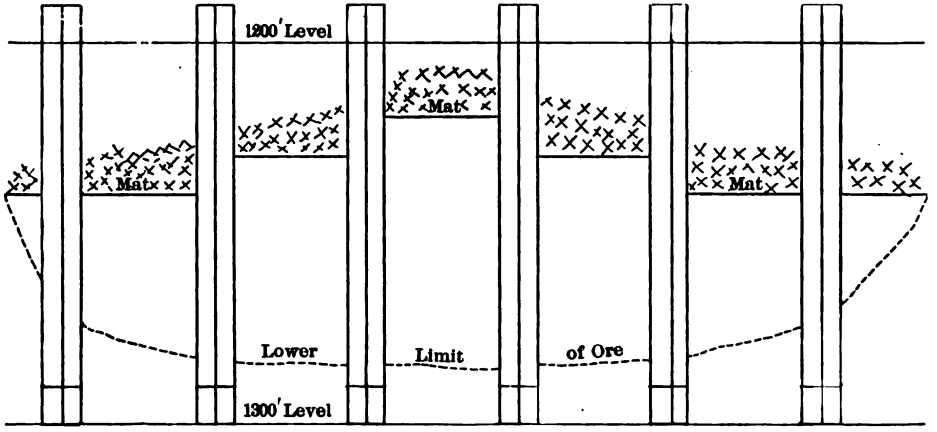


FIG. 6.—TOP SLICING.

in vogue almost exclusively in the past. It must be remembered, however, that there will always be a large proportion of the ore in these properties extracted by the square-set method, as it has undoubtedly some advantages under varied conditions that the other systems do not have.

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### *Ventilation*

Until recently, natural ventilation, aided by exhaust from drill machines, small 5-h.p. blowers, and compressed air, was the only means of ventilating the extensive workings of the entire mine. While the temperature in stopes was not very high, the relative humidity in most places exceeded 90 per cent., which made the mine air oppressive. A mechanical ventilation system was completed in the Gardner during August, 1913. The results thus far attained in improved working conditions and increasing the efficiency of the men have justified the installation of similar systems in the Lowell and Sacramento divisions.

The pressure system of ventilation has been adopted in the Gardner. Two Sirocco blowers, situated near the 900 station, deliver a total of 70,000 cu. ft. of air per minute. This entire volume of air is so coursed as to ventilate the workings from the 1,000 to the 600 levels, whence it exhausts through the shafts of the Calumet Arizona Mining Co.

### *Lowell Fire District*

In the Lowell division, an old fire extends from the 1,000 to the 1,300 levels. Water is being run into this fire area, and after it penetrates the hot zones the water is charged with copper sulphate. On the 1,300 level there is a concrete precipitating plant, which is 500 ft. long and 4 ft. wide. The acid waters percolate among tin cans and scrap iron, and deposit their copper content.

In order to have drifts and raises that are secure for the purpose of conducting gases which come from the fire district, those that are most important have been heavily lined with concrete.

### *Concrete Pockets and Raises*

It has been found very economical for certain kinds of ore to put in concrete pockets and cylindrical raises in storage chutes, as the upkeep cost is practically nothing, whereas, the maintenance of timber in storage chutes is expensive.

Fig. 7 shows a sketch of a concrete pocket built particularly to handle sticky ores from the Dividend slice.

Figs. 8 and 9 show a concrete storage pocket at the Sacramento hoisting shaft designed to handle wet aluminous ores.

### *Copper Queen*

An interesting feature of the mine is that, at present, operations are being conducted through large areas of old stopes, and ore which was regarded as waste in former years can now be mined at a profit. A large amount of this work is being done in the Czar and Holbrook divisions.

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It may also be stated that the mine, during 1913, has produced 867,481 tons of ore, yielding 97,181,725 lb. of copper; 15,573 tons of lead ore have produced 5,701,628 lb. of lead.

About 104,000 ft. of development work was done for the year, of

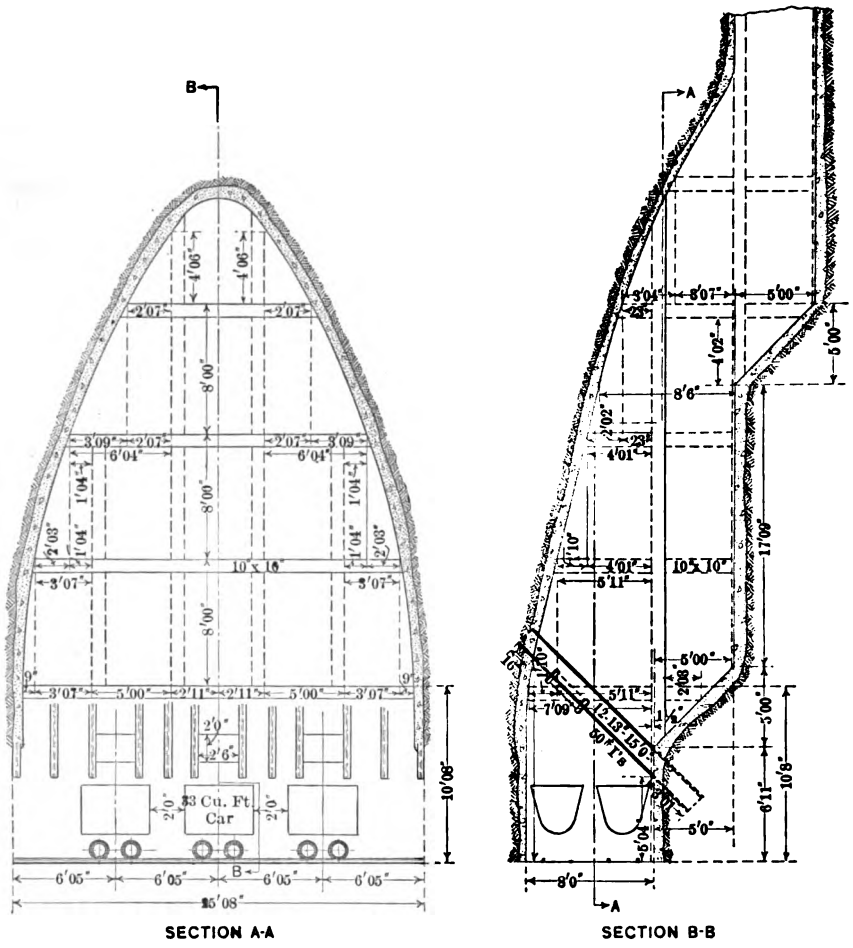


FIG. 7.—SECTIONS OF CONCRETE ORE POCKET.

which 70,000 ft. was done on contract. The amount of timber used for the year was 18,645,713 ft.

The mine has produced a total of 1,176,718,905 lb. of copper to January, 1914.

The Copper Queen, which is often referred to as a mine, constitutes a group of mines that are operated by the Copper Queen Consolidated Mining Co.

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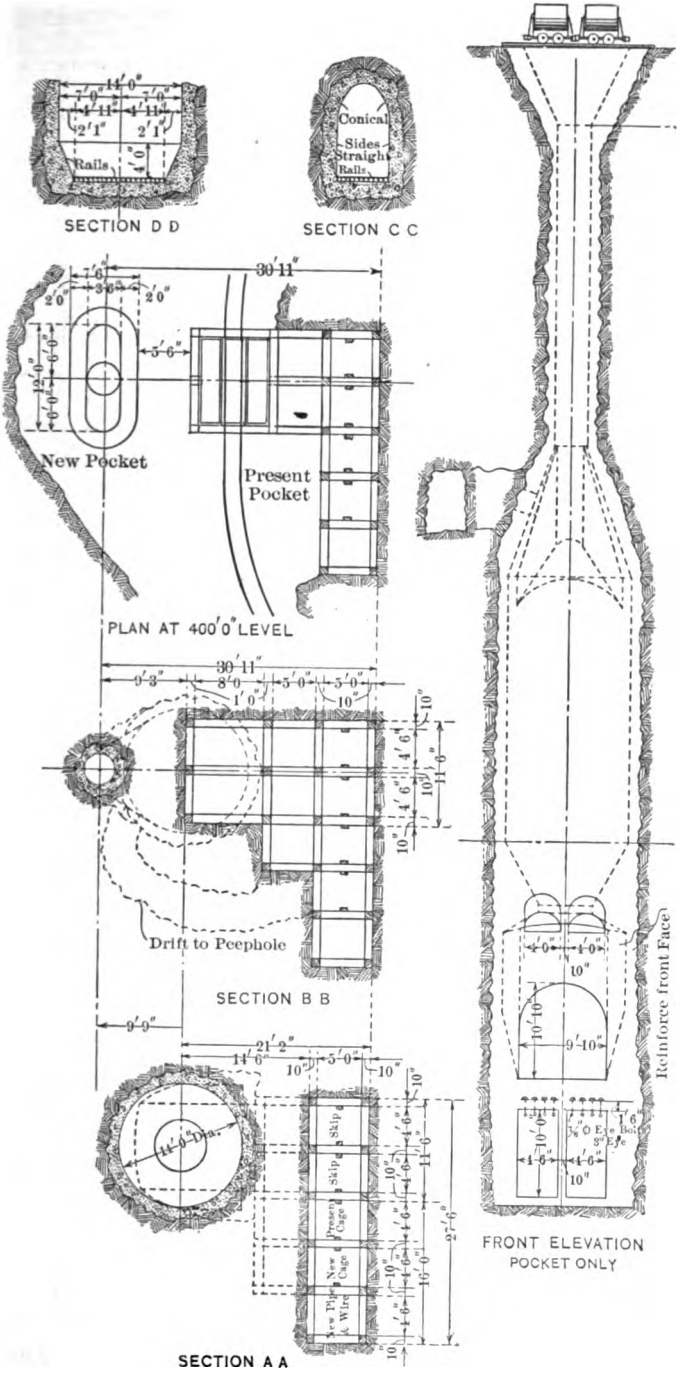


FIG. 8.—PLAN, ELEVATION, AND SECTIONS OF 260-TON CONCRETE ORE POCKET.

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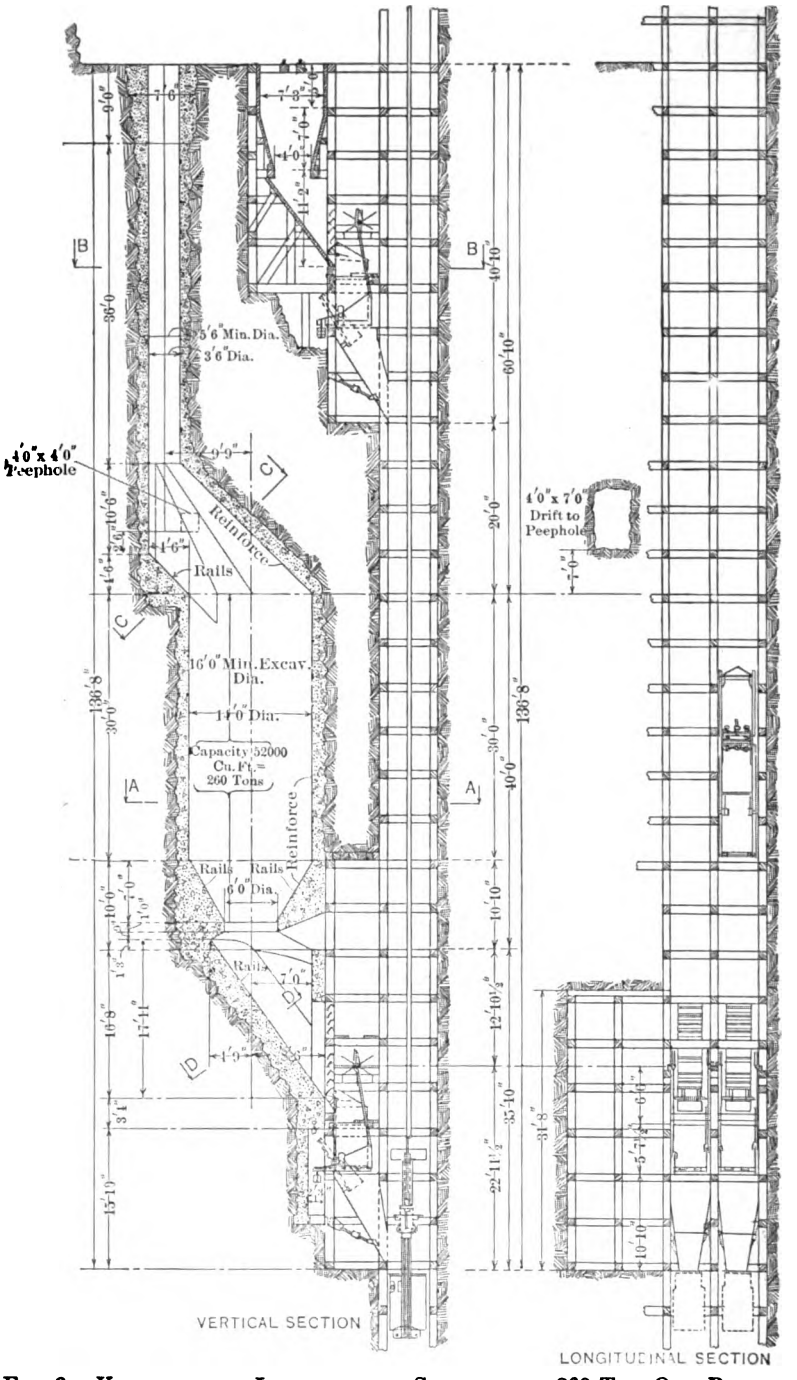


FIG. 9.—VERTICAL AND LONGITUDINAL SECTIONS OF 260-TON ORE POCKET.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the peper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper willclose Oct. 1, 1914. Any discussion offered thereafter should be in the form of a new paper.

# The Design, Construction, and Cost of Two Mine Bulkheads

BY SIDNEY L. WISE AND WALTER STRACHE,* NEW YORK, N. Y.

(Salt Lake Meeting, August, 1914)

WHILE the installation of mine bulkheads to retain water under high pressure is by no means a rarity, the following points which arose in the designing and placing of two of these bulkheads may be of interest:

The Hibernia magnetite mine, located about 40 miles west of New York, in the State of New Jersey, is partly filled with water. This mine is located on an ore lens, the outcrop of which is over a mile long. The ore has been mined from this lens from the surface to a depth of more than 1,500 ft. It was held desirable to separate the old workings from the new, and to allow the former to fill to the 850-ft. level. Two weak places existed between the new and old workings below this level: namely, a temporary bulkhead on the 10th level and a rock bulkhead of indeterminate thickness on the 16th level. When the matter of allowing the old workings to fill was investigated, the fact was developed that if these old workings were filled, the bulkhead on the 10th level would be subjected to a water pressure of about 50 lb. per square inch, and that 200 lb. per square inch would act on the 16th-level bulkhead. As the above-mentioned barriers were not deemed of sufficient strength to permit of these pressures, it became necessary to design and install new bulkheads.

## *Design*

As the 16th-level bulkhead was required to withstand a pressure of 200 lb. per square inch, it presented some difficulties. A careful consideration of the various types of mine dams now in use led to the adoption of a design of the form of a truncated wedge. In this, the pressure side of the dam is of greater area than the back, so that the resultant action is similar to driving the wedge. By cutting generous skewbacks in the walls, roof, and floor, this type in reality becomes an invisible arch. The wedge feature tends to compress the materials in the bulkhead, thereby adding to its imperviousness.

* Non-member.



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Concrete was chosen as the material. In order to lessen the labor and simplify the construction of the forms, straight forms were placed on both the front and back of the dam; the arch in this bulkhead is therefore invisible.

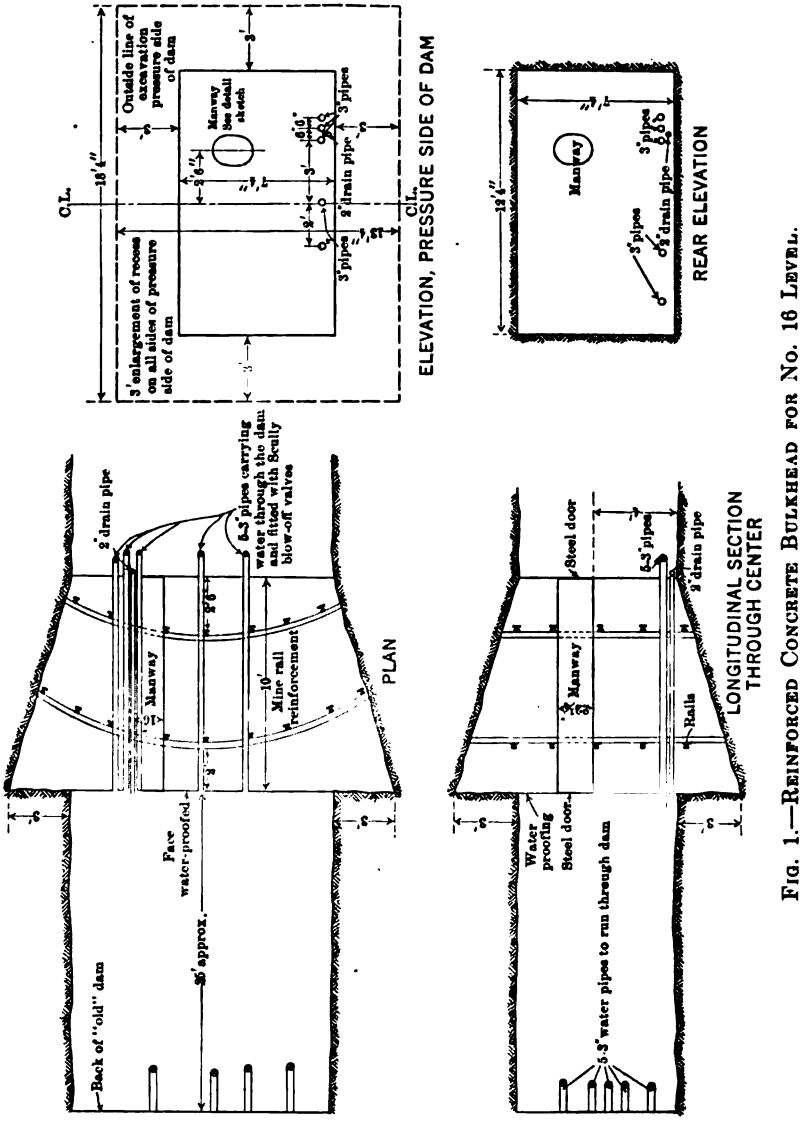


FIG. 1.—REINFORCED CONCRETE BULKHEAD FOR NO. 16 LEVEL.

As ordinary concrete is by no means impervious to water under the head of 200 lb. per square inch, it was decided to water-proof this bulkhead. The most economic method, consistent with good construction, was to face the entire pressure side with a 3-in. layer of a water-proofing com-

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pound. As described in detail under "Construction," this facing was carried up with the concrete, thus insuring a perfect bond. The subsequent test proved the efficacy of this facing.

To provide for future contingencies, involving a possible further waterproofing of the pressure side, it was decided to place a manway through the dam, permitting inspection or repairs.

The relative positions of the bulkheads on the 16th level are shown in

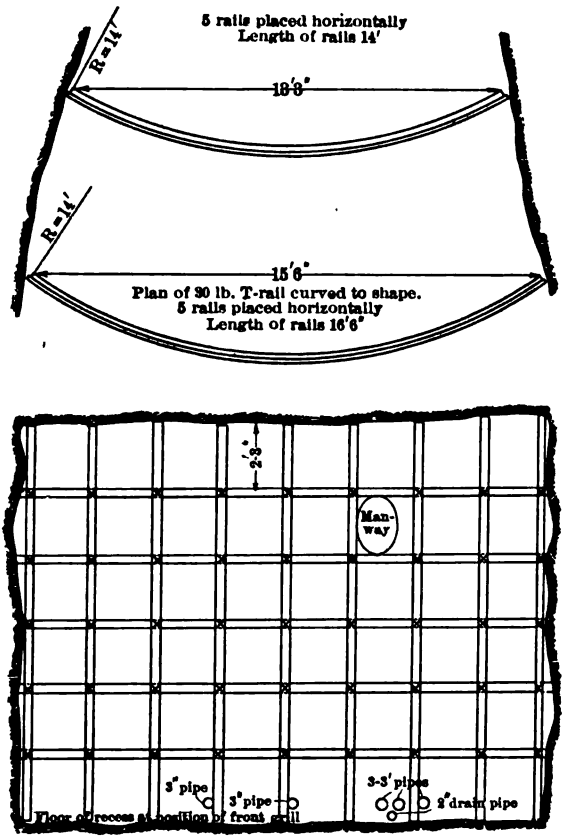


FIG. 2.—DETAIL OF REINFORCEMENT GRILL BUILT OF 30-LB. MINE RAILS FOR NO. 16 LEVEL BULKHEAD. RAILS SPACED AT 2-FT. CENTERS.

Fig. 1. Attention is called to the fact that the old bulkhead leaked to the extent of about 6 gal. per minute, and that five pipes pass through it for the purpose of draining the water in the old workings by pumping it to the surface from the 16th level of the new workings after it had passed through the old bulkhead. The new bulkhead was designed to continue this function of drainage in case it should be so desired, and for this reason prolongations of the pipes pass through it. Due to the

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afore-mentioned leak, when the new bulkhead is completed and its manway sealed, the water which passes through crevices in the old bulkhead will soon fill the space between the old and the new dams. The new bulkhead will then be assuming the entire load.

The details of the steel reinforcement placed in the dam are shown in Fig. 2, the horizontal rails being curved to the radius of the invisible arch. Great care was taken to thoroughly coat all metal surfaces with mortar.

The design of the manway is shown in detail in Fig. 3.

It was decided to use a 1:2:4 mixture of concrete for this bulkhead. Atlas Portland cement was used, and a local sand, carrying less than 3 per cent. of foreign matter, was obtained. The broken stone employed

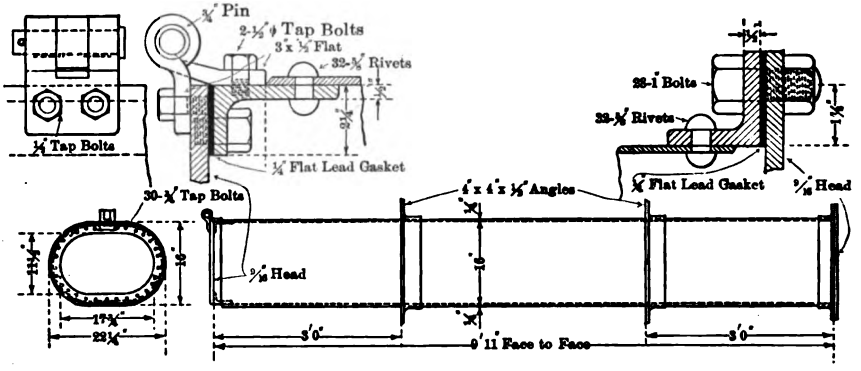


FIG. 3.—DETAILS OF MANWAY.

was a gneiss. This was the result of former milling operations, and was screened and washed before being used.

### *Construction*

The construction of the 10th-level bulkhead presented no great difficulties. Ample storage space was available on this level and all the necessary materials were lowered and stored near the site. After thoroughly cleaning and washing the mushroom-shaped cavity, Fig. 4, the forms were placed, and braced from behind with 6-in. and 8-in. round timbers. The inside of the forms was covered with tar paper, and a 3-in. drain pipe, for possible future use, was run through the forms. This pipe was fitted with a gate valve on the working side. Two-thirds of a cubic yard comprised a batch of concrete, which was mixed rather wet, so that after a batch had been placed water rose slightly above the level mass. Three iron rails were placed across the mouth of the cavity for reinforcement. The 12 yd. of concrete were placed in 10½ hr.

In drilling the recesses for the 16th-level bulkhead, care was taken to

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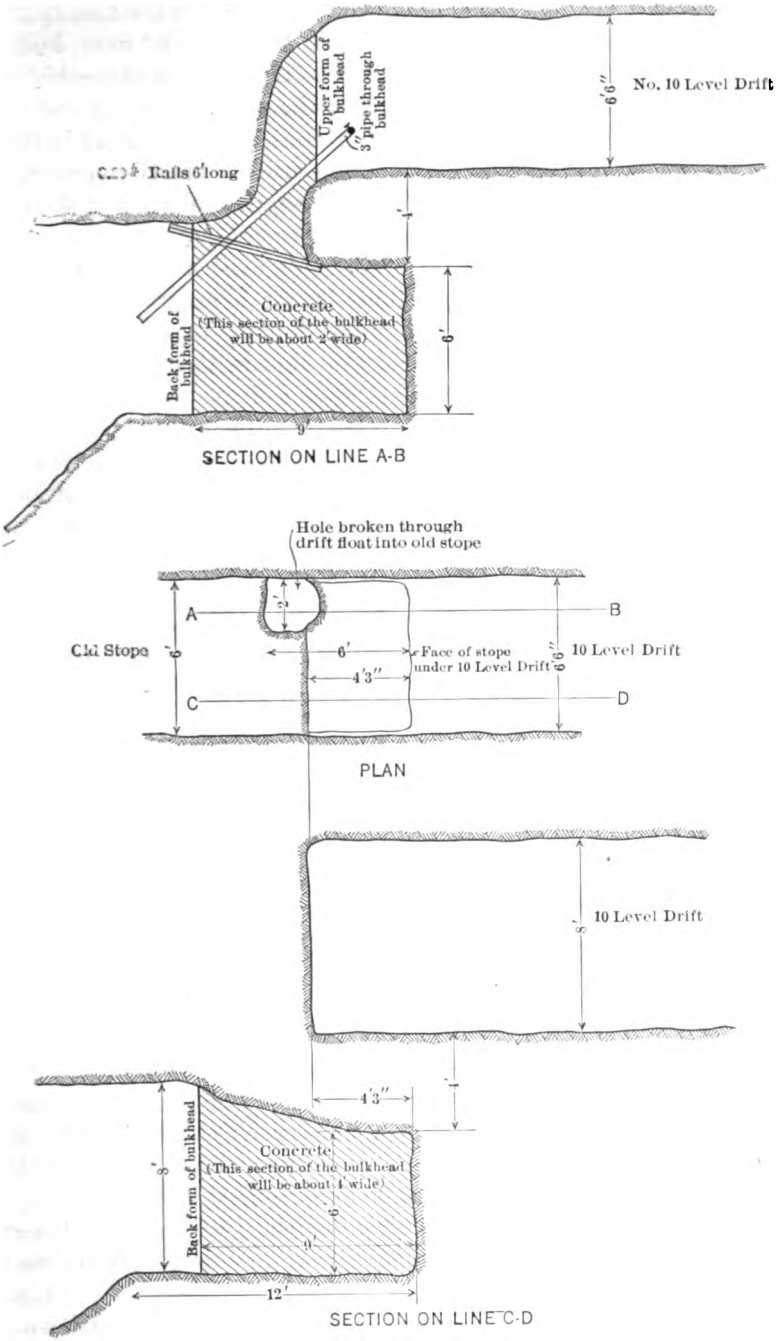


FIG. 4.—BULKHEAD AT NO. 10. LEVEL.

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so point the holes that the excavation would coincide in form and dimensions to the design. At a distance of 25 ft. from the old dam, holes 36 to 39 in. in length and spaced 1 ft. apart were drilled in the sides, roof, and floor, at right angles to the course of the drift. Stoping and hand drills were used in this work and four men constituted the gang in this as well as the subsequent drilling with column drills. Thirty-five feet from the old bulkhead a series of holes 4 ft. in length and from 1 to 1½ ft. apart were placed slanting to conform approximately with the inclinations of the skewbacks. These holes were only burdened with about 1 ft. of ground. Under ordinary conditions longer holes would have been drilled, but the proximity of operating pumps made extraordinary precautions necessary for their protection during the shooting, and so heavy blasting was not attempted. A round of six holes was shot at a time. A third series of holes was drilled slanting to conform with the deeper portions of the recesses. When blasted, this series broke evenly at the line of the 3-ft. holes first drilled and the resultant recess conformed almost exactly to the figure determined upon, and the total excavation agreed with the original estimate of 60 yd. The muck was economically disposed of in a nearby chute.

The materials required for the work were unloaded and stored close to the mouth of the shaft. Due to the lack of space in which to store the materials on the 16th level, the matter of lowering and delivering the required materials without interrupting the work was one of the most troublesome obstacles encountered. Eight to twelve men were employed on the surface in sacking sand and stone while the excavation was in progress on the 16th level. The empty bags produced as the cement was used augmented the 200 old cement bags purchased for the sacking. While enough sand could be stored on the 16th level for this entire bulkhead, there remained insufficient room for the storage of the daily requirements of stone and cement. It was found advantageous, therefore, to employ a small night crew, who lowered much of the material required for the next day's work.

The ten curved rails were bent to a 14-ft. radius over a form in half a day. This was done on the surface.

The forms on the 16th level bulkhead were built of 2-in. undressed lumber and 6 to 10 in. round posts were used for studding and braces. The forms were thoroughly braced and were wired to stiffen them further. The interior faces of the forms were covered with tar paper, and the junction of the forms with the rock was plastered with a 1:1 cement mortar on all sides. The pressure-side forms were carried to the roof of the level at once, but did not extend into the recess.

The recess was thoroughly cleaned of loose rock and washed down, and all the reinforcing material, pipes, and the manway were placed in position before the concreting was started. Furthermore, the floor and sides

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of the recess were plastered with a 1:1 cement mortar before placing the concrete.

The concrete was made of one part cement, two parts sand and four parts stone, these proportions being determined by actual measurement. A batch of concrete contained $\frac{2}{3}$ cu. yd. The sand was first placed on the mixing platform and the heaps flattened down. On this was emptied the cement, and these two materials were thoroughly mixed and flattened out before receiving the stone. This mixing took place about 12 ft. from the front form of the bulkhead. Enough water was used to make a wet mixture. Two men did the first mixing and turned the mass, then passed it on to the next two, who again turned it, passing the finished concrete to the last two men at the mixing board. These men shoveled directly into the form. In this manner, while each two men received a short rest of a few minutes between batches, fresh material was being placed on the starting end of the mixing platform while the men nearest the form were still disposing of the concrete mixture. This also insured a thorough mixing. One man remained in the form to level off each batch. The best day's work consisted in the placing of 12 yd. of concrete.

The water-proofing compound, "Impervite," was carried up as a 3-in. facing, its level being kept the same as that of the concrete. An even thickness of the water-proof layer was maintained by the use of three forms of $\frac{3}{16}$-in. plate, 6 ft. long by 6 in. wide, fitted at the upper corner with 3-in. spreading bolts. These forms, placed across the entire width of the face, were raised 3 to 4 in. at a time, and enough concrete was then shoveled against them to keep them in place. The almost semi-liquid water-proofing compound was mixed on the level and was carried to the forms in buckets.

Before leaving at night, sharp man-size (about 100-lb.) stones were set at least 6 in. apart in the concrete mass. This made a strong bond, and before concreting the next day this rough surface was freshly plastered with a thin 1:1 mortar.

As the roof was reached, false forms were placed, and the work was finally finished in tightly bonded dovetailed blocks.

Throughout the work, the leakage from the old dam, 6 gal. per minute, passed through the 2-in. drain pipe of the bulkhead.

Seven 2-in. grout pipes, four on the pressure side and three on the opposite side, were placed in the concrete as the work neared completion. They were all located near the roof and were directed to such places as were most difficult to fill with concrete. As the work had to be hurried, but a day and a half elapsed after completion of the cement work before grouting was begun. The grout mixture was a mortar consisting of one and one-half parts of sand to one part of cement made fluid with water-dissolved "Impervite." A mine-made grout "gun" was used, and the grout was forced successively into the several pipes by means of air

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under the pressure of 85 lb. per square inch. As the grout was forced through the different pipes the ejection of some of this material through the other pipes indicated that the greater voids were filled. As the "gun" connections were changed those pipes giving the greatest discharge were plugged, and the discharge was finally limited to one pipe. This too, was filled and plugged. The first day's grouting was allowed to set over night, and the following day all the pipes were again tested. This time there was no communication between the pipes, and as little or no grout could be forced into any one of the pipes the grouting was considered most satisfactory.

Three weeks were determined upon as the period which should elapse before the new bulkhead should receive any load. During this time the 2-in. drain pipe was left open.

At the expiration of this time the completed 16th-level bulkhead was tested by pumping water up to the pressure of 160 lb. per square inch into the space between the old and new bulkheads through the 2-in. drain pipe. The results were entirely satisfactory, as the total seepage amounted to only ½ gal. per minute at first. This small leakage subsequently stopped almost completely.

### *Cost*

A cheap class of labor was employed exclusively, the men receiving $2 per 10-hr. shift.

Following are tables showing the cost of the work. The interference caused by the necessity of keeping two large pumps in operation within 50 ft. of the 16th-level bulkhead was perhaps the greatest cause for the apparent high cost. The labor cost of lowering materials was also very high for the amounts handled, which, in the case of the 16th-level bulkhead, had to be lowered 1,350 ft. in one skip.

*Summary of Costs*

| Division | 16th-Level Dam | 10th-Level Dam |
|---|---|---|
| Labor | $790.00 | $134.00 |
| Superintendence | 130.00 | 30.00 |
| Transportation | 50.46 | 4.50 |
| Materials | 503.88 | 37.23 |
| Totals | $1,474.34 | $205.73 |

*Costs per Cubic Yard*

| Division | 16th-Level Dam | 10th-Level Dam |
|---|---|---|
| Labor | $13.17 | $11.15 |
| Superintendence | 2.17 | 2.50 |
| Transportation | 0.84 | 0.37 |
| Materials | 8.38 | 3.10 |
| Totals | $24.56 | $17.12 |

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Economy and Efficiency in Reverberatory Smelting

BY C. D. DEMOND, ANACONDA, MONT.

(Salt Lake Meeting, August, 1914)

IN reverberatory smelting, fuel is the chief item of expense, as it commonly is in processes using large percentages of it. Hence the most suitable supply is eagerly sought; that is, the supply which, in the end, yields the greatest net profit. The better grades are not infrequently bought, even at an advanced price, because they are apt to be decidedly more economical. High price, of itself, however, is no merit, for freight is generally the largest part of the cost of fuel, and the nearby supply may yield as high efficiency as that from a greater distance. If a poorer grade is reasonably good, and if its cost is low enough to more than offset the decreased technical efficiency, it is of course the more suitable.

Any smelting operation depends for both efficiency and economy on raising the furnace atmosphere considerably above the "critical" temperature; that is, above the temperature at which smelting is completed. If this temperature is not exceeded, smelting stops, no matter how much coal is burned; and the real value of a fuel depends on the number of degrees that the temperature is continuously kept above this point. For example, if one fuel maintains 2,500° F. and another only 2,100° F., while the "critical" temperature is 2,000° F., the first is worth at least five times as much as the second, although the increase above atmospheric temperature is only one-fifth more in one case than in the other. This "margin of temperature" above the smelting point is much more significant than the total number of heat units developed from the fuel. The case is parallel to that of water flowing from a pipe. The reservoir may contain a billion gallons of water; but, even though the fluid has come to the nozzle, not a drop will run out, where it can be of practical use, unless the level in the reservoir is higher than the nozzle; and the speed and volume of flow depend on how much higher the level is in the reservoir than at this point. If the temperature in a furnace is not above the critical level there will be no slag to flow over the skimming plate, even though there are enough heat units to bring an unlimited quantity of charge up to the point of slag formation; and just as the value of a water supply for the purpose of putting out a fire depends largely on the effective head, so the difficulties of furnace management

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vary inversely with the margin of temperature,[1] provided other conditions are alike in the different cases. The most important of these other conditions is the presence of a good flame in the hearth, which is referred to in the discussion of several of the fuel tests and is taken up in some detail under "Character of the True Coal" (p. 1856).

The heat must be applied at the right place, and the right place in a reverberatory furnace is the charge on the hearth. Some experiences are here presented showing the unfortunate condition, with some coals, of getting an excessive proportion of the heat into the flue or on the grate, and the high cost, in some cases, of getting the charge above the critical temperature.

### Experiences with Various Fuels

At its old reduction works, the Anaconda Copper Mining Co. used wood for reverberatory smelting during a short period in its early history; but this fuel was by no means as efficient as fairly good coal; nor was it as economical, even though it was cut within 5 or 10 miles of the plant, while the coal had to stand the freight charge for a haul of 200 miles in one case and 500 in another.

Coal from Belt, Mont., was tried; but in a 20 by 12 ft. furnace with a 4 by 5 ft. fire box this could not be successfully used with natural draft, for four reasons: 1, there was a large loss of coal through the grate because the ash did not clinker, and the draft was only enough for a comparatively thin bed on the grate; 2, it contained too little of the heavy hydrocarbons to maintain a proper flame in the hearth; 3, its total heating value was too small to develop a very high temperature under any circumstances; and 4, the large amount of ash (25 to 30 per cent.), intimately mixed with the true coal, so interfered with efficient combustion as to still further increase the difficulty of getting a high temperature. With forced draft, however, and with an 8 by 5 ft. fire box, enough flame was driven into the hearth so that moderate smelting conditions were obtained; and it was possible to carry such a depth of ash that the loss of fuel through the grate was much reduced. The intense forced-draft combustion at the bottom of the fuel bed, however, caused troublesome clinkering. The extent of this will be appreciated when it is stated that, although steam was introduced with the blast to lessen the clinker, so much of the latter built up that the entire grate had to be dumped and a new fire started every 12 hr. After using Belt coal for a year, a coal from Diamondville, Wyo., was adopted, and was successfully used with natural draft. The change resulted in a 40 per cent. decrease of the fuel cost per ton of material smelted; and it increased the capacity of the fur-

[1] The nicest proof of this principle of critical temperature and the margin is found in the explanation of Gayley's success in applying dry blast to iron smelting. See J. E. Johnson, Jr., *Trans.*, xxxvi, 472 (1905), and H. M. Howe, *Trans.*, xxxvii, 216 (1906).

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naces 66 per cent., thus leading to a decrease in the other items of cost per ton. This was a case where a very decided increase in the price of coal resulted in a remarkable net saving.

The size of furnaces was gradually increased at the old works till hearths 42 by 14 ft. were built, which resulted in striking economies of fuel, labor, and repairs. This increase continued at the new (Washoe) works with even greater advantage, as already recorded in the *Transactions*,[2] and the benefits of this enlargement were so marked that the management determined to find whether the low-priced Belt coal could be successfully used with strong natural draft (1.6 in. water column in the flue close to the furnace and 0.75 in. at the bridge) in the fire box 7 by 16 ft. by 24 in. deep attached to a 112 by 19 ft. hearth, which are the dimensions of most of the present furnaces. The trial was made in a furnace that was already in normal operation with Diamondville coal; but after the Diamondville coal was all burned out only one 15-ton charge could be smelted, and even this could not have been done except for the reserve heat in the immense pool of slag and matte (180 tons combined) that is kept in the hearth. The furnace froze an hour before the end of the test, though using Belt coal at the rate of 98 tons per 24 hr. After getting back to Diamondville coal, it took 7 hr. to get the charge hot enough, in front, to skim; though it was hot enough at the back to take a charge in 3 hr. The usual rate of smelting was regained in about 8 hr. Before the test began, the flue temperature averaged 2,200° F., as usual; for 4 hr., while the fire box contained both Diamondville and Belt coal, it was 2,000° F.; but for the last 2½ hr., with Belt coal alone, it was only 1,660° F. During the first 7 hr., with a good clinker bed, and with considerable heat from Diamondville coal, the boilers attached to the furnace developed 584 boiler horse power; but during the last 8 hr., with Belt alone, they developed only 389 h.p. This still left Belt coal with its bad reputation; but several years later, when on account of inherent losses in gas producers the management of the company's Great Falls plant pretty definitely decided to abandon their producers, which were situated at a considerable distance from the furnaces, it was thought that this coal might be a commercial success at Great Falls, in a gas-producing fire box attached directly to the furnace, the freight being much less from Belt to Great Falls than to Anaconda, while the freight on Diamondville coal is higher to Great Falls than to Anaconda. They had always used, in their gas producers, coal from near Belt, and of the same general character as the Belt coal. Since their regenerative furnaces, designed like an open-hearth steel furnace, were not so well adapted to such an experiment as the furnaces at Anaconda, a 102 by 19 ft. hearth at the latter plant was equipped with a fire box that had the immense shaking-grate area of 19 by 21 ft. Under these conditions smelt-

[2] E. P. Mathewson and William Wraith: *Trans.*, xliv, 783 (1912).

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ing succeeded even with natural draft, as shown in trials 1 and 3 of Table I. The surface of the fuel was kept 5½ ft. above the grate, this being the best depth found by experience. In the first trial nothing but roasted concentrate ("calcine"), containing a small amount of lime rock flux, was smelted; but in No. 3 there was added a portion of flue dust, which is harder to smelt than calcine. In parallel runs with Diamondville coal in one of the regular furnaces (trials 2 and 4 in the table) the results were normal when smelting calcine alone, but 8 per cent. better than normal when flue dust was added to the charge. The comparison of coals was still decidedly in favor of Diamondville normal results, the tons smelted per ton of coal being a good deal more than double, the cost for coal being 36 per cent. less per ton of material smelted, and the tonnage smelted per furnace being about 20 per cent. greater. Moreover, Diamondville requires only three men per furnace on each 8-hr. shift, while the Belt required six men at best. Most of the extra labor was due to the troublesome ash and clinker.

With forced draft in the gas-producing fire box, and carrying the surface of the fuel bed 7 ft. above the grate, the consumption of Belt coal was 19 per cent. less than with natural draft, while the smelting capacity was increased 12 per cent., thus increasing the ratio between tons of material smelted and tons of coal used from 1.99 to 2.74 (compare trial 5 with trial 1 in Table I). However, even this was much short of Diamondville results, which yielded 9 per cent. less cost for fuel, per ton smelted, and 8 per cent. more smelting capacity for the furnace (compare trials 2 and 5); and there was, with Belt coal, the decidedly increased labor cost, a much larger capital outlay for equipment, the power to supply blast, and a large consumption of steam under the grate to prevent clinkering. The power used for forced draft was 23.1 h.p., and 11.5 h.p. for the secondary air introduced over the bridge. The steam used under the grate was 4,600 lb. per hour (143 boiler horse power). Even with natural draft there was a steam consumption of 2,220 lb. per hour (69 boiler horse power). The steam generated in the boilers attached to the Belt furnace was 532 boiler horse power with forced draft, and 619 with natural draft. In trial No. 4, with Diamondville coal and natural draft, the horse power developed was 469.

The total cost under these best conditions for Belt coal would be at least 15 per cent. more than with Diamondville at Anaconda. Possibly some other style of gas producer would be better; but it is not likely that Belt coal can ever compete with good Diamondville here, unless by some very radical change in conditions. At Great Falls, however, with the very different freights from the two coal camps, Belt may be the more economical coal.

Rock Springs (Wyoming) coal was used at the old works for several years; but was not as good as Diamondville, because it flashed too much

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TABLE I.—*Reverberatory Tests at Anaconda*

| No. | Coal and Draft Conditions | Tons Smelted per Furnace per 24 hr. | | | Tons Coal per 24 hr. | Ratio (Tons Smelted per Ton of Coal) | "Efficiency" (Tons Smelted per Million B.t.u.) | Boiler Horse Power |
|---|---|---|---|---|---|---|---|---|
| | | Calcine | Flue Dust | Total | | | | |
| 1 | Belt run of mine; gas-producer fire box; natural draft. | 247.2 | ..... | 247.2 | 124.3 | 1.99 | 0.122 | 619 |
| 2 | Diamondville run of mine; parallel to preceding. | 297.8 | ..... | 297.8 | 64.5 | 4.62 | 0.207 | ..... |
| 3 | Belt run of mine; gas-producer fire box; natural draft. | 212.2 | 19.3 | 231.5 | 117.4 | 1.97 | 0.120 | ..... |
| 4 | Diamondville run of mine; parallel to preceding. | 269.3 | 21.5 | 290.8 | 58.8 | 4.95 | 0.221 | 469 |
| 5 | Belt run of mine; gas-producer fire box; forced draft. | 275.8 | ..... | 275.8 | 100.8 | 2.74 | 0.164 | 532 |
| 6 | Bear Creek run of mine; forced draft and 38-in. fuel bed. | 118.0 | ..... | 118.0 | 65.1 | 1.81 | 0.089 | 438 |
| 7 | Bear Creek run of mine; forced draft and 24-in. fuel bed. | 180.0 | ..... | 180.0 | 80.4 | 2.24 | 0.110 | ..... |
| 8 | Diamondville run of mine, parallel to the two preceding. | 256.6 | 10.2 | 266.8 | 60.8 | 4.39 | 0.178 | 458 |
| 9 | Bear Creek run of mine; natural draft. | 201.7 | ..... | 201.7 | 81.6 | 2.47 | 0.122 | 585 |
| 10 | Bear Creek run of mine; forced draft. | 237.0 | ..... | 237.0 | 75.1 | 3.16 | 0.153 | 582 |
| 11 | Diamondville run of mine; parallel to the two preceding. | 234.9 | 21.6 | 256.5 | 63.1 | 4.06 | 0.177 | ..... |
| 12 | Bear Creek run of mine; natural draft. | 199.7 | ..... | 199.7 | 86.7 | 2.30 | 0.112 | 557 |
| 13 | Diamondville run of mine; parallel to preceding. | 303.2 | ..... | 303.2 | 62.6 | 4.84 | 0.214 | ..... |
| 14 | Bear Creek run of mine; natural draft. | 160.8 | 26.5 | 187.3 | 87.3 | 2.15 | 0.104 | 559 |
| 15 | Diamondville run of mine; parallel to preceding. | 216.1 | 34.2 | 250.3 | 57.6 | 4.35 | 0.187 | 522 |
| 16 | Bear Creek run of mine; forced draft. | 234.7 | ..... | 234.7 | 82.8 | 2.83 | 0.133 | 568 |
| 17 | Diamondville run of mine; parallel to preceding. | 299.6 | ..... | 299.6 | 57.2 | 5.24 | 0.222 | ..... |
| 18 | Bear Creek run of mine; forced draft. | 204.2 | 24.2 | 228.4 | 85.5 | 2.67 | 0.125 | 571 |
| 19 | Diamondville run of mine; parallel to preceding. | 237.9 | 30.4 | 268.3 | 56.0 | 4.79 | 0.207 | 507 |
| 20 | Roundup run of mine; natural draft. | 219.5 | ..... | 219.5 | 92.9 | 2.36 | 0.114 | 537 |
| 21 | Diamondville run of mine; parallel to preceding. | 303.5 | ..... | 303.5 | 59.8 | 5.08 | 0.222 | ..... |
| 22 | Roundup run of mine; natural draft. | 200.3 | 19.5 | 219.8 | 90.6 | 2.43 | 0.117 | 525 |
| 23 | Diamondville run of mine; parallel to preceding. | 248.6 | 28.8 | 277.4 | 57.4 | 4.84 | 0.205 | 433 |
| 24 | Roundup run of mine; forced draft. | 246.2 | ..... | 246.2 | 88.5 | 2.79 | 0.131 | 575 |
| 25 | Diamondville run of mine; parallel to preceding. | 323.8 | ..... | 323.8 | 56.9 | 5.69 | 0.248 | 473 |
| 26 | Roundup run of mine; forced draft. | 213.2 | 20.2 | 233.4 | 96.0 | 2.43 | 0.118 | 615 |
| 27 | Diamondville run of mine; parallel to preceding. | 297.4 | 25.8 | 323.2 | 58.9<br>B.C.19.38<br>Dia.43.03 | 5.49 | 0.232 | ..... |
| 28 | Bear Creek lump and Diamondville mine run; natural draft. | 219.4 | 28.6 | 248.0 | 62.41 | 3.97 | 0.176 | ..... |
| 29 | Diamondville; parallel to preceding. | 211.9 | 42.1 | 254.0 | 58.7 | 4.33 | 0.190 | ..... |
| 30 | Diamondville lump..... | 294.7 | 33.6 | 328.3 | 63.3 | 5.19 | 0.211 | ..... |
| 31 | Roundup lump; natural draft. | A 218.7 | ..... | 218.7 | 89.3 | 2.45 | 0.117 | ..... |
| | | B 231.4 | ..... | 231.4 | 85.0 | 2.72 | 0.132 | 607 |
| | | C 263.4 | ..... | 263.4 | 79.4 | 3.32 | 0.159 | 532 |

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in the fire; that is, it gave off its volatile matter so rapidly that there was a good flame in the furnace for a few minutes, but for most of the time there was practically no flame. To minimize this effect, the fire-box temperature was kept as low as possible, by letting a thick bed of ash accumulate. This procedure also prevented much loss of fuel through the grate; but, as the ash was fine and did not clinker, the grating had to be done cautiously. A furnace would use about 25 per cent. more of Rock Springs than of Diamondville coal, but would not smelt more than 90 per cent. as much material.

A number of trials have been made at the new works with mine-run coal from Bear Creek, Mont. The first shipment was received before the mines were much developed; and the natural-draft results were like those with Belt. (The Belt mines, however, were well developed). With the coal 24 in. deep in the 7 by 16 ft. fire box the furnace froze in 6 or 8 hr., despite the best efforts with different methods of firing, and while using coal at the rate of 72 tons in 24 hr. The cause of this failure was lack of flame. This same shipment of coal succeeded, however, under forced draft, even though in one case it was used at a slower rate than during the natural-draft trial; and this was due to the intense combustion on the grate distilling some heavy volatile matter into the hearth, and developing flame there. This is parallel to the experience with Belt coal. Having the Bear Creek coal 38 in. deep, with an average air pressure of 2.6 in. under the grate, 118 tons of calcine were smelted per 24 hr., with 65 tons of coal (ratio 1.8); but with a fuel bed only 24 in. deep, and a 1.7 in. under-grate pressure, the decidedly better result was obtained of smelting 180 tons of calcine with 80 tons of coal (ratio 2.2). (See trials 6 and 7 in Table I.) A parallel run with 61 tons of Diamondville coal smelted 267 tons of calcine and flue dust (ratio 4.4), a normal result with this coal. (See trial 8.) The fuel cost for even the better of the Bear Creek results was 75 per cent. higher, per ton smelted, than for the Diamondville result, and would be nearly 100 per cent. higher if flue dust had been included in the charge, as it was with Diamondville, because of the slower smelting of dust.

Even with forced draft a really good flame could not be obtained from this lot of Bear Creek coal; but after 15 months further development of the mines the coal was of a quality to give fairly good flame without forced draft. The efficiency, however, was 25 per cent. better with forced than with natural draft, as indicated by the ratio and "efficiency" columns for trials 9 and 10 in Table I. Trial 10 was the most efficient and economical of all the tests of Bear Creek coal, but its fuel cost per ton smelted was 14 per cent. higher than for Diamondville in a parallel test (trial 11), and 30 per cent. higher than for normal Diamondville results when the charge, as in trial 10, contains no flue dust. Another

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large shipment, six months later, was not as good as the one just mentioned (see trials 12, 14, 16, and 18 of the table.)

Another Montana mine-run coal, from Roundup, gave results which, as a whole, were about the same as with the Bear Creek (see trials 20 to 26), but the Roundup netted 5 to 10 per cent. less cost per ton smelted, because of a lower freight rate. The cost, however, averaged nearly 70 per cent. more with Roundup than with Diamondville coal, although the price of a ton of Roundup was one-sixth less than for Diamondville.

## What Constitutes Good Fuel

So far as the coal is concerned, there are four factors that affect smelting efficiency: 1, the character of the true coal substance; 2, the percentage of fines; 3, the percentage of moisture; and 4, the character and percentage of ash; and these call for some discussion.

### *Character of the True Coal*

As to the character of the real coal substance, something can be judged from proximate analyses; but these must be used with a good deal of caution, for they may be very misleading in regard to the ratio of fixed carbon to the volatile portion of the real coal, and they tell nothing as to the composition and qualities of the volatile matter. Frank Haas has shown that the two coals whose analyses are quoted below are undoubtedly identical in the character of the coal substance, though the proximate analyses suggest quite otherwise when considered in the usual way. They are from different parts of the same bed. His conclusion is based on intimate practical familiarity with the coals; on a study of the ultimate analyses; on consideration of the sulphur included as part of the volatile matter; and on changes in the ash which affect the apparent percentages of volatile matter and fixed carbon.[3]

*Proximate Analyses*

| | Moisture | Volatile Matter | Fixed Carbon | Ash | Sulphur | Ratio of F. C. / Vol. |
|---|---|---|---|---|---|---|
| | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | |
| No. 2.... | 1.76 | 35.06 | 56.54 | 6.64 | 1.63 | 1.61 |
| No. 9.... | 1.21 | 39.78 | 51.90 | 7.11 | 3.54 | 1.30 |

*Ultimate Composition of the True Coal*

| | Carbon Per Cent. | Hydrogen Per Cent. | Nitrogen Per Cent. | Oxygen Per Cent. | Ratio of $\frac{C}{H}$ |
|---|---|---|---|---|---|
| No. 2..... | 82.76 | 5.82 | 1.69 | 9.73 | 14.2 |
| No. 9..... | 82.98 | 5.98 | 1.50 | 9.54 | 13.9 |

[3] *West Virginia Geological Survey*, vol. ii (A), p. 651 (1908); and *Bulletin* No. 10, Consolidation Coal Co., Fairmont, W. Va.

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Mr. Haas's proximate analyses appear to have been made with a good deal of care; but with less care the results are still more misleading. The analyses in Table II for our trials 9, 10, and 11 illustrate what happens when the conditions, as to time and temperature, are not according to the standard directions for determining volatile matter. The ratio of volatile matter to fixed carbon, in these three cases, is respectively 1.01, 1.06, and 1.01; but in no other case in the table is the ratio higher than 0.89 for either of these coals. These three analyses were made together; and the fact that the ratio is equally high for both Bear Creek and Diamondville is practically conclusive evidence that the samples were heated too much in the determination of volatile matter. The errors were not appreciated till it was too late to make checks.

Even the moisture determination may lead to errors in the apparent percentage of volatile matter; P. L. Teed shows that in the standard method for moisture the loss from some coals amounts to 2 per cent. more than the true moisture.[4]

We know that while the volatile matter contains hydrocarbons, H and CO (combustibles), it also contains N, $CO_2$ and $H_2O$ (non-combustibles), this $H_2O$ having evidently been combined as part of the coal; and that the relative quantities of these different substances differ greatly from coal to coal. Porter and Ovitz found the the non-combustible gases from several coals (not including mechanically absorbed moisture), all treated in a similar way, varied from 1 per cent. to at least 15 per cent. of the total coal.[5] The ultimate analyses made by the government bureaus show oxygen content of unweathered bituminous coals, after reducing to an ash and moisture free basis, ranging from 5 to 17 per cent.; and in samples taken near the outcrop, where weathering has occurred, this figure runs as high as 27 per cent.[6] This oxygen not only displaces, so to speak, an equal amount of carbon and hydrogen, but also renders useless, for heat production, a considerable part of the carbon and hydrogen that are present, by being already combined with them. David White has shown, by examination of a very large amount of data, that each per cent. of oxygen has the same effect as 1 per cent. of ash in reducing the heating value shown by a calorimeter.[7] The commercial value is reduced in still greater measure, because the gas resulting from this oxygen has to be heated to the furnace temperature at the expense of the heat-producing portion of the coal, thus still further reducing the available heat and the margin above the critical temperature.

[4] *Bulletin No.* 104, *Institution of Mining and Metallurgy* (May 15, 1913).

[5] *Bulletin No.* 1, *U. S. Bureau of Mines* (1910).

[6] *Bulletin No.* 22, *U. S. Bureau of Mines* (1913).

[7] *Bulletin No.* 382, *U. S. Geological Survey* (1909), reprinted as *Bulletin No.* 29, *U. S. Bureau of Mines.*

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TABLE. II.—*Reverberatory Coals Used at Anaconda*

| No. | Kind of Coal. | Quality of Coal | | | | |
|---|---|---|---|---|---|---|
| | | Moisture | Volatile | Fixed Carbon | Ash | B.t.u. per pound |
| 1. | Belt run of mine..... | 8.5 | 23.6 | 38.7 | 29.2 | 8,180 |
| 2. | Diamondville run of mine.. | 8.0 | 36.6 | 45.4 | 10.0 | 11,180 |
| 3. | Belt run of mine ..... | 8.5 | 23.6 | 38.7 | 29.2 | 8,180 |
| 4. | Diamondville run of mine.. | 8.0 | 36.6 | 45.4 | 10.0 | 11,180 |
| 5. | Belt run of mine..... | 6.7 | 23.7 | 38.7 | 30.9 | 8,340 |
| 6. | Bear Creek run of mine... | 11.7 | 33.5 | 38.4 | 16.4 | 10,140 |
| 7. | Bear Creek run of mine.... | 11.7 | 33.5 | 38.4 | 16.4 | 10,140 |
| 8. | Diamondville run of mine.. | 6.7 | 39.7 | 45.3 | 8.3 | 12,310 |
| 9. | Bear Creek run of mine.... | 13.8 | 37.7 | 37.2 | 11.3 | 10,110 |
| 10. | Bear Creek run of mine.... | 13.1 | 39.4 | 37.1 | 10.4 | 10,340 |
| 11. | Diamondville run of mine.. | 6.9 | 41.4 | 41.0 | 10.7 | 11,470 |
| 12. | Bear Creek run of mine.... | 11.2 | 36.1 | 42.9 | 9.8 | 10,300 |
| 13. | Diamondville run of mine.. | 6.4 | 38.9 | 44.4 | 10.3 | 11,310 |
| 14. | Bear Creek run of mine.... | 11.2 | 35.6 | 43.7 | 9.5 | 10,300 |
| 15. | Diamondville run of mine.. | 6.5 | 39.4 | 44.9 | 9.2 | 11,600 |
| 16. | Bear Creek run of mine.... | 8.9 | 35.8 | 44.6 | 10.7 | 10,620 |
| 17. | Diamondville run of mine.. | 5.2 | 38.8 | 46.0 | 10.0 | 11,780 |
| 18. | Bear Creek run of mine.... | 9.7 | 37.1 | 43.3 | 9.9 | 10,700 |
| 19. | Diamondville run of mine.. | 6.4 | 39.5 | 44.2 | 9.9 | 11,550 |
| 20. | Roundup run of mine..... | 13.6 | 30.5 | 46.2 | 9.7 | 10,320 |
| 21. | Diamondville run of mine.. | 6.4 | 37.5 | 44.7 | 11.4 | 11,440 |
| 22. | Roundup run of mine..... | 13.3 | 31.2 | 45.3 | 10.2 | 10,420 |
| 23. | Diamondville run of mine.. | 5.8 | 36.9 | 47.6 | 9.7 | 11,780 |
| 24. | Roundup run of mine..... | 12.5 | 30.6 | 45.9 | 11.0 | 10,620 |
| 25. | Diamondville run of mine.. | 5.3 | 38.5 | 45.7 | 10.5 | 11,460 |
| 26. | Roundup run of mine..... | 13.0 | 31.8 | 45.1 | 10.1 | 10,300 |
| 27. | Diamondville run of mine.. | 9.1 | 36.2 | 45.0 | 9.7 | 11,820 |
| 28. | Bear Creek lump and Diamondville mine run..... | B.C. 9.7 | 35.1 | 45.8 | 9.4 | 11,010 |
| | | Dia. 6.7 | 37.1 | 44.1 | 12.1 | 11,390 |
| | | Total....7.6 | 36.5 | 44.6 | 11.3 | 11,270 |
| 29. | Diamondville..... | 6.7 | 37.1 | 44.1 | 12.1 | 11,390 |
| 30. | Diamondville lump..... | 6.4 | 39.2 | 46.4 | 8.0 | 12,310 |
| 31. | Roundup lump..... | A.....17.3 | 31.6 | 43.1 | 8.0 | 10,500 |
| | | B.....19.7 | 31.2 | 41.8 | 7.3 | 10,320 |
| | | C.....18.4 | 30.8 | 43.3 | 7.5 | 10,440 |

In most bituminous coal there is about 5 per cent. of hydrogen (on the moisture-free basis), which of course burns to water, and this water must be considered when interpreting reports on the heating value. Laboratory determinations of heating value are made with this water condensed to liquid; but in actual use it passes away as gas, and the error in the reported heat units amounts to about 500 B.t.u. per pound of dry coal.



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Manufacturers of illuminating gas find that the relative quantities of the different hydrocarbon gases and of the different tar-like substances volatilized from coal vary greatly at different temperatures with any particular coal, as well as from one coal to another: moderate temperatures yielding large quantities of the heavier products, while very high temperatures yield more of the lighter products. The lighter gases burn with a clear blue flame, while the heavier products burn with a more or less opaque yellow or white flame due to the incandescence of solid particles. All experience shows that this latter kind of flame is one of the chief essentials for successful reverberatory work, a very large part of the heat utilized by the charge being received by direct radiation from these highly incandescent particles. The trouble with a clear flame is that comparatively little of the hot gas comes in contact with the charge, to permit the direct absorption of heat; and not much of it comes in contact with the roof and walls, and so they receive comparatively little heat to radiate on to the charge; but the suspended incandescent particles have an immense surface per pound, from which they radiate a good deal of their heat to the charge. They also undoubtedly absorb some of the heat developed from the lighter gases and then radiate that to the charge also. With a good flame, the total surface of the incandescent particles in the furnace at any moment is far in excess of the entire surface of the roof and walls. The effect of this is like increasing the flow from a water reservoir, when a fire is to be extinguished, by multiplying the number of streams, and thereby greatly increasing the efficiency of action. This statement supplements the argument in regards to the "margin of temperature" on pp. 1847 and 1848. Two cases have already been cited in which a lack of incandescent flame was largely responsible for complete failure to do any smelting (Belt and Bear Creek coals). Another striking experience with this principle was had with the Brückner roasters at the old works, in which Belt coal gave very little flame at the fire-box end, and none at all at the other end even with forced draft. In order to get results it was necessary to drive the fire hard, which produced such a high temperature at the fire-box end that there was more or less trouble from sintering, while the charge never got hot enough at the other end to roast satisfactorily. With Diamondville coal this trouble did not exist. It should be noted that an *excessively* dense (smoky) flame is to be avoided, for two reasons: first, there is increased difficulty in supplying and mixing enough air for combustion; and second, it is almost certain that, in such a dense cloud, the heat radiated from most of the incandescent particles is obstructed and the charge not efficiently heated.

In order to get this luminous flame, even with good coal, a fairly thick fuel bed must be maintained on the grate; so that the condition is approximated in which just enough air comes through the grate to burn the fixed carbon to CO, while the rest of the air is admitted through adjustable

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checker holes above the bridge wall. The large hearth forms nearly the ideal combustion chamber demanded by these conditions. In boiler practice it is rather unusual to have more than an 8-in. fuel bed; but in large reverberatories with 1½-in. draft in the flue a 24-in. bed is more or less standard for good coal containing only a moderate percentage of fines. With coals that do not flame well under natural draft there is often a marked improvement by using forced draft. The latter so increases the temperature in the fire box as to drive off more volatile matter. This is illustrated by the experiences with Belt, Bear Creek, and Roundup coals, all of which did better smelting with forced than with natural draft.

*Percentage of Fines*

In ordinary boiler practice, the equipment can be so designed and used as to get reasonably good efficiency with any coal that is fit to be so called, although good coal is much more satisfactory than poor, and is likely to be more economical; but in reverberatory smelting there are much narrower limitations. The chemical properties have been indicated as the most important of these; but experience proves that the range of size is also very important. On account of conditions that the Anaconda Co. could not control, the quality of coal received from Diamondville became worse several years ago, the chief difficulty being that it was necessary to mine a soft coal, which crumbles badly and yields a large percentage of fines. Our standard furnace results, with only a moderate proportion of fines, consist in burning 60 tons of coal per 24 hr. and smelting 270 tons of charge (ratio 4.5); 80 to 85 per cent. of the charge being hot calcine, and the rest flue dust, which is harder to smelt than calcine. The poor coal dropped the results to 240 tons smelted with 60 tons of coal (ratio 4.0). The coal at this time contained only about 40 per cent. coarser than ¾ in. as against 60 to 75 per cent. coarser than this size when getting normal results. A test of Diamondville lump, which probably contained 80 per cent. or more coarser than ¾ in., smelted 328 tons of calcine and flue dust with 63 tons of coal (ratio 5.2). (Trial 30 in Table I). Comparing ratios, this lump coal yielded 30 per cent. better results than the poor mine run, and 15 per cent. better than good mine run. Another test, with one-third Bear Creek lump and two-thirds of the poor Diamondville mine run, smelted 248 tons with 62 tons of the mixed coal (ratio 4.0); but a parallel test with the Diamondville alone smelted 254 tons with 59 tons of coal (ratio 4.3). (See trials 28 and 29.) The best result in three days' test of Roundup lump was 263 tons of calcine (no flue dust) smelted with 79 tons of coal (ratio 3.3). (Trial 31 C.) Comparing the ratio and "efficiency" columns of trials 20 and 31 C, the Roundup lump gave 40 per cent. better results than Roundup run of mine under similar conditions.

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The three sets of results under trial 31 in the table show the need for the furnace men to learn by experience how best to handle each coal: there was a marked improvement each succeeding day, but we considered that the third day's result was the best that could be had with this coal. The shipment was not large enough for more than the three days' run; but in the other trials here recorded the preliminary experimenting was done before official testing was undertaken.

Neither the Bear Creek lump nor the Roundup lump is as economical as Diamondville run of mine. The Diamondville lump, however, is more economical; but, unfortunately, it cannot be supplied in sufficient quantity.

It may be stated that no coal larger than 7 or 8 in. ever goes into the fire box.

A coal sized anything like as closely as the regular commercial sizes of anthracite would probably give very poor results, because there would be such a free passage of air through the mass that the volatile matter would largely burn before it reached the hearth.

The reduced efficiency and increased cost resulting from a large percentage of fine coal are due to three considerations. First, the fresh supply of fine coal temporarily smothers the fire, thereby lessening the amount of air that can be drawn through; while the greater surface exposed per pound of fine than per pound of lump coal permits a sudden evolution of more volatile matter than can be burned by the available air supply, so that considerable fuel passes into the flue unconsumed; and to adjust the air supply through the checker work above the bridge is quite impracticable. Second, it is next to impossible to prevent a large loss of the fine coal through the grate. This loss at Anaconda is 25 to 30 per cent. of the coal when there is the excessive percentage of fines or an insufficient clinker bed, but only 12 or 15 per cent. under normally favorable conditions. Third, this dropping of fines is likely to leave large holes in the fuel bed for some time; but, as this condition does not occur till later than the excessive distillation of volatile matter, a good deal of unnecessary air enters and lowers the furnace temperature.

We are satisfied that the best solution of this question of fines is to pulverize all the coal and burn it by blowing it directly into the hearth of the furnace. Used in this way, all of the fixed carbon, and even the ash, as well as the heavy volatile hydrocarbons, supply incandescent surfaces for radiation of heat to the charge; there is absolutely no ash-pit loss; and the conditions are so uniform that the air supply can be adjusted to a nicety, with the assurance that there will be no decided shortage of air at one time and a considerable excess soon after. Moreover, there will be used only the minimum excess of air, so that a much greater "margin of temperature" will be attained. The extremely successful

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results of this method at the Canadian Copper Co.'s plant, together with the results at several steel plants, are so encouraging that the Anaconda Co. is preparing to give it a thorough trial. The conditions necessary to success are much better understood than they were a few years ago, and we expect that there will be no commercially insurmountable difficulties.

### *Moisture*

This is an impurity, which nobody wishes to pay for, and which requires heat to evaporate. Six per cent. of moisture in a pound of coal absorbs 120 B.t.u. in evaporating and being raised to the temperature of the reverberatory hearth; and in the case of Diamondville coal, burned in our large furnaces, reduces the temperature to 2,800° F. when it would otherwise be 2,825° F.; a reduction of the margin above the critical temperature that probably reduces the capacity and economy by *at least* 3 *per cent.*

### *Ash and Clinker*

Ash, like moisture, reduces the heat units per pound of coal, and yet it has a distinct value: with a great many coals the loss through the grate would be excessive if there was not a bed of clinker. In some cases, however, the ash clinkers so badly as to seriously clog the grate and interfere with combustion. This was the case with Diamondville coal when it was used under forced draft in 50-ft. furnaces at Anaconda. The average treatment of calcine, flue dust, and added lime rock, with forced draft, was 104 tons per 24 hr., while using 44 tons of coal (ratio 2.4); but with natural draft it was 144 tons of similar charge, using 46 tons of coal (ratio 3.1); an increase of 30 per cent. in efficiency. A little of this increase was due to certain changes in furnace design; but, as intimated, most of it resulted from the adoption of natural draft. The most important change in design was, in fact, a 58 per cent. increase in the grate area (from 53.3 sq. ft. to 84 sq. ft.) to suit the requirements of the coal under natural draft. With forced draft there was a tremendous volume of dense flame for some time after the grate was cleaned; so much, in fact, that a good deal of fuel passed away unburned. This excessive flaming was due to the high temperature in the fuel bed, caused by the intense forced-draft combustion; but this high temperature gradually covered the grate with a dense clinker, so that the passage of air was much reduced and there was practically no flame in the furnace. The blast had to be shut off every 4 hr. to clean the grate. An attempt to improve matters by introducing steam with the blast failed to help much, though with Belt coal steam was of marked benefit in reducing clinker troubles. A large percentage of ash prevents efficient contact of air with the fuel in the fire box, unless there is a very deep bed. When using Belt coal (25 to 30

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per cent. ash) with natural draft and only a 24-in. bed, it is altogether probable that there was a large excess of oxygen in the hearth (though we have no gas analyses to confirm it), for a good deal of the air passing through the bed would come in contact with very little except ash. There was not enough volatile combustible to utilize this extra air (the ratio of volatile matter to fixed carbon is low in this coal), which therefore acted to cool the hearth. With the deep-bed gas-producer conditions this excess air could not pass through. We have gas analyses to prove this.

The essence of the clinkering problem is, like that of the slag problem, the question of fusibility; but it is seldom possible to predict from analysis of the ash what will be the qualities or the clinker, or even whether it will or will not clinker. For example, there are decided variations in the four following analyses of Diamondville ash, yet the clinkering qualities are practically constant.

| $SiO_2$ | $Fe_2O_3$ | $Al_2O_3$ | CaO | MgO | Alkalies | $SO_3$ |
|---|---|---|---|---|---|---|
| Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| 37.5 | 6.8 | 23.6 | 16.6 | 2.0 | 6.3 | 6.0 |
| 50.3 | 7.4 | 25.4 | 9.4 | | | |
| 51.9 | 7.0 | 21.6 | 12.3 | | | |
| 60.0 | 9.6 | 19.6 | 7.4 | | | |

It often happens that there are great variations in clinkering properties of different portions of the ash; yet if one portion is easily fusible, at the temperature to which it is subjected after the fuel is largely burned away from it, it is quite certain to surround other portions, so that they contribute to the volume of clinker though they have not fused. The most satisfactory investigations of the clinker problem seem to be those made by E. G. Bailey.[8] He shows the present hopelessness of judging anything from analyses; but has adopted a laboratory fusion test of the average ash, and says:

"Even if the ash is made of a mixture of high and low fusing temperature material and if only that having a fusing temperature below that of the fuel bed is melted into a clinker, there will be a greater percentage of such ash fused into a clinker from a coal having a low average fusing temperature than from a coal having a high average fusing temperature. In actual practical tests it has been found that the percentage of ash which is formed into clinker as well as the obstructed grate area per pound of clinker hold a close relation to the fusing temperature of the ash from different coals when burned under similar conditions."

He shows plots of some results which indicate that, with good average conditions on steam-boiler grates, less than 25 per cent. of the total ash clinkers if its average fusion temperature is 2,500° F. or higher; but the proportion of clinker rapidly increases to 50 per cent. as the average fusion temperature falls to 2,300° F.

[8] Ohio Society of Mechanical, Electrical and Steam Engineers, Nov. 18, 1911; and *Bulletin No. 5*, Fuel Testing Co., Boston.

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## Description and Discussion of Coals Used

The Diamondville run of mine is decidedly the most economical of all the coals we have used, though it costs, delivered to the furnaces, from one-eighth to one-half more than any of the others except Rock Springs. This is because it is much the most efficient. It is a long-flame bituminous coal from which, in a properly designed furnace, with natural draft, the volatile combustible matter is distilled in just about the same length of time that the fixed carbon is burning in the fire box; keeping the entire furnace nicely filled with actively burning flame about half the time, and a very considerable amount of flame all the time. The distillation is rapid enough to produce a high temperature near the bridge, where the ore is charged, yet slow enough for the gases to give up a reasonable proportion of their heat before passing into the flue. The normal furnace results consist in smelting 270 tons of hot calcine and flue dust per 24 hr., while using 60 tons of this coal. The temperature of the gases averages 2,800° F. just beyond the bridge, while the formation temperature of the slag is about 2,000° F. The freezing temperature of the slag, after it is formed, is possibly not over 1,800° F., but it has an actual temperature of nearly 2,100° F. as it runs out of the flue end of the furnace. The gases enter the flue at an average of 2,200° F. These conditions guarantee good settlement of the matte, and easy skimming of the slag.

With good natural draft the Diamondville ash forms a clinker of moderate hardness, which is barred out without much trouble, and at the same time maintains an excellent fuel bed and keeps the total loss of fuel, through the grate, down to 12 or 15 per cent. when there is not an excessive amount of fines in the coal received. Most of this loss occurs at the time of grating. With forced draft, however, the clinker is so hard and tough, and forms in such large masses, as to require excessive labor in cleaning the grates. It is thus seen that, with natural draft, Diamondville approaches the ideal coal for smelting; and in fact it has the reputation of being one of the best coals available in the Rocky Mountain region for this purpose.

Belt is a bituminous coal, containing a good deal of carbonaceous shale and considerable iron pyrite, the 25 and 30 per cent. of ash in the mine run being a serious source of difficulty and inefficiency in its use. With moderate fire-box temperatures the ash does not clinker, but remains to a large extent in lumps; yet with the intense fire-box combustion accompanying forced draft, and sometimes with strong natural draft, it clinkers seriously unless steam is used beneath the grate. It ordinarily gives but very little good flame unless the temperature in the fuel bed is greatly increased, by means of forced draft, and even then the flame is far from ideal. Not only is the percentage of volatile matter low, but its ratio to fixed carbon is also low. When used in our regular fire box with

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natural draft it will do no smelting; and even with the immense gas-producing fire box the temperature a little beyond the bridge was only 2,500° to 2,600° as compared to 2,800° from Diamondville coal used in the regular fire box; but there were very large volumes of this lower-temperature gas, because of the great quantity of coal consumed.

Bear Creek is a sub-bituminous coal (between good bituminous and lignite). It flames sufficiently to smelt with natural draft on a normal-sized grate; but gives a better flame and more efficient smelting with forced draft. Under any conditions it flames, after fresh firing, only a quarter to a half as long a time as Diamondville, and so requires more frequent firing. With natural draft it produces a temperature just beyond the bridge 150° to 200° F. lower than Diamondville; and with forced draft it is 100° F. lower than Diamondville. It appears to deteriorate in heating value rather fast after mining, but does not slack seriously. It does not clinker much under either natural or forced draft; and, for that reason, is subject to a 25 per cent. fuel loss through the grate.

Roundup is also a sub-bituminous coal. It flames similarly to Bear Creek and gives about the same furnace temperatures. It slacks somewhat rapidly on exposure. Its ash clinkers somewhat more than Bear Creek, but not as much as Diamondville; and consequently suffers an intermediate fuel loss into the ash pit.

It seems reasonable to suppose that, after removing moisture, the higher the ratio of hydrogen to carbon the larger will be the proportion of total combustible that will be given off in the volatile form and serve to produce flame in the hearth; other conditions, of course, being the same. On the other hand, flaming is in all probability reduced the higher the ratio of oxygen to total combustible. If these two statements are true, then the relation between the first of these ratios and the second is one indicator as to the value of coals for smelting. The following figures are derived from average ultimate analyses of certain of the coals discussed in this paper. These analyses are taken from *Bulletin No.* 22 of the U. S. Bureau of Mines (samples from near the surface of mine workings being omitted); and the relation just mentioned is shown in the last column.[9]

| | 1 | 2 | 3 | 4 | 5 | Ratio of Column 4 to Column 5 |
|---|---|---|---|---|---|---|
| | H | C | O | $\frac{H}{C}$ | $\frac{O}{H+C}$ | |
| | Per Cent. | Per Cent. | Per Cent. | | | |
| Diamondville | 5.33 | 76.89 | 11.16 | 0.069 | 0.14 | 0.49 |
| Rock Springs | 4.96 | 74.28 | 13.72 | 0.067 | 0.17 | 0.39 |
| Bear Creek | 4.94 | 66.78 | 14.04 | 0.074 | 0.20 | 0.37 |
| Belt | 3.64 | 61.23 | 11.50 | 0.059 | 0.18 | 0.33 |

For these four coals the numbers in the last column stand in the same order as the normal smelting efficiencies that can be obtained with any

[9] The *Bulletin* gives no ultimate analyses of Roundup coal.

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ordinary provision short of the elaborate and expensive gas-producing fire box that was used for Belt.

### Other Important Factors

Of the factors other than fuel, the one that requires most careful attention is proper fluxing of the charges: and this means, not merely that a day's average sample shall have a suitable analysis, but that each individual charge shall have such composition as to yield a readily fusible and fluid slag, of low specific gravity; it also means that the speed of smelting depends on how intimately the various constituents of each charge are mixed. From the following facts it is clear that these conditions are not always attained. In December, 1903, Mr. Mathewson changed the practice of adding a small proportion of lime rock flux directly to the reverberatories, without any special mixing with the rest of the charge, to mixing it, with some regularity, with the concentrates and fine ore fed to the roasters. The result was to increase the reverberatory performance from 144 tons smelted with 46 tons of coal (ratio 3.1) to 163 tons smelted with 48 tons (ratio 3.4). The most striking example we have had of the effect of poor fluxing came when a charge of calcine failed to smelt in three times the usual 1 hr. and 20 min., though the entire contents of the hearth got so hot that the matte pool "boiled" (gave off sulphur) at the end of the 4 hr., and frothed considerable of the accumulated slag out of the furnace. It is, of course, understood that any approach to such poor results is not at all common.

#### *Preheating the Charge*

It is well known that the capacity and economy of a furnace are very much decreased if the hot calcine from the roasters is allowed to cool before smelting. At Argo, Colo., a 23 per cent. increase in capacity was obtained, with no increase of fuel consumption, by charging the hot calcine directly to the reverberatory instead of allowing an intermediate cooling. The calcine made up only one-half of the charge, the other half continuing to be cold ore.[10] At Anaconda the calcine enters the reverberatories at a temperature of 900° F.; and most of the flue dust is thoroughly hot.

#### *Preheating the Air*

Various attempts to save money by preheating the air for copper blast furnaces have been abandoned, though the fuel efficiency of the furnaces themselves was much increased when the air was raised to 700°

[10] Peters's *Modern Copper Smelting*, p. 446.

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or 800° F. There have been two reasons for the commercial failure: first, the cost of the extraneous fuel used to do the heating; second, the cost for much extra power to force the blast through the small and tortuous passages of the heating devices. Both of these troubles can be overcome for reverberatory work. The first step is to use the waste heat from the furnaces, instead of extra fuel. The gas comes away from these furnaces at a much higher temperature than from blast furnaces, and there is not the frequent inrush of large volumes of cold air such as occurs through the charge doors of the blast furnaces. In the few cases where preheated air has been used in reverberatories for copper smelting, natural draft and efficient heating devices have been utilized. The gas-fired furnaces at Great Falls have always had the checker work of the open-hearth steel furnace. The Peyton Chemical Works, near Martinez, Cal., heated the air for an oil-fired furnace in open flues, without checker work but with several turns purposely introduced. With hot air (varying from 750° to 1,830° F.) their furnace capacity was 50 per cent. greater than with cold air. It is easier and cheaper to remove dust from these open flues than from the checker work.[11]

### *Producing Steam with Waste Heat*

Since the practice of utilizing the flue gases to generate steam was adopted at Anaconda it has also been put into effect at several other large plants. At Anaconda about 25 per cent. of the heating value of the total coal is thus made effective; while only 15 per cent. of this value is actually utilized for smelting. This steam from the eight furnaces generates 30 per cent. of the 16,700 mechanical horse power used in the entire concentrating and smelting plant.

Boilers for this service must be arranged for the gases to pass through in a straight line, and with plenty of passage area, in order not to affect the draft. With the several changes of direction found in ordinary boiler practice the smelting capacity is seriously reduced; so much, in fact, as to affect the profits much more than the money value of all the steam.

For a good deal of information and suggestion, the writer is indebted to W. M. Kelly, general smelter foreman; F. W. C. Whyte, manager of coal mines; and especially to Edward O'Brien, general reverberatory foreman.

[11] Fred A. Leas: *Engineering and Mining Journal*, vol. lxxxvi, No. 19, p. 898 (Nov. 7, 1908).

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

. DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The International Lead Refining Plant

BY G. P. HULST, EAST CHICAGO, IND.

(Salt Lake Meeting, August, 1914)

THE Parkes process lead refinery of the International Lead Refining Co., at East Chicago, was built by the International Smelting & Refining Co. to treat the lead bullion produced by its Tooele plant, at Tooele, Utah. The plant was designed and constructed under the direction of the writer. Ground was broken Apr. 16, 1912, and exactly six months later the plant was in operation and lead was being cast. The capacity of the plant,

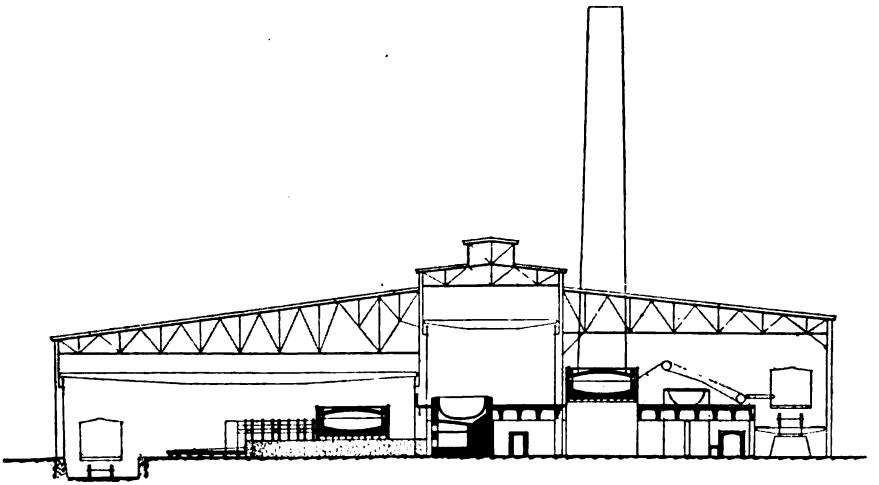

FIG. 1.—CROSS-SECTION THROUGH MAIN BATTERY.

running 25 days per month, is 5,000 tons. The four products of the plant are common lead, corroding lead, antimonial lead, and Doré bullion.

The site is an area, 64 acres in extent, lying north of 151st Street, between McCook Avenue and the Canal. The Indiana Harbor Belt Railroad, Baltimore & Ohio Chicago Terminal, and Pennsylvania Railroad enter the yard at the plant. There are five storage tracks and space for storing 100 cars in the yard. The main lead to the storage tracks passes over a 100-ton track scale on entering the yard.

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The plant is so arranged that all the principal operations are performed in one main building. This is a steel and concrete structure, 480 by 180 by 50 ft., whose various levels are shown in section in Fig. 1. Three broad-gauge railway tracks enter the building, and two 15-ton electric cranes, one 28 ft. and one 77 ft. span, travel the entire length. These cranes serve all departments and greatly facilitate the handling of material.

The main battery consists of the following apparatus:

| | Capacity Tons |
|---|---|
| Two sample kettles | 45 |
| Two liquating kettles | 60 |
| Two softening furnaces (inside dimensions, 13 ft. 6 in. by 28 ft. 2 in. by 31 ft. 2 in. deep) | 300 |
| Two desilverising kettles | 100 |
| One refining furnace | 300 |
| One molding furnace | 200 |

The arrangement is such that the lead flows by gravity from one piece of apparatus to the next and is finally hand-molded and loaded by trucks into cars.

For the treatment of by-products resulting from the main refining operations, the following equipment is provided:

| | Capacity |
|---|---|
| Three residue furnaces, each 8 by 16 ft. by 20 in. deep (inside measurement) | 30 tons each |
| Two circular blast furnaces, each 42 in. in diameter at tuyères by 14 ft. high, with five 3-in. tuyères | 40 tons each |
| Eight retort furnaces | 1,300 lb. each |
| Two tilting cupels, Rhodes type | 5 tons each |

Common lead is further refined to yield a product suitable for corroding by the Hulst improved crystallizing process. The equipment of this department consists of the following:

| | Capacity |
|---|---|
| Two crystallising kettles | 60 tons |
| Four heating kettles | 20 tons |
| One press. | |

Gases from the cupel, residue, and blast furnaces are conducted through brick and steel flues to a single bag house. The bag house is a building of brick and steel, 50 by 65 by 50 ft. The interior is divided into four separate chambers, each containing 144 woolen bags, 18 in. in diameter and 30 ft. long. The bags are shaken by an electric-driven automatic shaking device. The gases are delivered to the bag house by an 8-ft. American Blower Co. fan, driven by a 35-h.p. motor.

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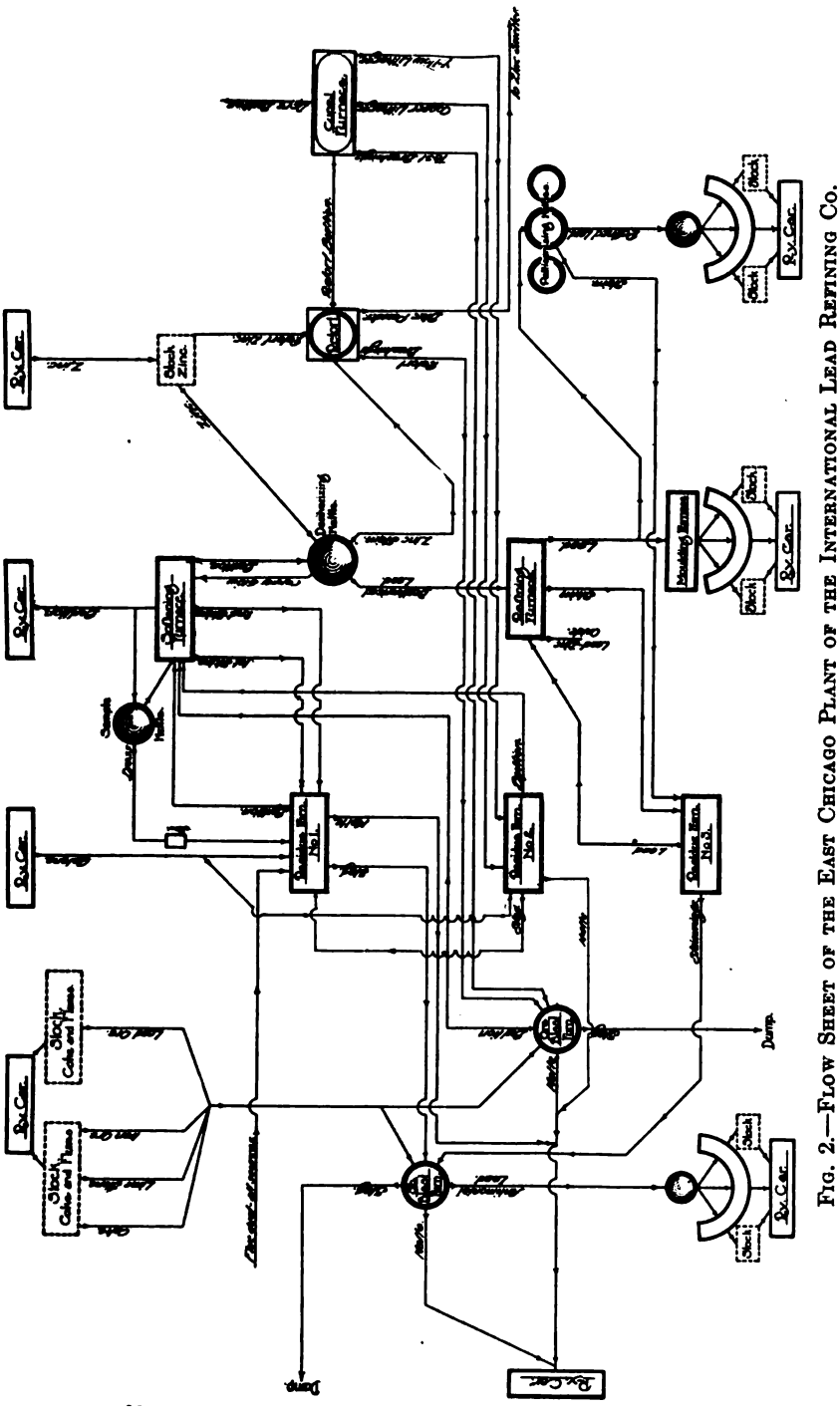
Casting Machine
Dumping Machine
Dumping Furnace
Dumping Furnace
Dumping Furnace

Steam, for use in the softening and refining furnaces, and for operating the air compressor, is furnished by two 90-h.p. Hawkes boilers. Electric power for all operations is purchased from the Northern Indiana Gas & Electric Co. Water is obtained from the city mains, which supply a 50,000-gal. tank elevated 50 ft. Waste water is returned to a 100,000-gal. sump tank and is pumped to the elevated tank.

The change house is a brick building, 35 by 85 ft. It is equipped with sanitary toilets, wash basins and lockers. One room is arranged as a lunch room for the men. The offices and laboratory are housed in a brick building 36 by 128 ft.

The accompanying flow sheet, Fig. 2, shows diagrammatically the handling of raw materials and by-products.

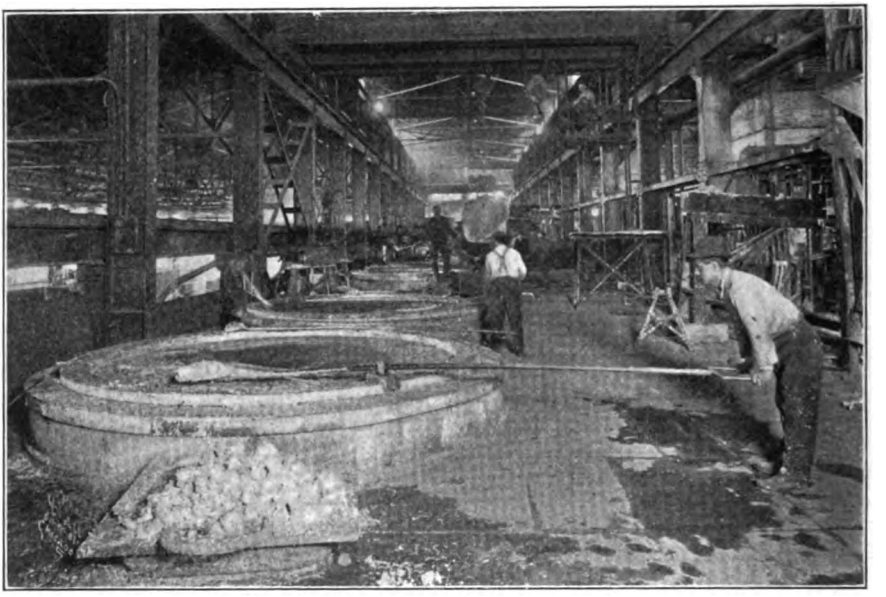

FIG. 3.—DESILVERIZING KETTLES.

Lead bullion from Tooele containing about 99.5 per cent. lead, 65 oz. silver, 0.4 oz. gold, and varying amounts of copper, antimony, arsenic, zinc, and bismuth, is received at the refinery in sealed cars and after being weighed is delivered into the softening furnaces by means of a steam-driven conveyor, constructed by Howe Scale Co. The sides and ends of these softening furnaces are water jacketed from the base plate to 3 in. above the slag line.

The products of the softening furnace are first skims, second skims, and softened bullion. The first two are sent to residue furnace No. 1. The bullion flows to the desilverizing kettles. The copper skimmings are charged into the softener.

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In the desilverizing kettle bullion is treated with zinc and skimmed, yielding zinc skims and desilverized lead. Fig. 3 is a view of the desilverizing kettles. In the kettle in the foreground the last skim is being taken and skimmings transferred to a large mold. In the background the crane is seen handling a large skim block.

The zinc skims go to the retorts and the desilverized lead to the refining furnace. The products of the refining furnace are lead-zinc oxide, refinery skim, and refined lead. The first product, depending on its composition, is treated in one of the residue furnaces or in the blast furnace. The skimmings go to residue furnace No. 3. The refined lead, in part, goes to the Pattisonizing kettles for further treatment, and the remainder passes to the molding furnace. Fig. 4 is a view of the molding

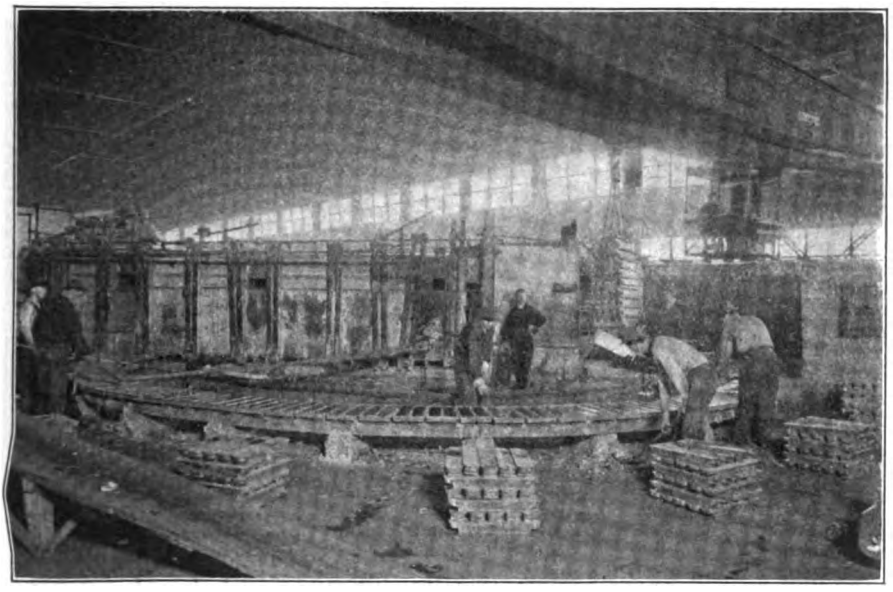

FIG. 4.—VIEW OF MOLDING FLOOR.

floor and shows the method of handling the bars of lead with the crane.

At the molding furnace the lead is siphoned into molds arranged in the arc of a circle, as shown. The bars are removed from the molds and stacked by hand and are placed in stock ready for shipment.

In the treatment of by-products, the zinc skim produced at the desilverizing kettles is treated in four tilting retort furnaces using oil as fuel. The products of this operation are retort zinc, retort breakings, blue powder, and retort bullion. The retort zinc is returned to stock to be used again at the desilverizing kettles. Retort breakings are sent to the ore blast furnace. Blue powder is shipped to zinc smelters for treatment.

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The retort bullion passes to two Rhodes type, cupel furnaces. These produce test breakings, copper litharge, yellow litharge, and Doré bullion.

The test breakings go to the ore blast furnace, the two litharge products go to residue furnace No. 2, and the Doré bullion is molded into anodes and shipped to the Raritan Copper Works for refining.

The Pattisonizing kettles, equipped for the Hulst crystallizing process, are shown in Fig. 5.

The lead received at the kettles from the refining furnace contains from 0.08 to 0.12 per cent. Bi. One crystallization reduces the bismuth from 0.08 to 0.05 per cent. and less. For lead containing 0.12 per cent.

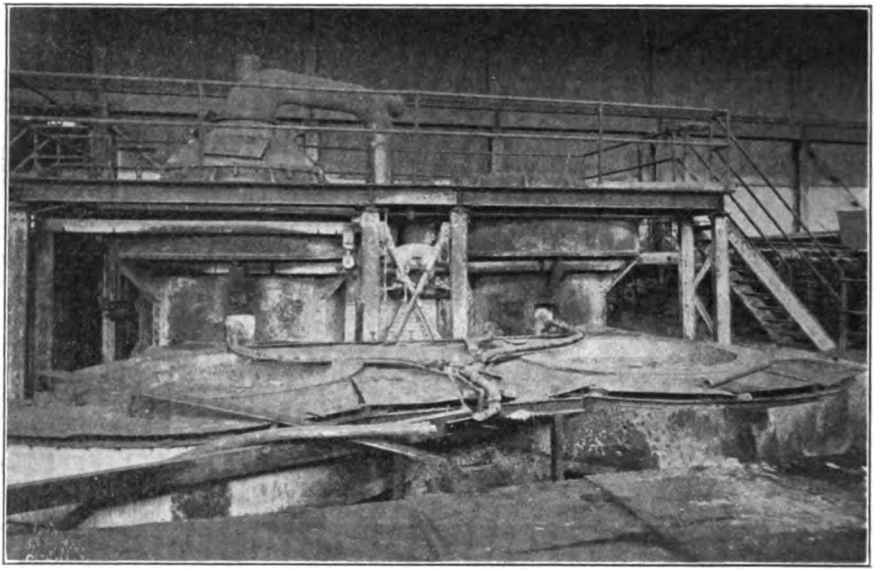

FIG. 5.—PATTISONIZING KETTLES.

Bi two crystallizations are necessary, if the crystals are drained by gravity; one is sufficient if crystals are pressed.

With such a low-grade product subsequent treatment for the recovery of bismuth is not profitable. This department will produce 150 tons of refined corroding lead per day, with a bismuth content of 0.05 per cent. or less.

Residue furnace No. 1 receives sample-kettle dross, softening-furnace skims, flue dust from all sources, and galena ore (80 per cent. Pb). The charge is weighed in over a small charging scale, and is so proportioned as to yield products of fairly constant composition. The products are bullion, refinery matte, and antimonial slag. The bullion is returned to the softening furnace. The matte is shipped to the smelter for treatment and the antimonial slag goes to the antimonial blast furnace.

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Residue furnace No. 2 receives only the litharge products from the cupel furnaces and galena. Its products are bullion, slag, and refinery matte. The bullion is returned to the softening furnace, the slag goes to the No. 1 residue, and the refinery matte is shipped to the smelter.

Residue furnace No. 3 treats the skimmings and dross from the refining furnace and from the Pattisonizing kettles. These yield lead and skimmings. The former goes to the refining furnace, the latter to the antimonial blast furnace.

The ore blast-furnace charge is made up of lead ores, coke and fluxes, and the following by-products: Retort breakings, test breakings, and slag from residue furnace No. 3. The three products are slag, matte, and bullion. The slag goes to the dump; the matte is shipped to the smelter, and the bullion is returned to the softening furnace.

The antimonial blast-furnace charge consists of ore, coke and fluxes, and, in addition, antimonial slag from residue furnace No. 1 and skimmings from residue furnace No. 3. The charge weighs 1,450 lb. The coke used is 12 per cent. of the weight of the charge. The products are slag, matte, and antimonial lead. If no ore is used the charge contains no sulphur and no matte or speiss is formed. The charge is carried low in the furnace under a light blast pressure (5 to 6 oz.). Arsenic is burned off and recovered in the flues and bag house.

The slag produced is sent to the dump. It has the following analysis:

| | Per Cent. |
|---|---|
| $SiO_2$ | 24.0 to 26.0 |
| $Al_2O_3$ | 10.0 |
| FeO | 36.0 to 38.0 |
| CaO | 10.0 to 12.0 |
| ZnO | 12.0 to 14.0 |
| Pb | 1.5 to 2.0 |

Matte, when produced, is shipped to the smelter.

The antimonial lead is run to a liquating kettle, from which it is cast into bars for shipment. Consumers of this product commonly specify that the lead should contain: Antimony, 15 to 18 per cent.; arsenic, less than 1 per cent.; copper, less than 0.5 per cent. This department has been remarkably successful in producing lead conforming to the specifications of the mixed-metal plants.

*Acknowledgments.*—A description of this plant would not be complete without an acknowledgment of the assistance rendered by R. Ruetschi, in the preparation of plans; C. W. Wilson, L. Crook, and F. E. Stolte, in the work of construction; and Herman Witteborg, in the operation of the plant.



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# Rope Idlers in the Raven Shaft

BY GEORGE A. PACKARD, BOSTON, MASS.

(Salt Lake Meeting, August, 1914)

THE shaft of the Raven mine, at Butte, Mont., is an incline 1,700 ft. in length and dipping at various angles. At the top the dip is 70° from the horizontal, but this is gradually flattened until at the 300-ft. level the inclination is only 47°. This angle continues to the 1,100-ft. level, below which it curves with a 125-ft. radius to 78°. In addition to these variations in dip, the shaft does not lie in the same vertical plane, with the result that the hoisting rope not only rubs at intervals on both the hanging- and foot-walls, but presses strongly against the west dividers near the collar, while 300 ft. below it runs close to the east end plates.

The early operators used no idlers, and wall plates cut 6 in. deep by the rope resulted. Later operators first attempted to overcome the excessive friction, and the wear of rope and wall plates, by introducing solid cast-iron idlers, 3 in. in diameter. To allow for the travel of the rope from side to side some of these had to be 3 ft. long and were very heavy. Judging from the appearance of the old idlers of this type found at the mine, they often failed to turn in the bearings, which is not surprising when it is considered that they would have to make 1,000 rev. a minute under ordinary hoisting conditions.

The next rolls were made of wood, 6 in. in diameter, with an iron band about each end, and a pintle of 1-in. round steel driven in at the ends to serve as a shaft. These wore rapidly, and were soon replaced by rolls made from water pipe, 5 or 6 in. in diameter, cut to the desired length and fitted with a wooden cylinder into which the pintle was driven. Where the idlers were used on the hanging-wall of the shaft the original bearing was simply a piece of $\frac{1}{2}$ by $1\frac{1}{2}$ in. strap iron, 10 in. long, turned up at the end in a circle of $1\frac{1}{8}$ in. in diameter to receive the shaft. A small hole served for oiling, and common black oil was generally used, although filtered oil from engine bearings and compressor bearings was also used. When the rolls were to be placed on the foot-wall the bearings were made from two pieces of 1 by 3 in. steel, 6 in. long. A half circle was cut on the flat side of each piece and the two half circles together formed a bearing. Oil holes were provided, and in some cases holes were bored through the two pieces so that they could be screwed or spiked to the wall plates.

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The later practice was to forge the bearings from 1 by 3 in. steel, and to drill two holes at each end for ¾-in. lag screws, by which the bearings were fastened to the timbers. These bearings were finally used on both foot- and hanging-wall. Similar idlers were so placed as to protect the dividers and end plates. The difficulty of proper oiling presented the greatest obstacle to satisfactory results from this type of idler. As the clearance between the skip and the hanging-wall plates was sometimes less than an inch, there was not room for large oil or grease cups. In addition, the bearings were liable to get full of grit, especially when wet ore was being hoisted. Grease cups were generally unsatisfactory, although several kinds of grease were tried and especial attention was paid to having that

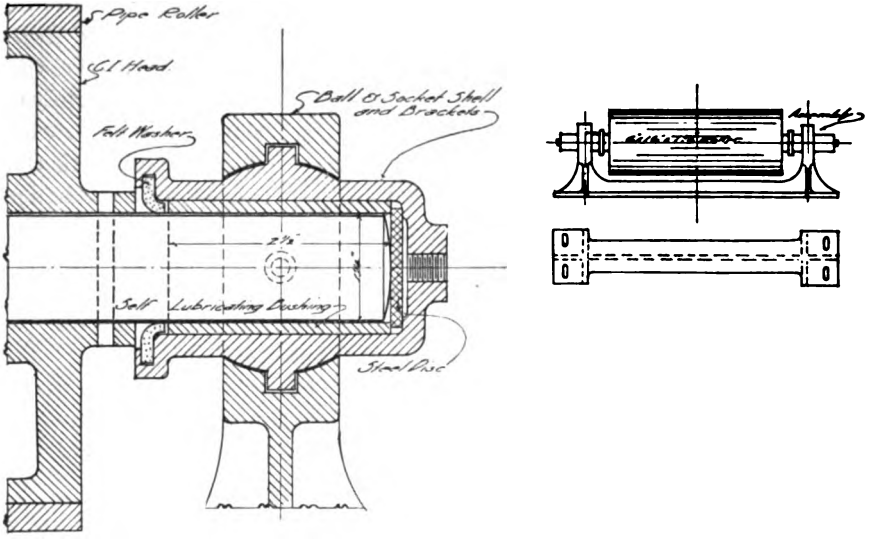


FIG. 1.—ROPE IDLER USED IN THE RAVEN MINE, BUTTE, MONT.

which was suited to the temperature of the shaft. In any event it was necessary that the rolls be examined and the oil cups filled every two days, which meant the cessation of hoisting for 2 hr. The bearings wore rapidly and the rollers tended to get out of line. The full skip weighed over 3 tons, and where the shaft flattened near the surface the pressure against the idlers was heavy. It was only by distributing this weight over idlers placed but 5 ft. apart that anything approaching satisfactory service could be obtained at this point.

To obviate the necessity for so much attention, an idler was devised, following my suggestions, by the Robins Conveying Belt Co. This idler is shown in Fig. 1. The roller is extra heavy 6-in. pipe, ⅜ in. thick, 20 in. long, in each end of which is pressed a cast-iron head, and through which passes a 1¼-in. steel shaft. This turns in a self-lubricating

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bearing carried by a bracket in a ball-and-socket shell, which prevents cramping. As a preliminary to adopting these bearings two types of graphite and bronze self-lubricating bushings were tried side by side in the incline for three months. The one proving most satisfactory had cylindrical bodies of graphite ¼ in. in diameter set in the bronze, or "metalline," bushing at about ¾-in. centers. One end of the bearing is entirely closed, the end thrust being taken by a steel disk, which also serves for forcing out the bushing when it is worn. The other end of the bearing is protected from grit by a felt washer. This, however, also retains the fine particles of metal and graphite, and in time this gummy matter causes the bearings to bind. Occasional cleaning of the bushings with kerosene obviates this trouble. The cap is hinged at one side and fastened at the other with a hinged bolt so the roller and bushing can be easily removed. The bearings can be turned through 90°, and the roll turned end for end, permitting the advantage of full wear. The whole is carried in a casting which is fastened to the wall plates with ¾-in. lag screws. These idlers have been in use nearly a year and are very satisfactory. While with rapid and continuous hoisting the bearings become quite hot, they do not bind if they are cleaned occasionally.

For the lower part of the shaft, where the rope runs true and the inclination is 78°, the idlers are merely common sheave wheels, cast solid and keyed on a shaft of 1-in. cold-rolled steel. This has a total length of 13 in. These sheaves are 9 in. in diameter with a 3-in. face, having a groove 1½ in. deep, 1 in. wide at the bottom. The bearings are maple blocks 4 by 4 by 6 in., bored to receive the shaft, and provided with an oil hole. These are fastened to the wall plates with six spikes.

The wood-filled pipe idlers with forged bearings cost at Butte about $8 each, including bearings. The idler with the self-lubricating bearings costs $15, but the difference is quickly saved in decreased cost of attention. The solid cast sheaves weigh 26 lb. They cost, when fitted with a shaft, but excluding the maple bearings, $2.75 each.

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# Separation of Lead, Zinc, and Antimony Oxides

BY RICHARD D. DIVINE, SOUTH CHICAGO, ILL.

(Salt Lake Meeting, August, 1914)

In the Parkes process of extracting precious metals from lead, zinc is added to the molten lead containing gold, silver, copper, and some antimony. These metals, with the exception of antimony, form an alloy with the zinc by reason of their greater affinity for it than for the lead, and this alloy, being lighter than lead, rises to the top, carrying the precious metals with it, whence it can be removed by skimming. These skimmings are further treated for the separation of the zinc therefrom, and the recovery of the precious metals, the zinc being driven off by heat and again used for another operation. In addition to the zinc that combines with the silver, etc., a certain amount is necessary to saturate the lead, and it is therefore necessary to treat the desilverized lead to free it from this residual zinc.

To do this it is heated again in a reverberatory furnace and air or steam is blown through it. This oxidizes the zinc together with a small amount of lead and antimony, leaving the lead free from zinc and antimony, and forming oxides of these metals, which rise to the top and are skimmed off.

This process relates to the separation of the metals in these oxides or skimmings.

The skimmings are cooled, crushed, and mixed with carbonate of soda and pulverized oil coke. The mixture is heated in a reverberatory furnace on a bath of lead, to about 1,000°C. The metals are reduced by the coke, the lead and antimony go into the lead bath, while the zinc is volatilized and burned to oxide.

This impure oxide is at the present time treated at Omaha by the Hall process of electrical precipitation, making a very pure metallic zinc. It can also be very easily refined by the old Schnabel process of using ammonia and carbon dioxide as a solvent. It seems to be in an ideal form for this purpose, and very few changes from the original method are required to obtain the zinc oxide practically C. P., leaving the lead and antimony as residues.

There has been considerable difference of opinion regarding the effect

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of the carbonate of soda in this process. That it is of material assistance is shown by the following experiments: Skimmings mixed with carbon alone and heated gave an extraction of about 10 per cent. of the zinc, while extractions as high as 92 per cent. were obtained when carbonate of soda was added to the mixture. Various explanations of this action have been suggested, the most probable one being that the extreme fusibility of the soda salt causes it to act as a protective coating, permitting oxidation by the air in the furnace; it also gives sufficient mobility to the mass to allow of good contact between the carbon and oxides.

The skimmings from the refining furnaces vary somewhat in zinc contents, due to different methods of working at the various plants.

At the National plant of the American Smelting & Refining Co., the zinc will average from 12 to 14 per cent., while at Omaha it is somewhat lower. In the first experiments the crushed skimmings were treated in the furnace without a bath of lead, the reduced lead forming the bath. The analysis of this lead showed it to contain 8.70 per cent. antimony, 91.25 per cent. lead, and less than 0.05 per cent. zinc. The soda skim resulting from these operations amounted to about 15 per cent. of the original charge. The resulting zinc collected in bags gave an extraction of between 80 per cent. and 85 per cent. of the zinc contents, and the analysis showed it to contain: Zn, 60 to 65; Pb, 12 to 18; Sb, 1.00 per cent. It was a light, fluffy, yellowish product, which when first collected contained some $CO_2$.

The present practice of treating the skimmings is as follows: After the furnace has been skimmed, the skimmings are cooled and put through a crusher set to about ½ in. The material is then mixed with finely crushed oil coke and sodium carbonate. The charge is made up approximately of 1 ton skim, 175 lb. soda ash, and 400 lb. coke. It is charged into a reverberatory furnace on a bath of molten lead. After the charge is started it is necessary to maintain the heat at the same point, as any cooling will stop the reaction, and to start it again requires much time and trouble. For this reason it has deen found advisable to use oil as a fuel in place of coal.

It would undoubtedly be possible to use producer gas for fuel, or possibly powdered coal. The success of the process depends upon the maintenance of a steady, uniform heat. The charge is stirred or puddled at regular intervals, only one working door being opened at a time to avoid cooling the furnace unnecessarily. After the zinc has burned off, which can easily be determined by the absence of any bluish flame, the oil is shut off and the furnace immediately skimmed. The resulting "soda skim" amounts to about 10 to 15 per cent. of the original charge. The furnace is then ready to be recharged. The antimonial lead is tapped from a lead well as often as necessary and taken to a refining furnace for the removal of the antimony.

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The analysis of the "soda skim" shows it to contain a somewhat higher percentage of zinc than the original skim, but we hope in time to improve on this. It was thought that it might be possible to save the expense of crushing and charging the skims, by simply adding the mixture of soda and carbon to the molten skim in the refining furnace itself, and although the percentage of extraction was good it was found that the repairs to the furnace were excessive, the lining being very quickly eaten out. The opposite is true: when the skim is charged cold the repairs to the furnace are very small.

Heretofore the zinc oxide in the skimmings has been lost, as there has been no way of separating it from the oxides of lead and antimony and saving it. The presence of zinc has also been a nuisance in clogging the smelting furnace by forming deposits on the sides, which retain other metals, fluxes, etc., and gradually make the furnace smaller and curtail the output.

At the National plant we have obtained a higher extraction of the zinc by skimming in two stages. Air is blown through the molten lead until the zinc is all oxidized and rises to the top. It is then skimmed off and kept by itself. The temperature of the furnace is then raised, air is blown through the bath again, and a second skim is removed, consisting largely of antimony with very little zinc. This is sent direct to the antimonial furnace to be treated.

The first skim requires less soda and carbon per ton treated, works better in the furnace, the zinc extraction is higher, and the soda skim is less.

We found that by making two skims we were able to raise the percentage of zinc from 12 to 14 per cent. to about 17 or 18 per cent. The antimony is oxidized after the zinc and most of it remains with the lead after the first skimming is taken off. The second, or antimonial, skim contains but a very small percentage of the original zinc contents, the loss of zinc being so small it is good practice to follow this method.

It requires about 24 hr. to work a charge of 18 tons. It is necessary to shake the collecting bags after each charge; unless this is done they become clogged, causing back pressure, which slows the operation and injuriously affects both recovery and costs.

A report of one of the early operations at the National plant will give an idea of the work accomplished. I may state, however, that the work at the present day is much better.

| Total Charge | Pounds |
|---|---|
| Skim | 20,000 |
| Coke | 3,500 |
| Soda | 1,750 |

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| Assay Skim | Contents Pounds Zn | Pounds Pb + Sb |
|---|---|---|
| Zn, 16.50; Pb, 63.50; Sb, 6.00 per cent. | 3,300 | 13,700 + 1,200 |
| **Produced** | | 14,990 |
| Antimonial lead | | 13,360 |
| ZnO fume | 3,825 | |
| **Assay** | | |
| Zn, 65; Pb, 14.65; Sb, 1.00 per cent. | 2,486 | 598 |
| Residue (Soda Skim), 3,320 lb. | | |
| Zn, 24.50; Pb + Sb, 9.8 per cent. | 813 | 325 |
| | 3,299 | 14,283 |

Some lead of course remained in the furnace.

While this process has not as yet been tried on a large scale with complex lead-zinc ores, I can see no reason why it should not prove successful in that field.

The additional cost of roasting would not be a serious matter. Most of the values would be taken up by the lead bath, and there would be only the residue or "soda skim" (amounting to from 10 to 15 per cent. of the original weight) to be sent to the blast furnace.

Experiments made in a small way gave returns that indicate a successful solution of the problem. The silica and sulphur in the roasted product do not seem to affect the zinc recovery. The silica would interfere only so far as it increased the amount of soda skim to be treated.

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# Leaching Experiments on the Ajo Ores

BY STUART CROASDALE, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

NOT long ago I was called upon to conduct some experiments on the treatment of ores from the New Cornelia copper mine, Ajo mountains, Arizona, for the Calumet & Arizona Copper Co. The problem was a very interesting one and contained some unusual features. Incidental with this problem was the utilization of a large amount of low-grade pyrite in the Bisbee mines of the company, provided this could be accomplished by any feasible method.

The New Cornelia mine is situated 45 miles south of Gila Bend, the nearest railroad point, on the Southern Pacific Railroad. It is about 160 miles, almost due west, from Tucson. This section is one of the most arid in the United States. The normal annual precipitation, as given by the United States Weather Bureau covering a period from 1870 to 1901, is 3 to 5 in. During June and July, 1913, the records kept by the company gave the following results. These are all I have available, but they show the enormous evaporation that can be expected, which may be a factor in leaching operations on a large scale.

| | Temperature, Degrees F. | | |
|---|---|---|---|
| | High | Low | Average |
| June | 112 | 62 | 84.0 |
| July | 115 | 71 | 87.8 |

| Evaporation | June | July |
|---|---|---|
| Concrete tank, inches | 11.7 | 10.72 |
| Iron tank, inches | 14.8 | 13.07 |
| Rainfall, inches | None | 0.77 |
| Cloudy days | None | 10.00 |

Aside from several springs about 10 miles from the property, the nearest known water in quantity at the time these experiments were made was the Gila river at Gila Bend, 45 miles distant. Part of the development of the property of course included drilling for water in the valley, which was in progress at the conclusion of my experiments.

The ore deposit itself is unusual. It consists of an intrusive plug of

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monzonite or granite porphyry. Chalcopyrite, bornite, and magnetite are finely disseminated throughout the magma in sufficient quantities to make the greater part of the mass commercial ore to a certain depth. Erosion has removed the surrounding country rock and left the plug sticking out at the base of the mountains as two or three cone-shaped buttes, without overburden of any description.

Oxidation has taken place *in situ*. Owing to the aridity of the country, there has been no leaching and no secondary enrichment. Instead of following the contour of the surface, as might be expected, the drill records (in the mine model) show oxidation to have taken place quite uniformly downward throughout the mass and the division between the oxidized and sulphide ores is almost a horizontal plane. The transition zone, or partly oxidized zone, between the oxidized and sulphide ores is likewise shown to be quite narrow—much narrower than would be expected.

The oxidized ores amount to about one-third of the total ore developed. The greater part of the copper exists as carbonate, although there is some silicate and some oxides, including cuprite. There is also a small percentage of sulphide, as residual chalcopyrite and intermediate forms.

The oxidized iron exists in both ferrous and ferric condition, both soluble in acid.

While the chalcopyrite is quite generally disseminated throughout the magma, there seems to have been a tendency for it to segregate in some portions, thus producing a higher-grade ore. The average grade of ore used for experimental purposes was placed between 1.5 and 2.0 per cent. copper. What the ultimate average grade will be will depend upon the cost of treatment.

## Methods of Treatment

From the foregoing it will be readily seen that the conditions of the problem demand simplicity of process and large scale of operations. Of course a railroad will be built to the mine, but whether the ore will be brought to water or water will be brought to the ore are yet factors to be considered and may have their bearing to a certain extent on the method of treatment adopted, when freight and fuel costs are estimated.

Methods of leaching with solutions of reducible salts, like ferric sulphate, ferric chloride, and cupric chloride, have some advantage in their solvent power on cuprite and chalcocite or other intermediate sulphides, but have little action on chalcopyrite. These salts are expensive and their value as commercial solvents depends upon their regeneration by some cheap method of oxidation after the dissolved copper has been precipitated. This difficulty has not yet been overcome and

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the results from attempts to use these methods have been disappointing even in the hands of skilled metallurgists.

The same may be said of sulphurous acid and ammonia processes.

Two methods of treatment, however, are available, and both are entirely reliable and feasible under the conditions involved. One of these is the old Henderson process of roasting with salt, and the other is leaching the raw ore with sulphuric acid. Under the head of "Chloridizing Roast with Salt," I include the so-called sulphatizing roast of Mr. Wedge, in which he finds the addition of a small percentage of salt very beneficial; and also the modification used by Mr. Laist at Anaconda, in which he reduces the consumption of salt, as well as the losses by volatilization, by giving the ore a preliminary roast before adding the salt.[1]

A comparison of these two methods may be made briefly as follows:

By using the chloridizing roast, all ores on the property could be treated by the same process—oxidized, sulphide, and intermediate. This would mean the construction of only one plant. The average extraction of the copper would probably be higher than that obtained from acid leaching and the small amount of silver in the ore would be recovered to a large extent. By mixing the ores, little if any sulphur would have to be added as pyrite. The only chemicals required would be salt, to the extent of 5 to 10 per cent. of the ore treated, and possibly a small amount of pyritic ore. Both of these could be transported in ordinary freight cars. There would be less corrosive solutions to handle. There would be a minimum amount of iron and alumina passing into solution. There would be no appreciable absorption of copper by slimes.

On the other hand, this method would require dry crushing of the ore to 20 or 40 mesh, which would be troublesome if not prohibitive on the oxidized ores on account of the difficulty in controlling the poisonous dust, in addition to the expense of grinding. It would involve a roasting cost. It would mean a more expensive plant, and a more extensive plant for the same capacity, due to the slower percolation of the fine material, which would require a much larger vat area of less depth to hold the tonnage required. It would mean a supply of cheap salt. A chemical precipitant would probably have to be used to recover the copper.

By leaching with sulphuric acid, the oxidized ore can be treated raw and need not be crushed finer than ½ in. or 2-mesh size, or perhaps coarser, as the experiments will show. Owing to the high oxidation of the surface ores and apparently small zone of partly oxidized ores, a large tonnage can be expected to give a comparatively high extraction by this method. Sulphuric acid can be made as a by-product at the roasting

[1] Stuart Croasdale: *Engineering and Mining Journal*, vol. lxxvi, No. 9, pp. 312 to 314 (Aug. 29, 1903); *Bulletin* No. 83, November, 1913, p. 2709; *Mining Magazine*, vol. x, No. 3, pp. 200 to 204 (March, 1914).

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plant of the Calumet & Arizona Co. at Douglas, or if the calcines are converted into sponge iron and used as a precipitant for copper, the low-grade pyrite from Bisbee can be cheaply transported and acid made at the mine. If electrolytic precipitation can be used, a small percentage of the acid consumed can be regenerated. This method of extraction will permit of less expensive plant construction for the oxidized ores.

On the other hand, sulphuric acid alone will extract none of the copper existing as sulphide, only half of the copper existing as cuprite, and may be indifferent about the silicate of copper. It will extract none of the precious metals. Owing to the character of the oxidation of the ore, already mentioned, considerable iron and alumina pass into solution as readily as the copper, which will seriously interfere with any form of electrolytic precipitation and render the amount of acid regenerated a doubtful asset when compared with the amount consumed. As an offset to this objection, some of the iron is in a ferric condition, which assists materially in the solution of the minerals mentioned above.

If acid is made at Douglas, it will have to be transported several hundred miles in tank cars, which will have to be returned empty. If acid is made at the mine from low-grade pyrite, a gain will be made in transportation, but the cost of an acid plant will be added to the cost of the leaching plant and the calcines will have to be utilized. Almost an entirely new plant will have to be constructed for treatment of the intermediate and sulphide ores.

The problem is clearly one in arithmetic as well as metallurgy, so laboratory experiments were made on both methods.

## Laboratory Experiments

The ore furnished me for these experiments came from the earlier development of the property and the oxidized ores were from very near the surface, but were less kaolinized than those received later for the larger experiments.

The analyses of the ores were as follows:

| | Oxidized Per Cent. | Sulphide Per Cent. |
|---|---|---|
| $SiO_2$ | 66.23 | 66.08 |
| Fe | 6.40 | 4.50 |
| $Al_2O_3$ | 13.75 | 11.35 |
| CaO, total | 0.56 | 0.55 |
| CaO sol. in dilute acid | 0.31 | 0.15 |
| S, total | 0.19 | 2.09 |
| S as sulphate | 0.03 | 0.03 |
| Zn | 0.10 | 0.10 |
| Cu, total | 2.03 | 2.75 |
| Cu sol. in 10 per cent. $H_2SO_4$ | 1.84 | 0.22 |
| Au, oz. per ton | 0.01 | 0.01 |
| Ag, oz. per ton | 0.22 | 0.18 |

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On a 2 per cent. ore, a variation of 0.01 per cent. in the percentage of copper in the tailings makes a difference of 0.5 per cent. in the percentage of extraction. This must be kept in mind when comparing the results of these experiments, for several hundredths of 1 per cent. may be within the limits of error in sampling and chemical analyses.

### *Chloridizing Roast with Salt*

*Oxidized Ores.*—The ore was crushed to 20 mesh. The experiments were conducted in an assay furnace at a low red heat or at estimated temperatures of 1,000° to 1,300° F. No attempt was made to condense the volatile copper chloride fumes.

The roasted ore was leached 24 hr. with a 1 per cent. sulphuric acid solution which was supposed to correspond to the solutions from the condensing towers at the Pennsylvania Salt Works, Philadelphia, where this method of copper extraction has been in use for a long time.

In a Wedge, or other multiple-hearth furnace, a particle of ore remains in the roasting or chloridizing atmosphere several hours. My first experiments were on variation in time of roasting, which ranged from 1 to 6 hr.

Sulphur was added, in the form of pyrite, in slight excess of that necessary to combine with the copper as CuS, or 1.2 per cent; 10 per cent. salt was added, which is twice the amount necessary to form the normal chloride of copper, but this might be of some advantage, as will be mentioned later.

At the end of 1 hr., 11.8 per cent. of the copper had volatilized and 65.8 per cent. had been rendered soluble in the roasted ore, which gave a total extraction of 77.6 per cent.

At the end of 6 hr., 69.0 per cent. of the copper had volatilized and 7.3 per cent. remained soluble in the roasted ore, which gave a total extraction of 76.3 per cent.

The tests, at 1-hr. intervals between these extremes, showed a gradual increase in the percentage of copper volatilized, with substantially the same total extraction.

In the next series of experiments the sulphur was varied from 1.2 to 2.5 per cent. The salt was kept constant at 10 per cent. and the time of roasting in each case was 1 hr.

With 1.5 per cent. sulphur the results were 15.5 per cent. volatilization of the copper, and 70.3 per cent. soluble copper in the roasted ore, or a total extraction of 85.8 per cent.; 2.1 per cent. sulphur gave 21.1 per cent. volatilization and 64.7 per cent. soluble copper, or a total extraction of 85.8 per cent. Both higher and lower sulphur contents gave lower total extractions.

In the third series of experiments the sulphur was kept constant at

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1.5 per cent. and the time of roasting at 1 hr. The salt varied from 5 to 15 per cent.

The lowest volatilization and highest total extraction were obtained with 10 per cent. salt, which, as stated above, were 15.5 and 85.8 per cent., respectively.

The result with 5 per cent. salt was a volatilization of 46.2 per cent. and a total extraction of 71 per cent. This amount of salt is near the theoretical quantity necessary to form the normal chloride of copper. My previous work on the volatilization of metals as chlorides proved that, other conditions being equal, volatilization greatly increases as the relations between the metal, salt, and sulphur approach theoretical proportions.

The use of 15 per cent. salt gave 23.2 per cent. volatilization and a total extraction of 80 per cent. Other proportions of salt gave intermediate results.

The foregoing experiments gave the best results when the ore, containing a little more than 2 per cent. copper, was roasted 1 hr. with 10 per cent. salt and 1.5 per cent. sulphur.

As a final test, these factors were all raised as follows: salt, 12.5 per cent.; sulphur, 2 per cent.; time, 1.5 hr. The results were, 24.2 per cent. of the copper volatilized and 62.3 per cent. soluble, or a total extraction of 86.5 per cent., which is substantially the same as before. The tailings from this test were leached an additional 24 hr. with a 5 per cent. sulphuric acid solution containing 7 per cent. salt, which increased the extraction to 90 per cent. In all probability a total of 48 hr. leaching with the weaker acid would have accomplished the same result.

*Sulphide Ores.*—The average analysis of these ores has been given. The copper existed as chalcopyrite. The experiments were conducted in a manner similar to those on the oxidized ores. The ore was crushed to 20 mesh. No sulphur was added. The percentage of salt ranged from 10 to 15 per cent. and, the ore being higher grade, the time of roasting was extended to 1.5 and 2 hr., although this was probably unnecessary. Owing to difficulty in controlling the furnace, the temperatures used were higher than they should have been.

The results were:

| | 1<br>Per Cent. | 2<br>Per Cent. | 3<br>Per Cent. |
|---|---|---|---|
| Salt used | 10.0 | 12.5 | 15.0 |
| Copper volatilized | 68.8 | 44.1 | 56.0 |
| Soluble copper | 20.0 | 47.9 | 37.2 |
| Total extraction | 88.8 | 92.0 | 93.2 |

Leaching these tailings an extra 24 hr. with 5 per cent. sulphuric acid solution containing 5 per cent. salt, extracted no more copper. There was no difficulty in extracting all the soluble copper in the roasted ore with a 1 per cent. acid solution.

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While the temperatures, as stated above, were higher than they should have been, these results confirm those made a number of years ago, when I found that copper is more readily volatilized from chalcopyrite by the chloridizing roast than from any other mineral.

*Mixed Oxidized and Sulphide Ores.*—The following experiments were conducted by the Wedge Mechanical Furnace Co., at Philadelphia, to determine the merits of a sulphating roast. Their report is as follows:

Test No. 1. The oxidized ore was ground to 20 mesh and the sulphide to 40 mesh. Two parts of sulphide were mixed with one of the oxide and the mixture was roasted 3.5 hr. at temperatures increasing from 550° to 1,180° F. The roasted ore was leached with 2° B. sulphuric acid solution. The extraction was 85.5 per cent.

Test No. 2. The same mixture of ores was roasted 3 hr., with 5 per cent. salt added, at temperatures ranging from 400° to 990° F. The roasted ore was leached with 2° B. hydrochloric acid solution, which gave an extraction of 94.4 per cent.

Test No. 3. The same ore mixture was used as before, and 5 per cent. salt was added. The mixture was roasted 2.5 hr. at temperatures from 500° to 860° F. The acid wash was 2° B. hydrochloric acid. The extraction was 96.0 per cent. of the copper.

Test No. 4. The preceding test was repeated with the addition of 2 per cent. pyrite (containing 47 per cent. sulphur) to the mixture. The ore was roasted 2.5 hr. at 600° to 900° F. and the roasted ore leached with 3° B. hydrochloric acid. The extraction was 93 per cent.

Test No. 5. Equal parts of oxidized and sulphide ore were used and 5 per cent. salt added. Other conditions being the same, 94 per cent. extraction was obtained.

Test No. 6. Both ores were ground to 40 mesh and mixed in equal parts with the addition of 5 per cent. salt and 2 per cent. pyrite. The extraction was 94 per cent.

Test No. 7. Test No. 6 was repeated, except that 3 per cent. pyrite was added instead of 2 per cent. The mixture was roasted 2 hr. at temperatures from 400° to 900° F. The extraction was 98 per cent. of the copper.

*Summary of Results.*—These experiments are sufficient to show that an extraction of 90 to 95 per cent. of the copper can be expected by a chloridizing roast on all the ores whether mixed or treated separately, but it would be preferable to mix them in order to secure the advantage of an excess of sulphur in the sulphide ores. If they are not mixed, from 1.5 to 2 per cent. sulphur will have to be added to the oxidized ores in the form of pyrite.

The ores will have to be crushed to 20 mesh, or finer.

The salt required will be 5 to 10 per cent.



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The temperature of raosting should not be above 900° F. to avoid excessive volatilization.

Large furnaces, where air supply and temperatures are under better control and where the benefits of mass action can be obtained, should give as good if not better results than the laboratory. At low temperatures, the slight volatilization losses are easily recovered by scrubbing towers, together with sufficient acid for washing the ore.

### *Leaching Experiments*

These were made only on the raw oxidized ores. The crushing varied from 8 to 2 mesh sizes. The experiments were conducted in glass and stoneware jars provided with small air-lift pumps for circulating the lixiviants. Both standing and circulating lixiviants were tested. The strength of acid solution varied from 3 per cent., or just enough to dissolve the copper in the ore, up to a 10 per cent. solution. Sulphuric acid was used. All references to this acid mean 100 per cent. $H_2SO_4$ and not commercial acid. Only enough lixiviant was used in each experiment to cover the ore.

The results were as follows on 24-hr. treatment:

*8-Mesh Ore*

| | Standing Lixiviant | | Circulating Lixiviant | |
|---|---|---|---|---|
| Per Cent. Acid in Lixiviant | Per Cent. Cu Extracted | Lb. Acid Used per Lb. Cu Dissolved | Per Cent. Cu Extracted | Lb. Acid Used per Lb. Cu Dissolved |
| 3.0 | 46.8 | 3.2 | 48.7 | 3.3 |
| 4.0 | 54.7 | 3.3 | 71.0 | 2.7 |
| 4.7 | 56.1 | 2.8 | 85.2 | 3.2 |
| 10.0 | 81.8 | 2.8 | 88.6 | 3.5 |

*4-Mesh Ore*

| | Standing Lixiviant | | Circulating Lixiviant | |
|---|---|---|---|---|
| Per Cent. Acid in Lixiviant | Per Cent. Cu Extracted | Lb. Acid Used per Lb. Cu Dissolved | Per Cent. Cu Extracted | Lb. Acid Used per Lb. Cu Dissolved |
| 3.0 | 42.9 | 2.4 | 70.4 | 2.0 |
| 4.0 | 44.3 | 2.4 | 74.4 | 2.3 |
| 4.7 | 48.3 | 2.3 | 82.3 | 2.4 |
| 6.3 | 58.2 | 2.3 | 80.8 | 2.8 |
| 10.0 | 78.4 | 2.2 | 85.7 | 2.8 |

These experiments showed that circulating the lixiviant increased the extraction of copper without seriously increasing the consumption of acid; that crushing to 8 mesh increased the consumption of acid without materially increasing the extraction of copper; and that increasing the

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strength of the lixiviant increased the extraction of copper and also the consumption of acid when the lixiviant was circulated through the ore. The fine ore produced in crushing to 8 mesh seriously interfered with the percolation of solutions.

Further experiments were made on 4-mesh material by increasing the time of leaching, with the following results:

| | 5 Per Cent. Acid Lixiviant | | 10 Per Cent. Acid Lixiviant | |
|---|---|---|---|---|
| Time of Leaching, Hr. | Per Cent. Cu Extracted | Lb. Acid Used per Lb. Cu Dissolved | Per Cent. Cu Extracted | Lb. Acid Used per Lb. Cu Dissolved |
| 24 | 82.3 | 2.4 | 85.7 | 2.8 |
| 48 | 83.7 | 2.6 | 90.1 | 2.6 |
| 72 | 84.2 | 2.7 | 87.6 | 4.2 |
| 96 | 86.7 | ... | 89.1 | 4.0 |

Additional time with 5 per cent. acid lixiviant increased the extraction very slowly—probably from the fact that the acid was nearly neutralized. The 10 per cent. acid lixiviant always had plenty of free acid remaining after the copper was all dissolved.

By referring to the analysis of this ore, it will be seen that the copper soluble in 10 per cent. sulphuric acid solution is only 90.6 per cent. when the sample is crushed to pass 100-mesh screen, so that a 10 per cent. acid lixiviant on a 4-mesh material for 48 hr. extracted about all of the copper that could be expected.

The above experiment was repeated on ore crushed to pass 0.5-in or 2-mesh screen. Only a 10 per cent. acid lixiviant was used. The results, given below, were as good as those obtained from finer crushing.

| Time of Leaching | Per Cent. Cu Extracted |
|---|---|
| 48 hr. without change of lixiviant..... | 86.0 |
| 48 hr. Lixiviant changed after 24 hr..... | 87.5 |
| 72 hr. Lixiviant changed after 48 hr..... | 90.0 |
| 96 hr. Lixiviant changed after 48 hr..... | 87.0 |

As already stated, the oxidized ore contained some cuprite. When clean sulphuric acid is added to this mineral, one-half the copper is set free in metallic condition, which remains insoluble in the acid. As nearly as could be determined by averaging a number of analyses, the amount of copper thus set free was about 0.2 per cent. of the ore, or say 10 per cent. of the total copper.

Analyses of the lixiviants from a number of leaching experiments showed that a certain amount of iron passed into solution simultaneously with the copper. This averaged from one-third to one-half of the amount of copper in solution, or from 0.8 to 1.1 per cent. of the solution. Some-

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times, with high iron and much kaolinized ores, it amounted to more than this. The iron was about equally divided as ferric and ferrous sulphates.

Using 10 per cent. acid, the lixiviant passing from a charge of leached ore contained from 4 to 6 per cent .free acid in addition to the copper, iron, alumina, etc.

This combination of circumstances enabled me to use the partly neutralized lixiviant from a leached charge of ore for the first lixiviant on a new charge of ore and thereby not only neutralize the free acid in the lixiviant and at the same time enrich it in copper, but have a solvent for the cuprite (or metallic coppper if formed) in the form of ferric sulphate. Since ferric sulphate was always present and the lixiviants were always circulated, it is scarcely possible that any cuprite could escape solution.

As soon as the first lixiviant was neutral, it was sent to the precipitating vats. If it was left on the ore beyond the neutral point, some of the copper was reprecipitated in the ore. A new 10 per cent. acid lixiviant followed the first and finished leaching the ore, which in turn was neutralized on a new charge of ore.

## Experimental Plant Tests

### *The Chloridizing Roast*

The chloridizing roast and the subsequent leaching of the ore with water and scrubbing-tower solutions is a well-established metallurgical process and has been in commercial operation for a number of years on 2 per cent. copper ores.

Laboratory experiments told what could be expected from this process and there is little danger of unlooked-for difficulties in larger-scale operations. The questions involved are therefore more of relative costs of construction and operation than of metallurgical experiment, so, for the time being, no experiments were conducted on a larger scale.

### *Acid Leaching*

*Percolation Experiments.*—A pipe 10 in. in diameter and 18½ ft. in length was used for an experimental percolation vat. For convenience in testing the various columns of ore, it was made in sections. The lower end of this vat was closed with a plate in which there was a 3-in. opening which served as a discharge gate for the ore and also for pipe connections to measure the flow of solution. This could also be attached to a centrifugal pump for circulating the solutions to the top of the vat if desired and in this way the working conditions of a commercial plant

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could be produced. Aside from a small screen to protect the valves, no filter was used in the vat. The ore itself corresponded to the gravel filter that was expected to be used in practice.

The height of any ore column in this vat was the same as that used in any commercial plant. The filtering area was 78.54 sq. in., or 0.545 sq. ft., which corresponded to the same section of a full-sized vat. The area of any commercial vat divided by this area and the result multiplied by the number of gallons per minute flowing through a given column of ore would give the approximate amount of solution that would have to be handled by circulating pumps in a large plant.

All percolation experiments were continued for three or four days if possible, in order to detect any irregularity in the flow of solution by the segregation of slimes or from other causes.

Since 8-mesh ore was found to be of no advantage in the laboratory leaching experiments, it was not used in these experiments.

The facilities at Douglas for crushing ore for experimental purposes at that time were very poor. There was no opportunity for stage crushing with intermediate screening. The ore was reduced so that it would nearly all pass a 2-mesh screen in one operation with one crusher and one set of rolls. This undoubtedly produced a different product than would be obtained in the usual practice.

*With 4-mesh ore.* A screen analysis of this material gave

| Mesh | Per Cent. |
|---|---|
| 4 to 8 | 30.6 |
| 8 to 16 | 22.4 |
| 16 to 20 | 8.4 |
| 20 and finer | 38.6 |

Between 3 and 4 per cent. of this product was a colloidal slime that could be suspended in water. This slime assayed 4.5 per cent. copper.

Ore crushed to this size absorbed 11 per cent. of its weight in water or 26.4 gal. per ton. This amount of water will be lost in tailings.

The additional quantity of water required to fill the interstices and cover a given quantity of ore in the vat was 15 per cent. of its weight or 36 gal. per ton. This will be known as one volume of water, or solution, in the experiments which follow, and its complete replacement will represent one cycle in the circulation of lixiviant, or one washing of the ore with water.

Experiment No. 1. The ore was charged into water to get the coarser particles on the bottom and form a better filter bed. Height of ore column, 5 ft. Fine ore and slime soon settled on top of the charge and practically stopped the percolation.

Experiment No. 2. Ore was charged into the vat dry. Height of ore column, 5 ft. Water was added on top of ore. Rate of percolation, 1

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gal. in 15 min. To the charge, 2.5 ft. of ore was added, making the total column 7.5 ft. Rate of percolation, 1 gal. in 15 min. Another 2.5 ft. of ore was added, making the total column 10 ft. Rate of percolation, 1 gal. in 15 min. At the end of 72 hr. the rate was 1 gal. in 23 min., after which it gradually decreased.

When the charge was drawn, the fine ore was found to be more or less segregated, which was perhaps due to the method of charging.

Experiment No. 3. The above experiment was repeated. With a 5-ft. column of ore the rate of percolation was the same, 1 gal. in 15 min. The ore column was increased at once to 10 ft., in which the rate of percolation was 1 gal. in 10 min. The rate remained the same when the column was increased to 12 ft.

Experiment No. 4. The ore was charged dry and the height of the column was 18 ft. Water was introduced into the bottom of the vat until the ore was covered, in order to remove the air more quickly. It was then circulated from the bottom of the vat to the top by means of a centrifugal pump. Rate of percolation was 1 gal. in 8 min. At the end of 72 hr. it was still 1 gal. in 8 min. The height of the ore column was then reduced to 12 ft. and at the end of 120 hr. the rate of percolation was 1 gal. in 9 min.

Experiment No. 5. Upward percolation. The ore was charged dry. Height of column, 12 ft. Water was introduced from the bottom of the vat and upward percolation was maintained with a 9-in. head of water above the ore from a stand pipe outside the vat. The rate of percolation was 1 gal. in 6 min.

With an ore column of 15 ft. and a 10-in. head of water above the ore, the rate of percolation was 1 gal. in 7.5 min.

*With 2-mesh ore.* The screen analysis was

| Mesh | Per Cent. |
|---|---|
| 2 to 4 | 40.0 |
| 4 to 8 | 25.8 |
| 8 to 16 | 14.2 |
| 16 to 20 | 6.0 |
| 20 and finer | 14.2 |

The colloidal slime was about 2 per cent. of the ore.

This ore absorbed 6 per cent. of its weight or 14.4 gal. of water per ton.

The volume of water necessary to cover the saturated ore was 27.7 per cent. of its weight or 66.5 gal per ton.

Experiment No. 1. Height of ore column, 12 ft. Water was circulated from bottom of vat to top by means of pump. Rate of percolation was 1 gal. in 30 sec. At the end of 96 hr., it was 1 gal. in 35 sec.

*Summary.*—The height of the ore column with 4 and 2 mesh ore has

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no retarding effect on the rate of percolation, unless the ore is very much disintegrated and produces a good deal of slime. If anything, the rate is increased with increased height. Vats can therefore be constructed with any convenient depth, but will probably not be practicable beyond 10 or 15 ft., depending of course upon the character of ore and method of crushing.

The ore should be charged dry or only moist.

Lixiviants can be introduced best from the bottom of the vat or down one side of the vat from the top of the ore. This allows the free escape of air and carbon dioxide gas and permits a rapid saturation of the ore and filling of the vat. After the ore is covered, the percolation of the lixiviant through the ore should be downward. This permits better control of solutions and the washing of the ore with the minimum quantity of water.

By keeping the ore covered with solution and having the pumping or circulating capacity a little above the rate of percolation, channels through the ore can be detected and remedied.

The rate of percolation in this experimental vat may appear slow, but when the proper calculation is made for a large vat, it will be found that considerable pumping capacity will be required to take care of the solution.

### *Leaching Plant Construction*

To meet the conditions at Ajo, reinforced concrete seemed to offer many advantages for large-sized leaching vats. This material has been used for a long time at the works of the Pennsylvania Salt Co., at Natrona and at Philadelphia, for leaching small tonnages of chloridized copper ores. The solutions there are a mixture of sulphate and chloride of copper and contain from 1 to 2 per cent. of free acid. No protective coating or lining has been used for the concrete although the new vat constructed at Philadelphia is lined with chemical brick laid with cement.

The stronger acid solutions used in leaching the Ajo ores necessitated some protective covering for the concrete. A paint of this character is made by Toch Brothers. It is known as "R. I. W. No. 44" and is advertised to have the following properties:

"1. It stands heat up to the point of carbonization.

"2. It withstands sulphuric acid, 20° strength, for two months without showing the slightest sign of decomposition.

"3. It withstands the action of 15 per cent. sulphuric acid, 10 per cent. copper sulphate, at a temperature of 135° F., for six months, without showing any sign of decomposition. Withstands influence of caustic soda up to about 10 per cent. strength.

"4. Two coats will last for several years on the interior of stand pipes. Steel must be clean and dry before material is applied.

"5. Has a record of three years and four months on the interior walls, floors and

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ceilings of storage battery rooms where the vapors of sulphuric acid are condensed twice a day. Three coats were applied.

"The dates of its longevity are simply minimum dates, as the material lasts much longer than our statements indicate."

The protection of these concrete vats was submitted to Toch Brothers and they recommended the following specifications:

" Make your concrete a 1 : 2 : 4 mix, using an aggregate graded up to 1½ in., adding 3 lb. of "Toxement" for each bag of cement necessary to complete the operation. After the concrete has set up thoroughly dry, clean the surface and then apply one coat of R. I. W. No. 89, allow about 36 hr. for it to dry, then apply a second coat of R. I. W. No. 89. Care should be taken after the first coat is applied so that every particle of the surface is thoroughly and completely covered. R. I. W. No. 89 is constructed slightly different from R. I. W. No. 44 in order to meet your specifications."

To make a concrete vat acid proof, it must first be made waterproof. For this purpose they recommended "Toxement," a patented preparation of their own which is advertised to be "a colloidal double resinate and silicate of calcium and aluminum."

A second method of waterproofing concrete was obtained from *Bulletin* No. 46, U. S. Department of Agriculture. In this method crude petroleum or preferably residuum is mixed with the concrete as follows:

"For most purposes where damp proofing is required, 5 per cent. of oil based on the weight of the cement in the mixture is all that is necessary. For each bag of cement (weighing 94 lb.) 4.7 lb. or about 2½ quarts of oil are required.

"Mix the concrete in proportions of 1 part cement to 2 of sand and 4 of broken stone or gravel. Add water and thoroughly mix until no trace of oil is visible on the surface. If oil-mixed mortar is desired, prepare in the same manner without the gravel.

"The use of 5 per cent. of oil increases the time of the initial set by 50 per cent. and the time of the final set by 47 per cent.

"Concrete with 10 per cent. oil has 75 per cent. of the strength of plain concrete at 28 days. At the end of one year the strength of 1 : 3 mortar suffers but little from the addition of oil in amounts up to 10 per cent.

"Oil-mixed mortar containing 10 per cent. of oil is absolutely water tight under pressures as high as 40 lb. to the square inch. Oil-mixed mortar is effective as a waterproofing agent under low pressures when plastered on either side of porous concrete."

Tests were also made on the acid-resisting qualities of the following brick, which might be used for lining concrete vats: Ordinary wire-cut shale brick from El Paso; "acid-proof tile" from El Paso; low-temperature fire brick from El Paso; "Star" fire brick from Pueblo, Col.; silica brick from Pueblo, Col.

These bricks were allowed to stand three months in 10 per cent. sulphuric acid solution; 5 per cent. sulphuric acid and 5 per cent. copper sulphate solution; and 10 per cent. copper sulphate solution (10 per cent. $CuSO_4 + 5H_2O$).

There was no apparent disintegration or injury to the physical structure of any of these brick at the end of the test.

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Chemical analysis of the solutions likewise showed that none of the brick had been appreciably attacked by either acid or copper sulphate solutions.* The solutions from the silica brick contained some gelatinous silica, which probably came from the decomposition of the sodium silicate used as a binder, but this did not weaken the brick.

Two concrete storage tanks were made for holding wash water, in order to bring it up to normal strength (about 10° B.) of acid to be used as a lixiviant. In practice it would be better to build these tanks of steel or wood and line them with lead, but in this experimental plant concrete was used to determine its efficiency as well as that of acid-proof paints. Both of these tanks were made of oil-mixed reinforced concrete.

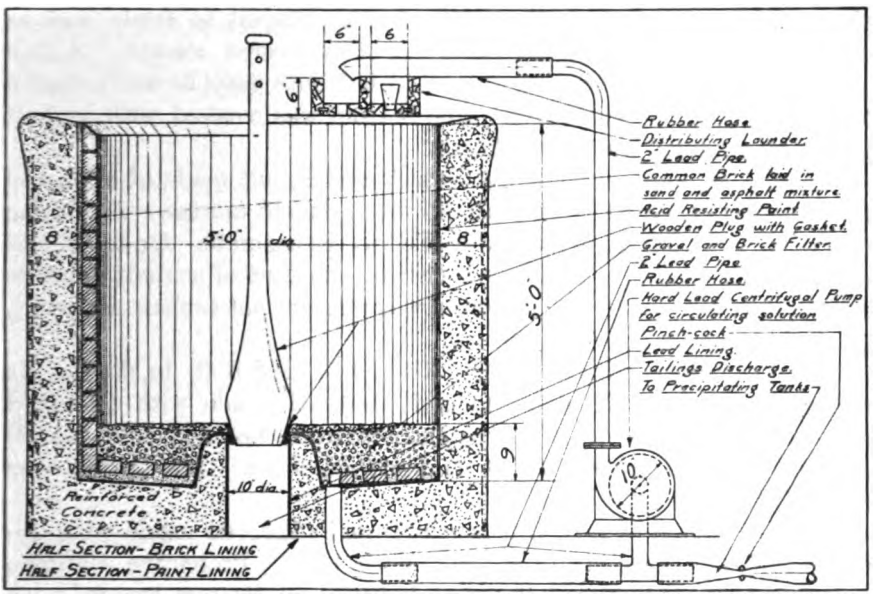


FIG. 1.—EXPERIMENTAL LEACHING TANKS.

They were 5 ft. in diameter and 5 ft. in depth, with walls 8 in. thick. One was plastered inside with "Toxement" mortar and the other with oil-mixed mortar. Both were painted inside with R. I. W. No. 89 paint as specified. For better protection from abrasion, the bottoms of these tanks were covered with ½ in. or more of hot asphalt and sand, troweled smooth and compact with a hot iron like street paving.

Five leaching vats were constructed according to the general design shown in Fig. 1.

For convenience in subsequent work these vats were designated as *A*, *B*, *C*, *D*, and *E*, and their individual construction was as follows:

Vat *A*. Five feet in diameter and 5 ft. in depth. Concrete, mixed

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with "Toxement" according to the Toch specifications. Plastered inside with cement mortar mixed with "Toxement." Painted inside with three coats of R. I. W. No. 89 paint. A false lining of wood was made to set inside this vat, but it was impracticable to retain this lining without scarring the paint, so it was abandoned at the start. It was also impossible to keep the paint from being worn off by the ore, so the vat was given one coat of asphalt dissolved in gasoline and then lined with El Paso common brick. The bricks were laid in hot asphalt, and hot asphalt and sand were poured between the brick lining and the concrete. The bottom was covered with asphalt and sand, troweled smooth while hot.

Vat *B*. Same construction and dimensions as Vat *A*. This was given two coats of hot asphalt, which was difficult to apply and on cooling it contracted and left pin holes of uncovered cement. A thin paint was then made by pouring hot asphalt into gasoline and a coat of this was applied to the above. The bottom was covered with asphalt and sand, troweled smooth while hot.

Vat *C*. Same dimensions as Vats *A* and *B*, but made of oil-mixed concrete and plastered inside with oil-mixed cement mortar. A priming coat of R. I. W. No. 89 paint was applied to the concrete. On top of this was placed a coat of hot asphalt followed by a coat of asphalt paint as described under Vat *B*. The bottom was made of hot asphalt and sand, as already described.

Vat *D*. This was rectangular, 5 by 8 ft. and 5 ft. in depth. It was made of oil-mixed concrete and plastered inside with oil-mixed cement mortar. The protective coatings were one coat of R. I. W. No. 89 and two coats of asphalt paint as described above. The bottom was covered with hot asphalt and sand.

Vat *E*. Made of oil-mixed concrete and plastered inside with oil-mixed mortar. Diameter 5 ft. and depth 10 ft. Protective coatings, one of R. I. W. No. 89 and one of asphalt paint. Bottom covered with hot asphalt and sand.

The capacity of Vats *A*, *B*, and *C* was between 3 and 4 tons, respectively. Vats *D* and *E* each held a little more than 8 tons.

Three solution or precipitating tanks, 5 ft. in diameter and 5 ft. in depth, were made and protected in the same manner as Vat *E*.

### *Leaching Plant Operation*

These experiments were made on about 300 tons of oxidized ore shipped from the various shafts and test pits and during the development of the property.

*Test* 1.—This was made on two cars of ore, or Lots 1 and 2, which were crushed and mixed for treatment. As already stated, owing to

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limited crushing facilities, it was necessary to crush with one crusher and one set of rolls, as much as possible to 2 mesh and finer in one operation. The result of this was only 7.25 per cent. of oversize and probably a greater production of fines than would be obtained in practice.

Lot 1 was hard unaltered ore similar to that used in the laboratory experiments. Lot 2 was considerably altered by weathering and produced a great deal of fine material and slime.

A screen analysis of the mixture gave the following results:

| Mesh | Per Cent. |
|---|---|
| 2 to 4 | 29 |
| 4 to 8 | 14 |
| 8 to 16 | 18 |
| 16 and finer | 39 |

The chemical analyses of the two lots are given below:

| | Dry Weight, Pounds | Copper, Per Cent. |
|---|---|---|
| Lot 1 | 62,172 | 1.84 |
| Lot 2 | 94,570 | 1.37 |
| | 156,742 | 1.55 |

*Chemical Analyses*

| | Lot 1 Per Cent. | Lot 2 Per Cent. | | Lot 1 Per Cent. | Lot 2 Per Cent. |
|---|---|---|---|---|---|
| $SiO_2$ | 65.7 | 63.6 | Cu, total | 1.84 | 1.37 |
| Fe | 4.0 | 4.7 | Cu sol. in 10 per cent. $H_2SO_4$ | 1.66 | 1.27 |
| $Al_2O_3$ | 14.3 | 13.4 | $CO_2$ | 1.70 | 1.33 |
| | | | | Ounce | Ounce |
| CaO sol. in acid | 0.8 | 0.6 | Au | 0.01 | 0.01 |
| S, total | 0.3 | 0.4 | Ag | 0.20 | 1.18 |
| S as sulphate | 0.08 | 0.1 | | | |

The leaching was conducted in the manner described under laboratory experiments, namely, the partly neutralized lixiviant from finishing one charge of ore was neturalized and enriched by using it as the first lixiviant on a new charge of ore.

The details of this test are summarized as follows:

| | |
|---|---|
| Total ore leached, pounds | 125,85 |
| Average copper contents, per cent | 1.507 |
| Average time of acid leaching, hours | 70.6 |
| Average time of washing, hours | 5.0 |
| Final tailing, total copper, per cent | 0.3 |
| Final tailing, soluble copper, per cent | 0.10 |
| Copper extracted per ton of ore, pounds | 24.42 |
| Per cent. extraction | 80.3 |

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| | |
|---|---|
| Total weight of material extracted from the ore, per cent | 3.5 |
| Actual percentage of copper extracted | 81.0 |
| Percentage of available copper extracted | 91.0 |
| Lixiviant used per ton of ore, gallons | 132 |
| Average per cent. of acid in lixiviant | 9.24 |
| Acid consumed per pound of copper dissolved, pounds | 3.4 |
| Wash water used per ton of ore, gallons | 138 |
| Rate of percolation, measured in inches of solution above the ore in vat, inches per hour | 90 |
| Copper accounted for in solution, of the total amount extracted from the ore by difference between heads and tails, per cent | 92.2 |
| Average value of nearly neutralized lixivium sent to precipitating vats: | |
| Copper, per cent | 2.22 |
| Iron, per cent | 0.42 |
| Alumina, per cent | 0.47 |

The average grade of the ore in this test was lower than that used in the laboratory tests. There is, however, not likely to be a corresponding decrease in value of the tailing from a 1.5 per cent. ore, below those obtained from a 2.0 per cent. ore, so the percentage of extraction decreases rapidly in proportion to the decrease in the grade of the ore.

The ore was charged dry into the vats. The lixiviants were admitted on top of the ore but at one side of the vat, and no faster than could be absorbed by the charge. By making a little dam on top of the charge, the lixiviant passed immediately to the bottom of the vat and saturated the charge from below, thus permitting the escape of air and gas.

The soluble copper in the tailing was determined by boiling the laboratory sample (100 mesh) in 10 per cent. sulphuric acid. The copper not soluble by this treatment must exist as sulphide or as other minerals in the ore, not capable of being extracted by sulphuric acid and therefore is not available by this metallurgical process.

*Tests 2 and 3.*—These were made on car lots 3 and 4 treated separately.

The ore was soft and much the same in character as Lot 2. It produced considerable fine material on crushing.

Lot 3 was crushed to ½-in. mesh.

Lot 4 under similar conditions crushed almost to ¼-in. mesh, as shown by screen analyses.

| Mesh | Lot 3 Per Cent. | Lot 4 Per Cent |
|---|---|---|
| −2+4 | 16 | 6 |
| −4+8 | 27 | 27 |
| −8+16 | 17 | 21 |
| −16 | 40 | 46 |

| | Dry Weight Pounds | Copper Per Cent. |
|---|---|---|
| Lot 3 | 111,230 | 1.48 |
| Lot 4 | 89,670 | 1.44 |

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*Chemical Analyses*

| | Lot 3 Per Cent. | Lot 4 Per Cent. | | Lot 3 Per Cent. | Lot 4 Per Cent. |
|---|---|---|---|---|---|
| $SiO_2$ | 64.00 | 63.30 | Cu total | 1.48 | 1.44 |
| Fe | 4.20 | 4.20 | Cu sol. in 10 per cent. | | |
| $Al_2O_3$ | 14.30 | 15.10 | $H_2SO_4$ | 1.32 | 1.30 |
| CaO sol. in acid | 0.45 | 0.60 | $CO_2$ | 1.42 | 1.43 |
| | | | | Ounce | Ounce |
| S, total | 0.20 | 0.20 | Au | 0.01 | 0.01 |
| S as sulphate | 0.04 | 0.06 | Ag | 0.19 | 0.17 |

The results from these tests are shown on the following table:

| | Test 2 | Test 3 |
|---|---|---|
| Pounds ore leached | 90,533 | 80,458 |
| Per cent. copper | 1.44 | 1.43 |
| Average hours acid leaching | 70 | 100 |
| Average hours washing | 7 | 23 |
| Final tailing, per cent. total copper | 0.27 | 0.40 |
| Final tailing, per cent. soluble copper | 0.23 | 0.38 |
| Pounds copper extracted per ton of ore | 23.3 | 20.6 |
| Percentage of copper extracted | 81.2 | 71.8 |
| Total per cent. of material extracted from ore | 3.0 | 3.0 |
| Actual percentage of copper extracted | 81.8 | 72.7 |
| Percentage of available copper extracted | 83.5 | 73.0 |
| Gallons lixiviant used per ton of ore | 123 | 137 |
| Average per cent. acid in lixiviant | 9.5 | 7.0 |
| Pounds acid consumed per pound copper dissolved | 3.5 | 3.3 |
| Gallons wash water used per ton of ore | 135 | 177 |
| Rate of percolation in inches per hour, measured in inches of solution above the ore in vat | 43 | 11 |
| Copper accounted for in solution | Combined with Test 4. | |
| Average value of nearly neutralized lixivium sent to precipitating vats: | | |
| Copper, per cent | 2.41 | 2.90 |
| Iron, per cent | 0.56 | 0.70 |
| Alumina, per cent | 0.53 | 0.73 |

These results show very clearly the effect of finer crushing on the same grade and character of ore. Both of these lots were substantially the same character of ore in every respect. They were very much altered by oxidation and contained practically no insoluble copper in the tailing, although the ore itself showed 0.08 to 0.10 per cent., probably as cuprite, which dissolved in the ferric salts extracted from the ore.

The extraction on the 4-mesh material was 9.4 per cent. lower and the rate of percolation dropped to one-quarter of that on the 2-mesh material. Hence much more time was consumed in leaching and washing the ore, with poorer results. Even on the 2-mesh ore the rate of percolation was only half that of the harder ore in Test 1, showing the effect of increased kaolinization of the ore.

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In order to determine the relative value of samples from the top, and other parts of the charge, and also to determine whether the charges were leaching uniformly or in channels, a number of samples were taken from different charges at each foot in depth. This was done by removing half the charge from the vat and sampling the standing face of the remaining half of the charge across each foot of vertical section at a point half way between the center plug and the side walls of the vat.

A general sample was also taken in this manner, to check, not only the sectional samples, but also the one-tenth portion of the entire charge which was quartered in the usual manner for a tailing sample.

These results show that the leaching is very uniform throughout the charge on evenly distributed 2-mesh material, and that samples taken from the top of the charge may be used for preliminary tailing.

| Lot 3, Test 4 | Charge 22, Per Cent. Cu | Charge 24, Per Cent. Cu | Charge 25, Per Cent. Cu |
|---|---|---|---|
| Sample | | | |
| First foot, top | 0.26 | 0.35 | 0.25 |
| Second foot | 0.27 | 0.30 | 0.25 |
| Third foot | 0.27 | 0.23 | 0.30 |
| Fourth foot | 0.29 | 0.26 | 0.35 |
| Average | 0.27 | 0.28 | 0.29 |
| One-tenth portion of charge | 0.26 | 0.27 | 0.26 |

Similar samples from the 4-mesh material (Car Lot 4, Test 3) did not give as uniform results. This contained not only much more fine material, but about 5 per cent. colloidal slime, as compared with 2 per cent. colloidal slime in the hard ore. Even though the ore was charged dry, this fine material was found to be irregularly segregated from the charging and apparently not from the action of the lixiviants. This segregation produced irregularities in the extraction by interference with the percolation and washing.

The results from the sectional sampling are given below:

| Sample | Charge 26, Per Cent. Cu | Charge 27, Per Cent. Cu | Charge 28, Per Cent. Cu | Charge 29, Per Cent. Cu | Charge 32, Per Cent. Cu |
|---|---|---|---|---|---|
| First foot, top | 0.24 | 0.31 | 0.23 | 0.46 | 0.24 |
| Second foot | 0.21 | 0.38 | 0.44 | 0.75 | 0.44 |
| Third foot | 0.33 | 0.40 | 0.27 | 1.10 | 0.34 |
| Fourth foot | 0.34 | 0.33 | 0.78 | 0.83 | 0.35 |
| Fifth foot | ..... | ..... | ..... | 0.31 | ..... |
| Sixth foot | ..... | ..... | ..... | 0.44 | ..... |
| Seventh foot | ..... | ..... | ..... | 0.46 | ..... |
| Eighth foot | ..... | ..... | ..... | 0.31 | ..... |
| Ninth foot | ..... | ..... | ..... | 0.75 | ..... |
| Average | 0.28 | 0.35 | 0.43 | 0.60 | 0.35 |
| General | 0.24 | 0.28 | 0.37 | 0.44 | 0.35 |
| One-tenth portion of charge | 0.26 | ..... | 0.37 | ..... | ..... |

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The soluble copper in all of the tailings from these two lots of ore shows its weathered condition by the almost total absence of copper as cuprite or sulphide.

The difficulty in extracting this copper led to a more detailed analysis of the tailings to determine whether the copper was retained by the coarse material alone, or whether the colloidal slime also had a tendency to absorb and retain copper even after it had gone into solution.

These results are interesting since they show that the colloidal slime absorbs, or retains mechanically, as much copper as is held by the coarse material. As anticipated, there is also an increase in the percentage of alumina in the fine material.

*Screen and Chemical Analyses of Charge* 25, *Lot* 3, *Test* 2

| Size of Material, Mesh | Per Cent. of Weight | Per Cent. of Value | Per Cent. of Copper | Per Cent. Sol. Copper | Per Cent. of Silica | Per Cent. of Alumina |
|---|---|---|---|---|---|---|
| +4 | 9.3 | 15.2 | 0.47 | 0.39 | 68.0 | 12.6 |
| +8 | 32.5 | 30.6 | 0.27 | 0.20 | 69.6 | 12.7 |
| +16 | 23.2 | 19.5 | 0.24 | 0.18 | 68.5 | 13.9 |
| +30 | 14.0 | 10.8 | 0.22 | 0.17 | 66.0 | 14.0 |
| +60 | 4.7 | 4.3 | 0.26 | 0.19 | 66.9 | 13.5 |
| +100 | 7.0 | 6.6 | 0.27 | 0.21 | 66.3 | 13.1 |
| −100 | 9.3 | 13.0 | 0.40 | 0.37 | 63.3 | 16.4 |
| | 100.0 | 100.0 | 0.28 | 0.22 | 67.7 | 12.6 |

*Screen and Chemical Analyses of Charges* 29 *and* 30, *Lot* 4, *Test* 3

| | Charge 29 | | | Charge 30 | | |
|---|---|---|---|---|---|---|
| Size of Material, Mesh | Per Cent. Weight | Per Cent. Cu | Per Cent. Sol. Cu | Per Cent. Weight | Per Cent. Cu | Per Cent. Sol. Cu |
| +4 | 11.8 | 0.65 | 0.65 | 7.5 | 0.62 | 0.61 |
| +8 | 25.9 | 0.54 | 0.53 | 24.6 | 0.47 | 0.46 |
| +16 | 16.0 | 0.51 | 0.50 | 15.1 | 0.55 | 0.55 |
| +30 | 16.8 | 0.54 | 0.54 | 17.0 | 0.58 | 0.58 |
| +60 | 7.0 | 0.59 | 0.58 | 9.4 | 0.65 | 0.64 |
| +100 | 8.0 | 0.59 | 0.59 | 15.1 | 0.70 | 0.70 |
| −100 | 14.5 | 0.57 | 0.57 | 11.3 | 0.77 | 0.77 |
| Average..... | 100.0 | 0.56 | 0.55 | 100.0 | 0.60 | 0.59 |
| General samples | ..... | .... | 0.44 | ..... | 0.42 | .... |

This retention of the copper by the slime was also noticeable in washing the ore. Although the charges in Lot 4, Test 3, were washed a much longer time and with 50 per cent. more water than those in the previous tests, the analyses of the wash waters showed a very slow removal of the copper. Where the percolation was slow and the tailings high, even the sixth volume of wash water contained a high percentage of copper, while the acid was removed as readily as in previous tests. This confirms the well-known properties of colloidal or argillaceous material for retaining mineral salts.

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The rate of percolation in Test 3, Lot 4, dropped to one-eighth of the rate in Test 1.

The acid consumption in these two tests was substantially the same as in Test 1. The average strength of the acid used in Test 3 was 7 per cent. as compared with 9.25 per cent. and 9.5 per cent. in the preceding tests. In this test partly neutralized lixiviants were used on new ore in every charge, which could not always be done in the other tests. This had a tendency to lessen the consumption of acid, and neutral or nearly neutral solutions could be sent to the precipitating tanks. As stated before, if the acid contents of these lixiviants are too low and they become completely neutralized, part of the copper is reprecipitated in the ore. It would therefore be advisable to send the lixivium to the precipitating tanks slightly acid. Lixiviums containing relatively high percentages of ferric sulphate, and perhaps aluminum sulphate in the form of alum, show an acid reaction that corresponds to 0.2 per cent. to 0.5 per cent. of sulphuric acid.

*Test* 4.—This was made on Car Lot 5.

| | |
|---|---|
| Dry weight, pounds | 96,924 |
| Per cent. copper | 1.20 |

This ore was hard and contained more sulphides and cuprite than the previous lots except Lot 1.

*Chemical Analysis*

| | Per Cent. | | Per Cent. |
|---|---|---|---|
| $SiO_2$ | 65.50 | Cu, total | 1.20 |
| Fe | 3.90 | Cu, sol. in 10 per cent. $H_2SO_4$ | 1.04 |
| $Al_2O_3$ | 14.60 | $CO_2$ | 0.59 |
| CaO sol. in acid | 0.80 | | Ounce |
| MgO | 1.15 | Au | 0.01 |
| S, total | 0.44 | Ag | 0.17 |
| S as sulphate | trace | | |

The ore treated was screened through ½-in. or 2-mesh screen. Only 27.5 per cent. was finer than 16 mesh as compared with 40 per cent. in the preceding tests.

*Screen Analysis*

| Mesh | Per Cent. Weight | | Per Cent. Copper |
|---|---|---|---|
| +4 | 26.2 | | 1.04 |
| +8 | 30.3 | | 1.11 |
| +16 | 16.0 | | 1.28 |
| +30 | 11.3 | | 1.40 |
| +60 | 5.5 | 27.5 | 1.07 |
| +100 | 5.0 | | 1.92 |
| −100 | 5.7 | | 1.94 |
| | 100.0 | | 1.37 |

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The results from leaching this material are shown on the following table.

The oversize from Lots 3, 4 and 5, amounting to 4 per cent. of the total ore treated and ranging from ½ to 1 in. in size, was recrushed to pass a 2-mesh screen and leached separately. This oversize was substantially the same in character as the finer material but had passed through the rolls uncrushed due to the irregular hand feeding.

A screen analysis of the recrushed oversize gave the following results:

| Mesh | Per Cent. |
|---|---|
| +4 | 32.0 |
| +8 | 26.3 |
| +16 | 14.7 |
| −16 | 27.0 |

The leaching tests on this product are also shown in the following table:

| | Test 4 | Recrushed Oversize Tests 2, 3 and 4 |
|---|---|---|
| Pounds ore leached | 77,636 | 10,300 |
| Per cent. copper | 1.29 | 1.49 |
| Average hours acid leaching | 77 | 90 |
| Average hours washing | 10 | 10 |
| Final tailings, per cent. total copper | 0.32 | 0.24 |
| Final tailings, per cent. soluble copper | 0.16 | 0.16 |
| Pounds copper extracted per ton of ore | 19.3 | 25.2 |
| Percentage of copper extracted | 75.0 | 83.8 |
| Total percentage of material extracted from the ore | 3.0 | 3.0 |
| Actual percentage of copper extracted | 75.8 | 84.4 |
| Percentage of available copper extracted | 82.9 | 91.2 |
| Gallons lixiviant used per ton of ore | 120 | 145 |
| Average per cent. acid in lixiviant | 9.3 | 10.8 |
| Pounds acid consumed per pound of copper dissolved | 4.3 | 3.8 |
| Gallons wash water used per ton of ore | 666 | 712 |
| Rate of percolation in inches per hour, measured in inches of solution above ore in vat | 161 | 75 |
| Per cent. of copper accounted for in solution, of the total amount extracted from the ore by difference between heads and tails. (Combined solutions from Tests 2, 3, and 4) | 99.8 | ..... |
| Average value of nearly neutralized lixivium sent to precipitating vats | | |
| Copper, per cent | 2.30 | ..... |
| Iron, per cent | 0.56 | ..... |
| Alumina, per cent | 0.66 | ..... |

The lower extraction in this test is due to the lower grade of the ore and the high percentage of sulphides in the ore. I have invariably



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found in experimenting with low-grade ores that the value of the tailing is a very constant factor, whether the value of the ore be $5 or $10 per ton.

In the analyses of the tailings in Tests 2 and 3 we found that practically all of the copper was soluble copper and that the extremely fine material carried more copper than some of the coarser sizes.

In this test several charges of the leached ore were washed with as much as 1,500 gal. of water per ton, or until the final wash water showed no copper or acid, which accounts for the high average volume of wash water in the results given above. This extra and complete washing had absolutely no effect on the soluble copper in the tailings, which remained as high as in charges washed with one-tenth of the water.

The following screen analyses on the tailings from two of these charges show that even the finest material retains copper soluble in sulphuric acid that will not wash out. Charge 41 came from *E* vat, 9 ft. in depth, and Charge 43 came from *A* vat, 4 ft. in depth.

| | Charge 41 | | | Charge 43 | | |
|---|---|---|---|---|---|---|
| Size of Material Mesh | Per Cent. Weight | Per Cent. Total Cu | Per Cent. Sol. Cu | Per Cent. Weight | Per Cent. Total Cu | Per Cent. Sol. Cu |
| +4..... | 27.8 | 0.30 | 0.23 | 23.5 | 0.32 | 0.24 |
| +8..... | 30.9 | 0.27 | 0.20 | 34.7 | 0.17 | 0.11 |
| +16..... | 16.6 | 0.32 | 0.21 | 16.5 | 0.12 | 0.08 |
| +30..... | 10.8 | 0.34 | 0.25 | 9.5 | 0.18 | 0.13 |
| +60..... | 4.6 | 0.49 | 0.30 | 5.2 | 0.34 | 0.19 |
| +100..... | 3.6 | 0.50 | 0.33 | 4.3 | 0.38 | 0.19 |
| −100..... | 5.7 | 0.55 | 0.36 | 6.3 | 0.55 | 0.32 |
| Totals..... | 100.0 | 0.33 | 0.23 | 100.0 | 0.24 | 0.16 |
| Original tailings | ..... | 0.30 | 0.13 | ..... | 0.24 | 0.15 |

This fact points to the importance of studying carefully the gangue of an ore, even if it may not be acid consuming. The ability of argillaceous and some colloidal material to absorb and retain mineral salts is well known and these experiments have shown it to be quite a factor in leaching raw oxidized porphyry ores with sulphuric acid.

An effort was made to determine the rate of extraction of copper from the ore by the several lixiviants. To do this the ore and lixiviant were sampled every 2 hr. The ore samples were taken from the top of the charge and the results obtained from them were not entirely satisfactory, but were the best obtainable.

The first charge tested in this way was Charge 20, Test 2. Two strong lixiviants were used on this charge—the first containing 14.2 per cent. acid and the second, 12.7 per cent. acid. The results are shown in the following table and are graphically presented in Fig. 2.

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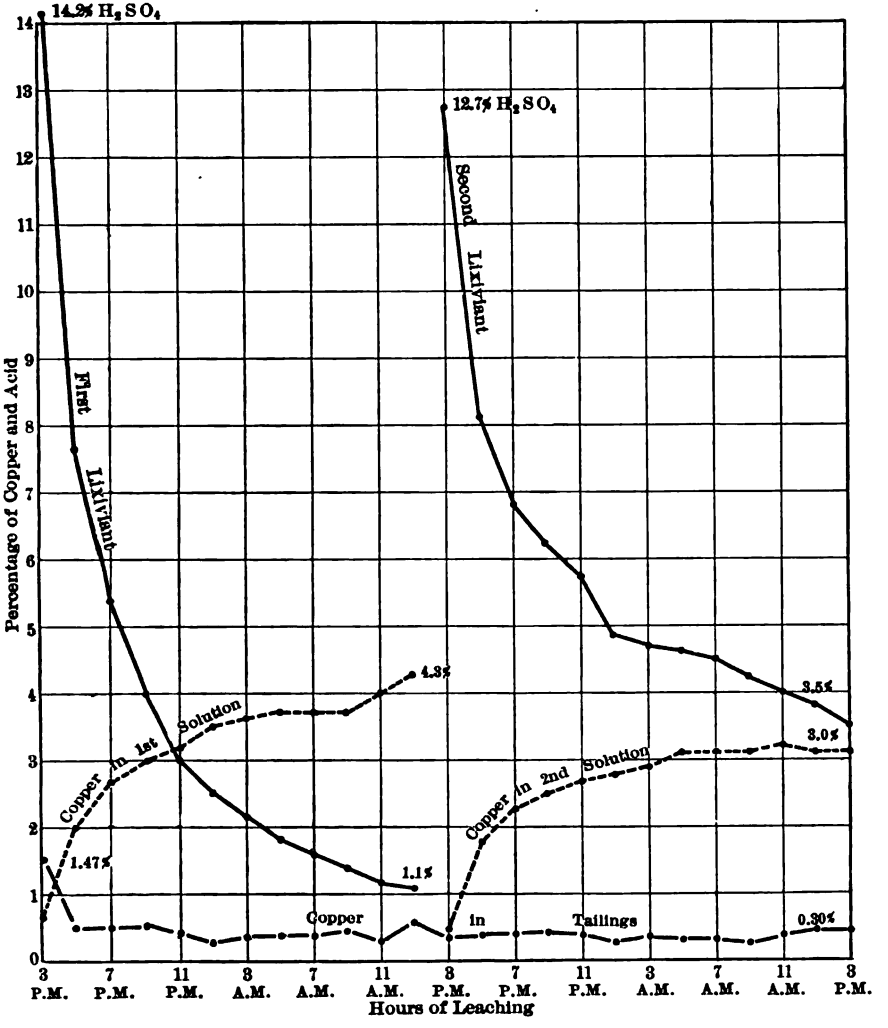


FIG. 2.—RELATIVE VARIATIONS IN COPPER AND ACID CONTENTS OF LIXIVIUM AND ORE, CHARGE 20, VAT *D*.

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*Leaching Experiments on Charge 20, Vat D. Relative Variations in Copper and Acid Contents of Lixivium and Ore*

| Sample Time | Ore Per Cent. Cu | Solution Cu, Per Cent. | Solution $H_2SO_4$, Per Cent. |
|---|---|---|---|
| First Lixiviant | | | |
| 3 p.m. | 1.47 | 0.7 | 14.2 |
| 5 p.m. | 0.47 | 2.0 | 7.7 |
| 7 p.m. | 0.55 | 2.8 | 5.4 |
| 9 p.m. | 0.55 | 3.0 | 4.0 |
| 11 p.m. | 0.38 | 3.2 | 3.0 |
| 1 a.m. | 0.28 | 3.5 | 2.5 |
| 3 a.m. | 0.35 | 3.6 | 2.2 |
| 5 a.m. | 0.35 | 3.7 | 1.8 |
| 7 a.m. | 0.35 | 3.7 | 1.6 |
| 9 a.m. | 0.41 | 3.7 | 1.4 |
| 11 a.m. | 0.30 | 4.0 | 1.2 |
| 1 p.m. | 0.59 | 4.3 | 1.1 |
| Second Lixiviant | | | |
| 3 p.m. | 0.31 | 0.4 | 12.7 |
| 5 p.m. | 0.31 | 1.7 | 8.1 |
| 7 p.m. | 0.32 | 2.2 | 6.8 |
| 9 p.m. | 0.32 | 2.5 | 6.2 |
| 11 p.m. | 0.28 | 2.7 | 5.7 |
| 1 a.m. | 0.32 | 2.9 | 4.8 |
| 3 a.m. | 0.28 | 2.8 | 4.7 |
| 5 a.m. | 0.27 | 3.1 | 4.6 |
| 7 a.m. | 0.25 | 3.1 | 4.5 |
| 9 a.m. | 0.35 | 3.1 | 4.2 |
| 11 a.m. | 0.29 | 3.2 | 4.0 |
| 1 p.m. | 0.29 | 3.1 | 3.8 |
| 3 p.m. | 0.34 | 3.0 | 3.5 |

The next test was made on Charge 38, Test 4. This is more complete and shows the relative variations in the copper and acid contents of lixiviants and ore and also the quantities of iron and alumina passing into solution.

The first lixiviant used here was a partly neutralized one from a previous charge. This was followed by two new lixiviants, each containing 10 per cent. acid.

| Time of Sample | Per Cent. Cu in Ore | Lixiviums $H_2SO_4$ Per Cent. | Cu, Per Cent. | $Al_2O_3$ Per Cent. | Fe (Total) Per Ct. | Fe (Ferric) Per Ct. | Fe (Ferrous) Per Ct. |
|---|---|---|---|---|---|---|---|
| First Lixiviant | | | | | | | |
| 1 p.m. | 1.30 | 4.00 | 1.40 | ..... | ..... | ..... | ..... |
| 3 p.m. | 0.99 | 1.99 | 2.06 | 0.66 | 0.83 | 0.50 | 0.33 |
| 5 p.m. | 1.02 | 1.39 | 2.27 | 0.70 | 0.80 | 0.41 | 0.39 |
| 7 p.m. | ..... | 0.81 | 2.40 | 0.74 | 0.82 | 0.44 | 0.38 |
| 9 p.m. | 1.10 | 0.34 | 2.52 | 0.74 | 0.80 | 0.40 | 0.40 |

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| | | | | | | |
|---|---|---|---|---|---|---|
| Second Lixiviant ..... | 10.31 | 0.90 | 0.57 | 0.62 | 0.49 | 0.13 |
| 11 p.m..... 0.56 | 6.52 | 1.68 | 0.57 | 0.63 | 0.30 | 0.33 |
| 1 a.m..... 0.61 | 4.74 | 2.18 | 0.61 | 0.72 | 0.38 | 0.34 |
| 3 a.m..... 0.53 | 3.51 | 2.40 | 0.65 | 0.71 | 0.35 | 0.36 |
| 5 a.m..... 0.64 | 3.02 | 2.48 | 0.67 | 0.72 | 0.33 | 0.39 |
| 7 a.m..... 0.36 | 2.58 | 2.52 | 0.61 | 0.86 | 0.45 | 0.41 |
| 9 a.m..... 0.35 | 2.13 | 2.54 | 0.77 | 0.85 | 0.43 | 0.42 |
| 11 a.m..... 0.38 | 1.78 | 2.61 | 0.79 | 0.81 | 0.42 | 0.39 |
| 1 p.m..... 0.36 | 1.48 | 2.69 | 0.87 | 0.76 | 0.34 | 0.42 |
| 3 p.m..... 0.28 | 1.32 | 2.78 | 0.77 | 0.87 | 0.43 | 0.44 |
| 5 p.m..... 0.26 | 1.01 | 2.90 | 0.87 | 0.89 | 0.38 | 0.51 |
| 7 p.m..... 0.28 | 0.82 | 2.99 | 0.89 | 0.88 | 0.38 | 0.50 |
| 9 p.m..... 0.29 | 0.72 | 3.00 | 0.94 | 0.89 | 0.40 | 0.49 |
| Third Lixiviant ..... | 10.00 | 0.40 | ..... | 0.42 | 0.16 | 0.26 |
| 11 p.m..... 0.32 | 6.35 | 1.38 | 0.58 | 0.60 | 0.28 | 0.32 |
| 1 a.m..... 0.47 | 4.70 | 1.84 | 0.62 | 0.67 | 0.31 | 0.36 |
| 3 a.m..... 0.29 | 3.88 | 2.02 | 0.67 | 0.70 | 0.29 | 0.41 |
| 5 a.m..... 0.38 | 3.44 | 2.03 | 0.76 | 0.74 | 0.33 | 0.41 |
| 7 a.m..... ..... | 3.06 | 2.32 | 0.77 | 0.83 | 0.42 | 0.41 |
| 9. a.m..... 0.28 | 2.68 | 2.40 | 0.87 | 0.87 | 0.46 | 0.41 |
| 11 a.m..... 0.31 | 2.30 | 2.47 | 0.89 | 0.88 | 0.47 | 0.41 |
| 1 p.m..... 0.24 | 2.00 | 2.54 | 0.93 | 0.96 | 0.53 | 0.43 |
| 3 p.m..... 0.29 | 1.80 | 2.54 | 0.96 | 0.98 | 0.43 | 0.55 |
| 5 p.m..... 0.35 | 1.64 | 2.54 | 0.99 | 1.00 | 0.49 | 0.51 |
| 7 p.m..... 0.32 | 1.48 | 2.54 | 1.03 | 1.04 | 0.51 | 0.53 |
| 9 p.m..... 0.33 | 1.44 | 2.54 | 0.98 | 1.09 | 0.55 | 0.54 |

The percentage of copper dissolved from the ore by the several lixiviants in this experiment, as accounted for by the copper in solution, was as follows:

| | Hours on Ore | Pounds Copper Taken into Solution | Per Ct. of the Total Copper Extracted |
|---|---|---|---|
| First lixiviant..... | 8 | 31 | 23.7 |
| Second lixiviant..... | 24 | 34 | 25.9 |
| Third lixiviant..... | 24 | 51 | 38.9 |
| Wash water..... | 7 | 15 | 11.5 |
| Totals..... | 63 | 131 | 100.0 |

The lixiviants slowly dissolved iron and alumina from the ore, reaching a maximum of about 1 per cent. of each constituent. About one-half of the iron was in the ferric condition, which is of value in dissolving copper as has already been explained.

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These results are graphically shown by Fig. 3.

In the foregoing tests the acid consumption was nearly 3.5 lb. instead of under 3 lb. as obtained in the laboratory experiments. While this was undoubtedly due to the more decomposed ore used in the larger experiments, yet it might be possible to secure a selective action of the acid for the copper by using weaker lixiviants and leaching the ore a longer time. If this is true, less iron and alumina would pass into solution and less acid would be consumed.

The reject from the charge samples of Tests 1, 2, 3, and 4 amounted

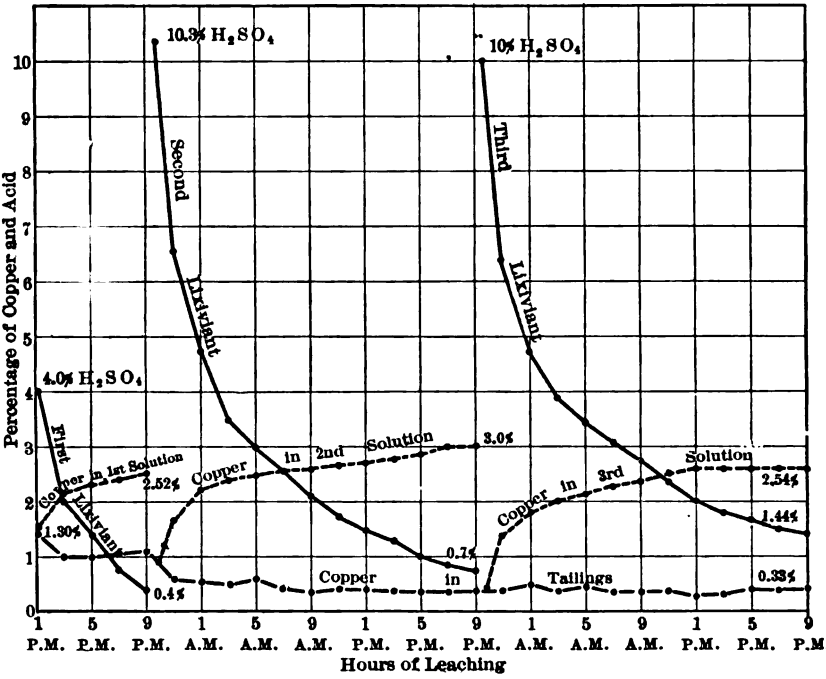


FIG. 3.—RELATIVE VARIATIONS IN COPPER AND ACID CONTENTS OF LIXIVIUM AND ORE, CHARGE 38, VAT D.

to 30,345 lb. In order to test the advisability of leaching the ore with weaker acid and taking more time for the operation, this reject was treated in the experimental plant with an average of 5.25 per cent. acid lixiviant. The first lixiviant used on each charge was the partly neutralized lixiviant from a previous charge that had been treated with a 10 per cent. acid, and already contained 1.16 per cent. copper. Each charge was also saturated with water before putting on the lixiviant, to determine if this would effect any saving in acid.

The results were:

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| Vat. No. | Charges | | | Tailings | | | Extraction | | | Time of Leaching, Hours | Lixiviant Changed Every | Per Cent. Acid | | Lb. Acid Used per Lb. Cu Dissolved |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Weight, Lb. | Cu, Per Cent. | Cu, Lb. | Total Cu, Per Cent. | Sol. Cu, Per Cent. | Total Cu, Lb. | Total Cu, Lb. | Cu, Lb. per Ton | Per Cent. | | | Going On | Going Off | |
| A..... | 6,020 | 1.42 | 85 | 0.72 | 0.60 | 43 | 42 | 14.0 | 49.5 | 96 | 24 hr. | 5.50 | tr. | 3.0 |
| B..... | 7,825 | 1.40 | 109 | 0.32 | 0.23 | 25 | 84 | 21.5 | 77.0 | 120 | 24 hr. | 5.20 | tr. | 2.0 |
| D..... | 16,500 | 1.76 | 290 | 0.40 | 0.36 | 66 | 234 | 28.4 | 77.2 | 144 | 24 hr. | 5.10 | tr. | 2.0 |
| Total... | 30,345 | 1.60 | 484 | 0.44 | 0.37 | 134 | 350 | 23.0 | 72.3 | 120 | ..... | 5.25 | tr. | 2.1 |

The relative percentages of copper, iron, and alumina dissolved by the several lixiviants on each charge are shown below. Owing to the incomplete extraction and shorter time of leaching on Vat *A*, those results are not comparable and are omitted.

Vat *B*

| Extracted by | Hours on Ore | Per Cent. Acid | | Relative Percentages Dissolved | | |
|---|---|---|---|---|---|---|
| | | Going On | Going Off | Cu | Fe | $Al_2O_3$ |
| First lixiviant..... | 24 | 5.60 | trace | 1.0 | 0.0 | 0.0 |
| Second lixiviant..... | 24 | 5.40 | trace | 20.3 | 31.9 | 17.6 |
| Third lixiviant..... | 24 | 5.40 | trace | 22.1 | 21.3 | 18.9 |
| Fourth lixiviant..... | 24 | 4.79 | trace | 28.3 | 21.3 | 29.7 |
| Fifth lixiviant..... | 24 | 4.79 | trace | 27.3 | 25.5 | 33.8 |
| | | 5.20 | | 100.0 | 100.0 | 100.0 |

Vat *D*

| Extracted by | Hours on Ore | Per Cent. Acid | | Relative Percentages Dissolved | | |
|---|---|---|---|---|---|---|
| | | Going On | Going Off | Cu | Fe | $Al_2O_3$ |
| First lixiviant..... | 24 | 5.60 | trace | 1.0 | 0.0 | 0 |
| Second lixiviant..... | 24 | 5.40 | trace | 16.4 | 18.2 | 8.8 |
| Third lixiviant..... | 24 | 5.40 | trace | 19.0 | 10.2 | 15.5 |
| Fourth lixiviant..... | 24 | 4.79 | trace | 21.9 | 21.6 | 26.3 |
| Fifth lixiviant..... | 24 | 4.79 | trace | 20.9 | 21.6 | 25.5 |
| Sixth lixiviant..... | 24 | 4.79 | trace | 20.2 | 28.4 | 23.9 |
| | | 5.10 | | 100.0 | 100.0 | 100.0 |

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The ore being saturated with water in each of these charges, each lixiviant put on represented one volume of solution, or enough to fill the interstices and cover the charge. The results in each case show that the first lixiviant, containing 5.60 per cent. acid, was but little more than enough to destroy the alkalinity of the ore and very little of the copper was dissolved. In fact, the first lixiviant coming from Vat *A* contained less copper than it did going on, showing that the acid was not sufficient to destroy the alkalinity of the ore and some of the copper was reprecipitated in the ore. This confirms the results already obtained in the experimental plant, where there was usually very little increase and sometimes a considerable decrease in the copper contents of the first lixiviant if the free acid and ferric sulphate in solution were not sufficient to destroy the alkalinity of the ore. Therefore it is a waste of time and expense to use lixiviants too weak to accomplish this purpose and to bring about a maximum solution of copper with the first lixiviant.

On the other hand, where a 10 per cent. or stronger acid was used, 75 or 90 per cent. of the total copper dissolved can be extracted by the first lixiviant in 36 to 48 hr.

Each succeeding lixiviant dissolved about the same amount of copper and a corresponding amount of iron and alumina. As in the laboratory tests, iron dissolved relatively as rapidly as the copper while the alumina dissolved in increasing proportions with each succeeding lixiviant.

The final lixiviants on each charge were made from new acid.

That the extraction was not completed, even at the end of the sixth day on Vat *D*, is shown by the complete neutralization of the acid in the final lixiviums and no decrease in their copper contents. Had the leaching been continued to the complete extraction of the copper, the increasing percentage of alumina, together with the iron passing into solution, would no doubt have increased the acid consumption to approximately that already obtained in the experimental plant.

It was found inadvisable to saturate the ore with water before applying the first lixiviant. This water must, of course, be replaced by the acid before the copper can be dissolved and the interchange takes place but slowly through the pores of the ore.

The acid lixiviants should be applied to the dry ore.

### *Summary of Operations*

Analysis of a composite sample of the five lots of ore treated gave the following results:

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| | Calculated from Five Samples, Per Cent. | Composite Sample, Per Cent. | | Calculated from Five Samples, Per Cent. | Composite Sample, Per Cent. |
|---|---|---|---|---|---|
| $SiO_2$ | 64.27 | 63.29 | Cu, total | 1.44 | 1.45 |
| Fe | 4.18 | 4.20 | Cu sol. in 10 per cent. $H_2SO_4$ | 1.31 | 1.32 |
| $Al_2O_3$ | 14.73 | 14.30 | $CO_2$ | 1.30 | 1.26 |
| CaO sol. in acid | 0.63 | 0.90 | | Ounce | Ounce |
| MgO | ..... | 0.80 | Au | 0.01 | 0.01 |
| MnO | ..... | 0.14 | Ag | 0.18 | 0.18 |
| S, total | 0.32 | 0.27 | | | |
| S as sulphate | 0.05 | 0.10 | | | |

The average grade of this entire quantity of ore was much lower than anticipated and lower than that used for the laboratory experiments.

All of the ore treated passed through a 2-mesh screen except the oversize from Test 1, which amounted to 9,818 lb. This oversize ranged between 0.5 and 1 in. in size and was treated by itself as a special experiment to conform more closely to the practice of the Arizona Copper Co. at Clifton.

The first charge was leached only 20 hr. and gave an extraction of 51.7 per cent. The second charge was leached 39 hr. and gave an extraction of 71 per cent., making an average leaching time of about 30 hr. and yielding an average extraction of 61.3 per cent. If the time of leaching had been prolonged to 70 hr., corresponding to that of the 2-mesh material, the extraction of the whole lot would undoubtedly have been as good as that of the 2-mesh material alone, for by combining the results, we have:

| Size of Material, Inches | Per Cent., Weight | Extraction: Lb. Cu per Ton | Extraction: Per Cent. |
|---|---|---|---|
| 1 to 0.5 | 7.25 | 21.6 | 61.3 |
| 0.5 and finer | 92.75 | 24.4 | 80.0 |
| Total | 100.00 | 24.2 | 78.4 |

Therefore in all probability the ore can be leached successfully if not crushed finer than will pass a 0.75-in. screen, which will avoid much of the fine material and slime.

The ore in Test 3, on the other hand, nearly all passed a 4-mesh screen. The fine material produced in crushing to this size very seriously interfered with the percolation of the lixiviants, and washing the ore; consequently it caused loss of time and gave low extractions.

Had the entire quantity of ore treated been thoroughly mixed and crushed to a uniform size, the results would have been better than the average of the results obtained by treating each lot separately, so the

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average results given below are probably not the best obtainable and are certainly on the safe side.

*Screen Analysis*

| Mesh | Per Cent. |
|---|---|
| +4 | 19 |
| +8 | 25 |
| +16 | 18 |
| −16 | 38 |
| | 100 |

| | |
|---|---|
| Total ore treated, pounds | 384,784 |
| Copper in ore, per cent | 1.43 |
| Total copper in ore, pounds | 5,526 |
| Available copper in ore (soluble in $H_2SO_4$ and lixiviant), pounds | 5,097 |
| Time of leaching, hours | 80 |
| Time of washing, hours | 10 |
| Copper in tailing, per cent | 0.32 |
| Total copper in tailing, pounds | 1,228 |
| Soluble copper in tailing, per cent | 0.21 |
| Total soluble copper in tailing, pounds | 799 |
| Copper in tailing insoluble in $H_2SO_4$, pounds | 429 |
| Total copper extracted from ore, pounds | 4,298 |
| Total copper extracted from ore, per cent | 77.8 |
| Copper accounted for in solution including that from oversize in Test 1, pounds | 4,269 |
| Copper accounted for in solution including that from oversize in Test 1, per cent | 97 |
| Total copper extracted from ore based on weight of tailing at 97 per cent. of ore, pounds | 4,336 |
| Total copper extracted from ore based on above weight of tailing, per cent. | 78.5 |
| Soluble or available copper extracted from the ore, per cent | 84.3 |
| Soluble or available copper extracted from the ore based on above weight of tailing, per cent | 84.7 |
| Copper extracted per ton of ore, pounds | 22.34 |
| Copper extracted per ton of ore based on above weight of tailing, pounds | 22.50 |
| Average strength of acid used, per cent | 8.9 |
| Acid used (100 per cent. $H_2SO_4$) per pound of copper dissolved, based on the difference in free acid in the lixiviants going on and going off each charge, as found by analyses, pounds | 3.6 |
| Acid used (100 per cent. $H_2SO_4$) per ton of ore on above basis, pounds. | 80 |
| Acid used (100 per cent. $H_2SO_4$) per pound of copper dissolved, based on the total combined acid found by analyses in all the solutions sent to precipitating tanks, wash waters, and lixiviants returned to storage, pounds | 3.15 |
| Acid used (100 per cent. $H_2SO_4$) per ton of ore on above basis, pounds. | 67 |
| Lixiviants used per ton of ore, gallons | 128 |
| Wash water used per ton of ore, gallons | 268 |
| Solution required to cover 1 ton of ore after saturation, equals one volume, gallons | 34 |
| Rate of percolation through ore, inches per hour | 78 |
| Rate of percolation per square foot of filter area, gallons per hour | 10 |

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*Summary of Construction*

Of the two methods recommended for waterproofing concrete, viz., by the use of "Toxement" and by the use of crude oil, the concrete mixed with crude oil appeared to be less attacked by the acid solutions at the end of two months' service than that mixed with "Toxement," although the difference was scarcely noticeable.

Both methods require a much longer time for the concrete to set than when neither is used.

So far as could be observed, the acid-resisting qualities of concrete and plaster prepared with either "Toxement" or oil were not any better than those of ordinary concrete and cement plaster made with siliceous sand and gravel.

Acid-resisting paint known as "R. I. W. No. 89" prepared especially for this work was of no value whatever in protecting the concrete vats from the 10 per cent. sulphuric solutions, the weaker lixiviants in the leaching vats, or the neutralized copper sulphate solutions in the precipitating tanks.

Asphalt applied hot or as a paint was likewise useless as a protective covering for the cement plaster under any of the above conditions. One difficulty in retaining this on the walls of the vats was the high temperature of the water used for washing the ore, which averaged 100° F. or more.

The use of such warm wash water may be criticised and would have been changed under ordinary conditions, but there was a possibility of the water supply coming from hot springs or wells, if developed near the mine, so it was thought best to put the asphalt coverings to the severest test. Besides, a glance at the atmospheric temperatures during the summer months, as given in the beginning of this article, is sufficient to show that even this part of the experimental work required some care in manipulation. Asphalt that would soften and run from joints between bricks or from walls under the heat of the sun, might be brittle enough to crack at night and allow the acid solutions to get at the cement behind it.

With the exception of Vat *A*, all the leaching vats, acid and precipitating tanks may be considered as operated for the entire two months on the cement plaster alone, for at no time did the protective covering last more than 24 to 48 hr. on the leaching vats, and on the acid and precipitating tanks the paint would come off in patches after a few hours' or a few days' exposure to the solutions. At the end of two months the "Toxement" plaster, ½ in. thick, was disintegrated to the concrete in places on Vat *B* and the oil-mixed plaster of the same thickness was disintegrated nearly to the concrete on Vat C. In the acid storage tanks both the "Toxement" and oil-mixed plaster were dissolved in patches by the 10 per cent. acid solutions and the solutions leaked through the

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8-in. concrete walls. These holes were patched with ordinary cement and sand plaster and also with straight cement, both of which held the solutions successfully for several weeks.

One of the storage tanks was replastered with cement and sand and thoroughly coated with paraffine, which was forced into the plaster with a hot iron. When filled with dilute acid lixiviant, the paraffine came off immediately and was of no value whatever as a protective covering.

A plaster, recommended by the Standard Oil Co., was also tried on the acid storage tanks. It was composed, by weight, of 10 per cent. litharge, 20 per cent. short fiber asbestos, and 70 per cent. sand. These were mixed into a mortar with 40° silicate of soda and applied in the usual manner.

This plaster immediately disintegrated on filling the tank with water. The tank was then replastered with the same materials and filled with dilute acid. So long as the plaster remained covered with acid, it was satisfactory protection, but on exposure to air it began to disintegrate.

Some of the leaching vats were plastered with ordinary cement and sand mortar and painted with several coats of crude petroleum. Each application of oil was allowed to soak into the cement and become perfectly dry before another coat was applied. This was done at the close of my experiments and I had no opportunity to test its merits.

The brick lining in Vat *A* proved to be very satisfactory. With the exception of where the asphalt was melted from the joints by the hot wash water, no deterioration could be noticed. This was due to the irregular size and shape of the common brick used for the lining and could be avoided by using pressed brick of uniform size which would permit of thin joints. In a large vat the brick would also be laid flat instead of on edge.

The sand and asphalt bottoms were entirely satisfactory when they were pressed solid with a heavy hot iron muller like that used in street paving. If this was not done, they were porous and spongy, and no protection against the solutions.

As now developed by the limited operation of the experimental plant, ordinary or oil-mixed concrete will be entirely satisfactory for leaching vats, wash-water and neutralized copper sulphate solution tanks. These can be made tight and acid resisting by lining with pressed brick of uniform size, laid in asphalt and backed with hot asphalt poured between the brick and concrete. Vitrified brick would be less porous and preferable if they can be obtained straight and uniform in size. The vats should be plastered inside with cement and sand mortar before putting in the brick lining.

The cement we used in construction came from the regular run of

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the mill at El Paso and contained more or less free lime. It is my belief that with cement and other materials containing *no* free lime, a concrete vat can be built and lined with brick, laid with thin joints of cement and sand mortar, that will be entirely satisfactory for acid leaching. After the brick soak full of mineral salts, there is very little if any transfusion of the corrosive solutions to the concrete back of the lining and they form a protective coating for both ore and solutions.

Oil-mixed cement plaster will last two or three months without patching or renewal. A brick lining laid in oil-mixed cement mortar would probably last longer than plaster, but the asphalt and brick lining is recommended if it can be held in place.

The cracks that form after construction are the most serious, if not a fatal defect in concrete for leaching vats. These are often so small that they are scarcely noticeable and yet are sufficient to start a leak that is almost impossible to stop even with asphalt itself.

The shape of the vats is immaterial, although for economy of construction and operation a rectangular concrete vat is preferable. The tailings can then be removed from the top, and the bottom of the vat made solid with no openings.

The bottoms of the vats should be paved with sand and asphalt, thoroughly pressed down with hot iron mullers or otherwise to make a *compact* covering. On top of this should be a brick paving laid in asphalt.

The storage or acid mixing tanks for the 5 to 10 per cent. acid solutions should be made of steel or wood and lined with lead.

The hard-lead centrifugal pumps used for circulating the solutions showed no wear except the steel shafts. It was difficult to keep packing tight enough around the shafts to prevent the leakage of solutions, and at the same time not crack the lead casing or stuffing gland. The shafts on all the pumps were worn out in two months and were replaced with bronze. The wear on the bronze shafts is undetermined. The repairs on the pumps will not be a serious matter since new casings, runners and shafts are easily made.

Low-pressure air-lift pumps offer more advantages, on account of freedom from moving parts in the corrosive solutions, and are recommended.

The pure-rubber acid-proof hose used for the transfer of solutions showed no deterioration at the end of two months. While the initial cost is high, it is the only hose that will resist the corrosive solutions and it is indispensable for handling solutions from the pumps on top of the vats.

Lead-lined iron pipe is recommended for the rigid pipe connections and wooden, lead-lined or concrete launders provide the simplest means for the transfer of solutions from one leaching vat to another.

The tailings will have to be removed from the vats by machinery. No

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portion of them will run out by gravity unless they contain a high percentage of water.

Since the tailings will be discharged containing small percentages of free acid and copper sulphate, the kind of metal used for discs, plows or buckets in the excavating machinery was given consideration, although it is improbable that the quantity of acid or copper sulphate in the tailings will be sufficient to prohibit the use of cast iron, steel, or perhaps a high-silicon iron which has high acid-resisting properties.

Krupp bronze and "Monel" metal were both submitted to the following tests. Pieces of each metal were allowed to stand in acid solutions and in acidified copper sulphate solutions of varying strengths—the bronze for two and one-half and the Monel metal for three months.

The bronze precipitated the copper completely from all but the two strongest solutions. The Monel metal precipitated no copper, but passed into solution to a greater extent than the bronze. The results were as follows:

*With Sulphuric Acid*

| Strength of Acid Per Cent. $H_2SO_4$ | Per Cent. Loss in Weight | |
|---|---|---|
| | Bronze | Monel Metal |
| 0.12 | 3.22 | 3.89 |
| 0.25 | 3.26 | 5.92 |
| 0.50 | 3.77 | 8.86 |
| 0.75 | 3.54 | 7.62 |
| 1.00 | 3.64 | 7.55 |
| 1.50 | 4.55 | 10.07 |

*With Acidified Copper Sulphate Solution*

| Strength of Solution, Per Cent. Cu | Per Cent. Loss in Weight | |
|---|---|---|
| | Bronze | Monel Metal |
| 0.05 | 1.22 | 14.50 |
| 0.10 | 1.22 | 7.20 |
| 0.20 | 0.76 | 4.73 |
| 0.30 | 0.82 | 5.08 |
| 0.40 | 0.10 | 2.17 |
| 0.50 | 0.22 | 7.87 |

The Monel metal was also allowed to stand about six weeks in the following solutions:

| | Loss in Weight, Per Cent. |
|---|---|
| 10 per cent. sulphuric acid | 2.81 |
| 10 per cent. sulphuric acid and 10 per cent. copper sulphate (2.5 per cent. Cu) | 4.88 |
| 10 per cent. copper sulphate (2.5 per cent. Cu) | 1.99 |

A high-silicon iron, under the trade name of "Duriron," was tested in a similar manner by these solutions. So far as could be observed at

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the end of a few weeks this metal was not corroded in the least but the exprimental work was closed before this test was finished. The acid-resisting quality of this alloy is well known but it has the disadvantage of being very hard and brittle.

## Precipitation of the Copper from Solution

I have purposely divided my subject into two parts because there have been almost as many methods proposed for precipitation of the copper from solution as there have for leaching it from the ore, and to-day there is probably more uncertainty and more difference of opinion concerning the methods of precipitation than there is concerning the methods of leaching. Of all the methods proposed there is just one that has been demonstrated a commercial success, and that is precipitation on metallic iron.

Lime or limestone was used in some of the earlier methods of hydrometallurgy, but this gave a precipitate that was little better than the ore itself.

Sulphurous acid came into prominence a few years ago, but it involved difficulties of manipulation that have not been overcome, even in small-scale operations. Hydrogen sulphide, produced from iron matte, has been tried but has not been developed on a large scale and is likely to prove a troublesome reagent under such conditions.

Electrolytic precipitation is always attractive and has probably received more attention and intelligent experimentation from skilled metallurgists than any other method While some small plants use this method from time to time, it cannot, as yet, be called a commercial success.

When working on a metallurgical problem, where the margin is narrow, one must always consider the materials at hand as the cheapest obtainable and most likely to permit profitable operations.

As possible precipitants for the copper, I considered the following in the order named: Natural sulphides of iron, artificial sulphide of iron, sponge iron, pig iron, and electric current.

### *Natural Sulphides of Iron*

If acid leaching is used for the oxidized ore some other process will have to be used for the sulphide ore. It is probable that this would be some form of mechanical concentration.

Tests had already been made by the ordinary methods of wet concentration at the University of Arizona before my arrival. These, by very careful manipulation, gave a recovery of 68 per cent. of the copper, 77 per cent. of the gold and 63 per cent. of the silver, but the concentrates carried only 10.2 per cent. copper from an ore that assayed 2.3 per cent. copper, 0.02 oz. gold and 0.22 oz. silver per ton.

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By a combination of flotation and concentration I was able to raise the recovery to over 76 per cent. and the grade of the concentrates to nearly 14 per cent. copper, although the flotation concentrates themselves assayed nearly 24 per cent. copper. As is often the case when chalcopyrite has its source in igneous rocks, it is intimately associated with magnetite. Even fine crushing to 40 mesh failed to separate the magnetite from the chalcopyrite in this ore and this destroyed the properties of the latter for flotation methods to a great extent, as well as lowered the grade of concentrates. However, by treating these concentrates on a magnetic separator I was able to raise the grade to 18.5 per cent. copper. The magnetite itself carried no value.

This, then, was the product, and probably the best that can be obtained from the sulphide ores by the usual methods of treatment. It must be shipped to the company's smeltery at Douglas for treatment, or it must be smelted into matte and blister copper on the ground. If shipped to the smeltery, it would certainly be advisable to raise its grade if possible either with or without removing the magnetite.

It is generally conceded among geologists that secondary chalcocite if formed by the action of copper sulphate solutions on pyrite. Barring the recognized but unknown intermediate reactions, the final result is supposed to be represented by this reaction:

$$5FeS_2 + 14CuSO_4 + 12H_2O = 7Cu_2S + 5FeSO_4 + 12H_2SO_4$$

While later investigators have found this reaction "incompatible with the actual volume relations observable," there seems to be no doubt about ferrous sulphate and sulphuric acid being formed when the chalcocite is deposited, and, if the above reaction is true, nearly all of the acid originally combined with the copper is regenerated.

The reaction with chalcopyrite might be very simple, producing cupric instead of cuprous sulphide, thus:

$$CuFeS_2 + CuSO_4 = 2CuS + FeSO_4$$

I had therefore a recognized precipitant for copper in the concentrates from the sulphide ores. Its efficiency as a precipitant was important but not essential, since any precipitation of the copper from solution and consequent enrichment of the concentrates would be a distinct gain. The only factor to make it a commercial success was the speed of the reaction and in this it failed. However, having secured sufficiently speedy reactions from other natural sulphides of iron on previous occasions, I am not prepared to say the last word has been spoken in this case.

### *Artificial Sulphides of Iron*

Two sources of this precipitant would be available. First, the matte produced by smelting the concentrates at the mine, and second, the matte

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that could be produced by smelting the low-grade pyrite at Bisbee. If soluble sulphides precipitate cupric sulphide from copper solutions and natural sulphides of iron precipitate cuprous sulphide, it is reasonable to expect the artificial sulphide of iron to do the same.

Earlier experiments had shown me that iron sulphide precipitates metallic copper and not sulphide of copper from solution. This investigation was carried further and I found the reaction to be, not a simple transposition of the metals, but the following:

$$4FeS + CuSO_4 = Cu + FeSO_4 + Fe_3S_4$$

I found also that the precipitation of copper stops at this point, although free sulphuric acid will readily decompose the $Fe_3S_4$ with the liberation of hydrogen sulphide. Therefore less than 25 per cent. of the iron in pure iron sulphide is available for the precipitation of copper and the resulting product cannot carry more than 17.7 per cent. copper.

Increase in the copper contents of a matte decreases its precipitating power so much that a matte containing over 30 per cent. copper precipitates practically no copper from solution. Nevertheless the iron from the matte passes into solution to an appreciable extent. The only explanation I can offer for this phenomenon is a reduction of the cupric sulphate to cuprous sulphate and an oxidation of the cuprous sulphide of the matte to cupric sulphide, according to the following reaction, but I am not prepared to say that this is correct.

$$FeSCu_2S + 2CuSO_4 = Cu_2SO_4 + FeSO_4 + 2CuS$$

I could produce no condition that would make a copper matte precipitate copper from solution within a reasonable length of time.

This method of precipitation was pronounced a failure. For further details see my original paper on this subject.[2]

### *Sponge Iron*

This was first used as a precipitant for copper in England in 1837, although Gossage, in 1859, was the first to use it in connection with the wet extraction of copper from ores. It was made by heating a mixture of finely crushed iron ore and coal in a reverberating furnace, with a reducing flame.

Since then a great deal of sponge iron has been produced by iron manufacturers and a number of furnaces have been invented for its production, all of which are described in books on the metallurgy of iron and steel. It is now produced commercially at Hoganas, Sweden, by heating a mixture of fine magnetic iron concentrates and coal in pots or retorts by means of producer gas.

[2] *Engineering and Mining Journal*, vol. xcvii, No. 15, pp. 745 to 748 (April 11, 1914).



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According to Sir I. Lowthian Bell,[3] oxide of iron is easily reduced by solid carbon, but carbon monoxide gas is greatly preferred as a reducing agent.

The reduction of $Fe_2O_3$ begins at 420° F.

The reduction of FeO begins at 1,300° F. and is complete at 1,475° F.

Sponge iron readily combines with oxygen when exposed to the air at a red heat. It is also capable, at certain temperatures, of splitting up the carbon dioxide formed by its own reduction.

The necessary reaction for the reduction of iron oxide by carbon monoxide seems to be

$$Fe_2O_3 + 9CO = 2Fe + 3CO_2 + 6CO$$

or, that there is as much oxygen in the CO escaping as in the $CO_2$ formed. This reaction is not so simple as it appears, however, for carbon monoxide gas is split into carbon dioxide and carbon in the presence of iron oxide or metallic iron, thus:

$$2CO = CO_2 + C$$

Hydrogen greatly assists in the reduction of iron oxide.

Dr. Frankfurter of the University of Minnesota found that finely pulverized iron ore begins to reduce at about 300° F. in an atmosphere of hydrocarbon gas and is completely reduced below 1,000° F.

S. H. Stupakoff, in a paper read before the Engineering Society of Western Pennsylvania, states that carbon monoxide begins to reduce precipitated iron oxide at 285° F., roasted carbonate at 390° F., and is active on all ores at 750° F. It is most active at 1,000° F.

Solid carbon begins to reduce iron oxide at 800° F.

The reverse action, of $CO_2$ being reduced to CO by metallic iron, begins at 800° to 1,100° F., depending upon mass action, and is most active at 1,475° F.

A mixture of $3CO_2$ and 2CO is oxidizing at 1,000° F. and a mixture of $1CO_2$ and 2CO is oxidizing anywhere above 1,500° F.

My plan for the production of this precipitant was to manufacture sulphuric acid at the leaching plant from the low-grade Bisbee pyrite and then reduce the iron oxide in the calcines to sponge iron for precipitating the copper. The Bisbee pyrite contains about 38 per cent. iron and 1.5 to 2 per cent. copper. The copper would pay for the mining.

The freight rate on this material from Bisbee to Ajo would be very low and divided equally between the acid and precipitating departments. The acid plant would be constructed so as to have the same life as the oxidized ore in the mine. In this way the cost of acid made at the leaching plant should compare favorably with that made at the smelting plant at

[3] *Principles of the Manufacture of Iron and Steel.*

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Douglas, when the cost and maintenance of acid trains and storage equipment are considered in conjunction with the high freight rates on this commodity.

The calcines from this pyrite would be in ideal condition for the production of sponge iron. They would contain a finely divided, porous, artificial oxide of iron which could easily be reduced at low temperatures. The product obtained from the reduction of these calcines would contain 60 to 65 per cent. metallic iron in a finely divided, porous condition that would present the greatest possible surface for the rapid precipitation of copper and the copper contained in the original pyrite would all be recovered.

Laboratory experiments were made by mixing iron ore with 30 to 35 per cent. coal, both crushed to 16 mesh, and heating in a closed clay crucible in an assay muffle for 1 hr. at a temperature of 1,600° to 1,800° F. From 75 to 77 per cent. of the iron in the ore was reduced to metallic iron.

These experiments were repeated by mixing calcines with 25 to 30 per cent. coal and heating in the same manner; 98 per cent. of the available iron was metallized.

Calcines were then placed in an iron tube and heated to a low red heat in an atmosphere of hydrocarbon gas for 1 hr. Over 90 per cent. of the available iron was metallized.

This experiment was repeated and the tube was heated to incipient redness for 90 min.; 93 per cent. of the available iron was metallized. The gas used for heating the tube in this experiment was the impoverished gas coming from the reduction of the calcines.

The available iron in these experiments was the iron existing as oxide and not that existing as sulphide.

These experiments were again repeated on calcines from Bisbee pyrite crushed to ½-in. mesh instead of 16 mesh. At the end of 1 hr. 76.6 per cent. of the available iron was metallized and at the end of 2 hr. 88.3 per cent. was metallized.

Sponge iron has been produced in quantity by iron manufacturers in small reverberatory furnaces and, as I have stated, is now produced commercially in Sweden from magnetite concentrates in intermittent furnaces of the brick or pottery kiln type, so there is no doubt about the metallurgical conditions necessary for its production, nor the feasibility of its production for the precipitation of copper.

The idea of using calcines as a source of iron is new, so far as I know, and was first proposed by me during the course of these experiments in the summer of 1912. The advantages of using this material are at once apparent from every view point.

My efforts, therefore, were not to determine if sponge iron can be produced commerically, but to devise a furnace that will receive the calcines hot from the roasting furnaces, reduce them continuously and

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deliver the metallized product or sponge iron to the precipitating department of the leaching plant. The problem is simply one of mechanical construction with no impossible or prohibitive features. The production of sponge iron is surprisingly easy, when the conditions are right.

After trying several small furnaces of different designs, I erected a Wedge double-function roasting furnace of special design to meet my conditions. This furnace was 9 ft. 9 in. in diameter, had six roasting

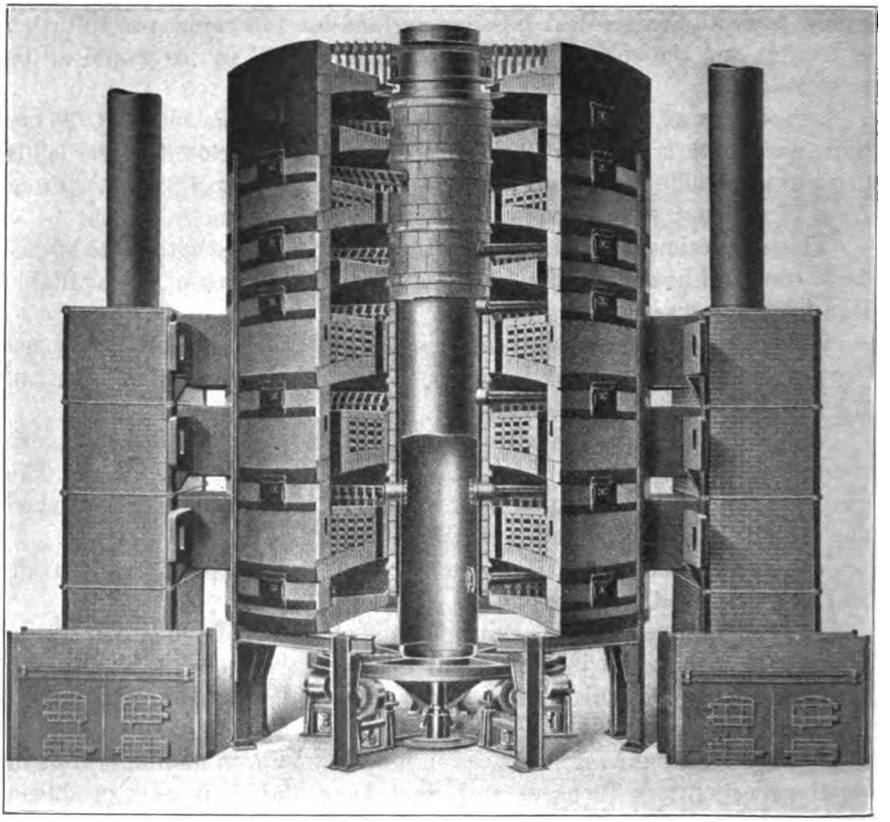

FIG. 4.—WEDGE DOUBLE-FUNCTION ROASTING FURNACE.

hearths, one drying hearth and one cooling hearth. The three upper hearths were of the ordinary type for roasting sulphide ore. The three lower were muffle hearths heated with oil from outside fire boxes. The two sets of hearths were sealed from each other by automatic cast-iron valves. The entire furnace was sheathed with sheet iron except the top and bottom. Luting rings were placed around the center shaft, and every effort was made to make the furnace as air tight as possible. Each hearth was connected with the flue so the gases could be controlled as desired.

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A screw feeder was attached to the side of the furnace to deliver pulverized coal on the upper muffle hearth.

A general but incomplete idea of the furnace may be obtained from Fig. 4.

I expected to be able to roast Bisbee pyrite on the three upper hearths and deliver the red-hot calcines on the top muffle through the automatic valves, where they would be mixed with powdered coal from the feeder. In this way no heat would be lost and reduction would begin at once.

I soon found that I had made a serious mistake in trying to put two furnaces into one, for I had neither a roasting nor a metallizing furnace. With all six hearths for either purpose, I would have obtained better results.

The spaces for the passage of hot gases under the muffles were too small. In order to maintain the temperature that was desired on top of the muffle, the heat directly opposite the fire boxes was high enough to soften the clay tiles and cause them to warp. This allowed the calcines, already reduced to ferrous oxide, to drop into the joints and form a fusible slag which soon made a hole through the muffle and admitted air and products of combustion just where I wanted a reducing atmosphere.

My greatest trouble probably came from air leakage through the bottom or cooling hearth and around the central shaft. The bottom hearth was made of brick arched in the usual manner. It was plastered on the under side with cement, but with the constant contraction and expansion of the furnace it was impossible to make it air tight. I would frequently get metallization well started on the middle muffle hearth and the iron would oxidize again before discharging.

The luting rings around the central shaft, which were supposed to seal by dipping into the ore on the hearth, were not satisfactory for this purpose.

I endeavored to overcome these difficulties by producing a gas pressure inside the furnace from the combustion of the coal. In this I was partly successful, and for short periods of time, until something broke or temperatures fell, I got encouraging results.

The highest temperatures I was able to obtain on the muffles were as follows:

| | Degrees Fahrenheit |
|---|---|
| Top muffle | 1,000 |
| Second muffle | 1,320 |
| Third muffle | 1,460 |
| Cooling hearth | 1,040 |

but these were maintained only a short time before a muffle gave way.

When reducing with coal, I should have had higher temperatures or at least 1,400° on all the muffles. The soft coal used I thought

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had a tendency to deposit soot on the calcines at low temperatures, which retarded the reducing action. I tried lignite coal, but with so many other adverse conditions to contend with, I could not notice any difference. I also tried powdered coke, but at that time other conditions were unfavorable and the test was not completed.

I was anxious to use producer gas, but had no opportunity to do so. My own laboratory experiments and those of all other investigators have proved that iron oxide can be reduced much more easily with gas than with solid carbon, and producer gas made from lignite or oil (which would be the cheapest fuels obtainable at Ajo) would contain the most active reducing agents known for this purpose.

I regret that my work in this direction was stopped before it was carried to a conclusion, but enough was learned to satisfy me that it will be entirely feasible to metallize calcines for copper precipitation at a cost of not over $5 per ton of metallic iron. Our estimated costs at Ajo were not much more than this, and they included half the cost of the pyrite.

I believe a muffle type of furnace can be made practicable for metallizing calcines continuously, but it must be made gas tight. The top and sides should be sheathed with iron and the calcines fed through a double bell as on an iron blast furnace or through a screw feed that would always be kept full. The bottom of the furnace should be made of cast iron or, preferably, water-jacketed segments. The central shaft should be water-sealed top and bottom.

Other material than the 3-in. thick clay tiles used in the Wedge furnace could no doubt be found for the floors of the muffles. It should be of some basic material. If producer gas is used for reduction, it might be possible to make these floors of cast-iron sections, because the temperature of reduction is much lower with gas than with solid carbon.

Producer gas should be used for reduction and the impoverished gas used for heating the furnace. The Mond Nickel Co. has been reducing nickel oxide by means of carbon monoxide for years in a similar type of furnace, without accident or difficulty of manipulation.

Subsequent experiments lead me to believe that a shaft furnace can be developed that will also do the work with much less complicated operation.

In any case the metallized product must be cooled in air-tight hoppers that will seal the furnace, or discharge from the furnace directly into water. The fine iron oxidizes immediately when exposed to the air at a red heat, but if cooled in a reducing atmosphere or in absence of air and kept dry, it will remain unaltered for months. If it is discharged from the furnace into water, it must be kept under water until used. It is difficult to take it from the water and dry it without some oxidation, but it will not oxidize under water within any reasonable time.

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*Pig Iron*

When iron is mentioned for copper precipitation, no one in this country seems to consider anything but scrap iron as available for this purpose. It takes but a small amount of investigation, however, to find that pig iron is not prohibitive in cost at almost any point in the United States not remote from the railroad. It is reported that pig iron can be delivered on the Pacific coast from China or India at a cost of $10 to $12 per ton.

The use of ordinary pig iron requires long launders and more or less handling of the iron to effect complete precipitation of the copper. Of course the rapidity of precipitation depends upon the surface exposed, so I granulated the pig iron I used. This can be done best by shattering a small stream of molten iron with a jet of steam, and then cooling in a stream of water. The product was very hard and compact and most of it was in the form of shot or pear-shaped drops. It contained 93.5 per cent. iron. A screen analysis gave the following results:

| Mesh | Individual Percentages | Cumulative Percentages |
|---|---|---|
| +4 | 2.0 | 100.0 |
| +8 | 20.0 | 98.0 |
| +16 | 31.0 | 78.0 |
| +30 | 28.6 | 47.0 |
| +60 | 12.4 | 18.4 |
| −60 | 6.0 | 6.0 |

The relative precipitating values of the different sizes of this product were obtained by treating equal weights of each size with equal quantities of pure copper sulphate solution and also with lixivium obtained from leaching the oxidized ore. Each test covered the same period of time, at the same temperature, and was made under the same conditions

| Mesh | Copper Sulphate Solution | | Lixivium | |
|---|---|---|---|---|
| | Grams Cu Precipitated | Relative Speed of Precipitation | Grams Cu Precipitated | Relative Speed of Precipitation |
| +4 | 0.050 | 1.0 | 0.06 | 1.0 |
| +8 | 0.105 | 2.1 | 0.11 | 1.8 |
| +16 | 0.385 | 7.7 | .... | .... |
| +30 | 0.540 | 10.8 | 0.59 | 9.8 |
| +60 | 0.925 | 18.5 | 1.05 | 17.5 |
| −60 | 1.175 | 23.5 | 1.31 | 21.8 |

At the Gumeshevesky mine in Russia 12 tons of granulated iron is reported to have the same precipitating capacity as 120 tons of iron plates.

The method of using granulated iron as a precipitant was also a

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small problem. Sponge iron from calcines or even from iron ore is attacked inside and out by the copper solutions and each particle, large or small, is soon a mass of cement copper. Granulated iron is attacked only on the surface and unless agitated continually during precipitation it soon cements together into a solid mass which retards further action.

To overcome this difficulty, I made a tube mill out of an iron pipe 10 ft. long and 20 in. in diameter. It was fitted with 6-in. openings at each end for the admission and discharge of solution. It was filled to these openings with granulated iron and revolved at the rate of 12 rev. per minute.

The solution going into this tube mill was a neutralized lixivium from leaching the oxidized ores. It contained

| | Per Cent. |
|---|---|
| Cu | 1.64 |
| Fe | 0.34 |
| Free acid | trace |

It was passed through at different rates of speed to get the precipitating capacity of the mill. The results obtained from the solution passing out of the mill were as follows:

| Rate, Gallons per Minute | 4 | 6 | 7 | 10 | 12 |
|---|---|---|---|---|---|
| Cu, per cent. | none | none | none | 0.06 | 0.09 |
| Fe, per cent. | ..... | ..... | 1.98 | 1.88 | 1.84 |
| Free acid, per cent. | ..... | ..... | trace | trace | trace |
| Iron (100 per cent.) pounds consumed per pound copper precipitated | ..... | ..... | 1.00 | 0.97 | 0.97 |
| Granulated iron, pounds consumed per pound copper precipitated | ..... | ..... | 1.07 | 1.04 | 1.04 |

The copper precipitate contained 73.6 per cent. copper. In a previous experiment, it contained 86.8 per cent. copper.

By this method of precipitation the iron remained in the mill, always bright and clean, while the precipitated copper passed out with the neutral solution, from which it settled rapidly and could be removed by decantation. The operation is continuous.

As seen by these results, the consumption of iron need not exceed 1 lb. for each pound of copper precipitated, provided the solutions are neutralized before precipitation.

In a large plant these mills should be made longer and of less diameter than an ordinary tube mill. The shell should be made of copper and lined with silex, to prevent abrasion. The galvanic current set up by the copper and iron would hasten the precipitation.

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*Electrolytic Precipitation*

As usual, before starting a series of experiments, I compiled all the data available on this subject. These were obtained not only from published works, but from private notes and correspondence.

The vital factors governing this method of precipitation are cost of power, material for anodes, and interference of other metals.

The power required for deposition from clean copper sulphate solutions is about eight or ten times that required for refining purposes, or about 1 kw. per pound of copper deposited. The most economical current density is between 10 and 11 amperes per square foot and the voltage required is about 2 volts, although with magnetite anodes it is said a current density of 15 amperes can be used. With an average cost of electric power of not more than 1c. per kw. the power costs for this method of precipitation need not be prohibitive.

The anode material heretofore has been confined to lead or antimonial lead. Fused magnetite is now being tried and is said to be very satisfactory, although fragile and expensive. Its merits are yet to be demonstrated on a large scale. Lead anodes are gradually consumed, forming the peroxide of lead, which can be collected, reduced and used over again if necessary.

Interference of other metals has proved the chief stumbling block in most of the attempts to use the electrolytic method for the recovery of copper. Arsenic and antimony are troublesome, but are seldom found in appreciable quantities in leachable ores. Iron is the principal source of trouble and with it may be considered manganese if present, for its influence is substantially the same as iron. Iron, by its alternate oxidation and reduction, consumes electrical energy without the deposition of copper. It must, therefore, be rendered innocuous by one of the following methods: Removal by precipitation, the use of diaphragms, or the use of depolarizers.

Removal by precipitation on a large scale may be discarded at once as impracticable, for any effort in this direction would enable one to precipitate the copper direct by means of the same amount of chemical reagents and at about the same expense.

The use of diaphragms has been tried repeatedly, but never successfully, on long-continued, large-scale operations. Under the conditions involved, it is improbable that they can ever be made to operate successfully, for diaphragm material that does the work required, increases the electrical resistance beyond commercial limits.

The use of a depolarizer seems to offer a way out of this difficulty. The most feasible depolarizer and perhaps the only one sufficiently cheap for this purpose is sulphur dioxide. This has some advantages and some serious disadvantages. Theoretically, sulphur dioxide used in this

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manner will regenerate 3 lb. of acid for each pound of copper deposited. In experimental practice, 2 lb. has been the maximum obtained, owing to difficulties of manipulation. A certain amount of electromotive force is also generated in the direction of the current used, which reduces power consumption. These advantages, however, are more apparent than real. If much iron and alumina pass into solution from the ore, the actual acid regenerated in the electrolytic cells is relatively too small to be of any importance. On these Ajo ores, granting the best possible regeneration of acid from electrolytic cells, I estimated that 75 per cent. of the acid used will have to be made in an acid plant. In actual practice it would no doubt exceed this amount.

The mechanical difficulties in making the copper lixiviums absorb sufficient sulphur dioxide gas to produce the desired results have not been solved. Introduction of the gas into the electrolytic cells at the anode secures a very small absorption and renders the electrolytic plant uninhabitable for human beings. Acid makers of experience do not offer much hope of success for absorption of this gas by means of scrubbing towers.

Those who have tried to make sulphur dioxide act as a depolarizer on a commercial scale report these claims a fallacy.

Moreover, it has been found that when electrolytes containing sulphur dioxide are used the current density must be reduced to the neighborhood of 3 amperes per square foot in order to get a satisfactory deposition of the copper. This means an electrolytic plant installation three times the size ordinarily required.

It has been found impracticable to deposit copper from electrolytes containing less than 1 per cent. copper. The average lixivium including enough wash water and acid to maintain a standard electrolyte will average between 1.5 and 2 per cent. copper. Assuming then, for the sake of argument, that 35 to 50 per cent. of the copper can be deposited in good form by electrolysis (35 per cent. is actually the case in the only plant in operation), if relatively large quantities of iron and alumina pass into solution at each cycle of the lixiviant it will be necessary to discard the entire lixiviant in a short time or a portion of it at each cycle, in which the copper will have to be precipitated by a chemical reagent. This discarded lixivium will either be strongly acid or, if neutralized by fresh ore, will carry its full quota of copper. In any leaching operation, there will be also a gradual accumulation of wash water or weak solutions, which will have to be treated with a chemical precipitant.

The conditions governing the successful deposition of copper from solution by electrolysis have been determined quite conclusively after long and careful experimentation by men skilled in the art.

With ores that will yield lixiviums free from interfering elements, this method will no doubt prove a commercial success in spite of its expensive

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installation, but with ores like these where the iron passes into solution as readily as the copper, when the raw ore is leached with acid, it is almost a mathematical certainty that electrolytic precipitation cannot be made a commercial success and further experimentation along this line is useless.

A much more feasible method of overcoming this difficulty with iron would be to keep it from going into solution in the first place.[4] This might be done by roasting the ore after crushing to the size required for leaching. The expense would be small. While there might be some danger of forming insoluble ferrites of copper, the iron and alumina would be rendered relatively insoluble in acid, the ore would be made more porous for leaching, there would be no argillaceous slimes to absorb valuable constituents, the copper sulphides and cuprite would be rendered soluble and the lixiviums might be clean enough for electrolysis.

My work was finished before this was tried.

[4] See my discussion, *Engineering and Mining Journal*, vol. xcvii, No. 17, p. 871 (April 25, 1914).

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Development of the Round Table at Great Falls

BY ARTHUR CROWFOOT, GREAT FALLS, MONT.

(Salt Lake Meeting, August, 1914)

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## Introduction

The principal object of this paper is to present data on the development of the revolving convex round table as a concentrator for the through 0.07-mm. slimes from the ores of the Butte district, although some brief notes on the earliest recorded use of this type of concentrating table in the ore-dressing plants of the Lake Superior region and of the Butte district are also presented.

The revolving convex round table belongs to the class of concentrating tables known as film-sizing tables, these tables using the relative transporting power of a film, or thin sheet of water, as it flows over a quiet inclined surface, to separate the minerals of a sorted product.

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The grains of heavy mineral move down the slope of the table deck slowly, or, in some cases, not at all, after the initial force of the feed entry is expended, since, being smaller, they are acted upon only by the slowly moving portion of the water which is in contact with the surface of the table; on the other hand the grains of light mineral (gangue), being larger, are exposed to the more rapidly moving water of the upper current and are therefore moved much faster down the slope of the table.

The speed with which the coarser grains of gangue material move down the slope of the table is also accelerated intermittently by the action of the waves of pulp flowing down the table. The coarser grains of gangue material are submerged, partly in the slow-moving under current and partly in the swifter upper current, and their speed of travel is greater than that of the former and less than that of the latter. Observations made on the action of the pulp on the table have shown that these coarse grains of gangue material are subject to distinct blows from the waves of pulp which momentarily increase the speed of these grains.

The speed at which the under current, or friction film, moves is dependent upon the character of the surface of the table, a rough surface retarding the flow of this current while a smooth surface accelerates the flow.

The revolving round table is a development of the stationary Cornish buddle, one of the oldest forms of slime concentrators, and it is a fact of great interest that a machine so closely related to one of the oldest forms of concentrating machines should be selected as the right machine for an important position in the flow sheet of our latest milling practice on copper sulphide ores in Montana.

The origin of the buddle seems to be lost in obscurity, and there is considerable doubt as to whether Cornwall or Germany first introduced this method of concentration. Cornwall has laid claims to having taught the art of tin mining and metallurgy to the Germans, and a story is related by one Matthew Paris, a Benedictine monk, by birth an Englishman, who died in 1259, that a Cornishman who fled to Germany on account of a murder which he had committed, first discovered tin there in 1241, and that in consequence the price of tin fell greatly. This would seem to point to the probability of the buddle having originated in Cornwall.

Georgius Agricola in his *De Re Metallica* devotes considerable space to descriptions of dressing implements which were similar to the early forms of the box buddle. These consisted merely of a wide inclined stationary trough, with twigs or riffles placed so as to catch the heavy particles, the principle being the same as that used in Cornwall to-day. Agricola was born in 1494, and therefore this method of dressing or concentrating ores evidently dates back earlier than this time.

In the third century B. C. the Greeks practiced a system of milling

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and concentrating, using a method of washing the crushed ore which was very similar in some respects to the box buddle.

Edward Ardaillon in *Les Mines du Laurion dans l'Antiquite* gives the following details:

"From the mills it (crushed ore) was taken to washing plants, which consisted essentially of an inclined area, below which a canal, sometimes with riffles, led through a series of basins, ultimately returning the water again to near the head of the area. In washing, a workman brushed upward the pulp placed on the inclined upper portion of the area, thus concentrating there a considerable proportion of the galena; what escaped had an opportunity to settle in the sequence of basins somewhat on the order of the buddle."

The square or box buddle of Agricola's time would hold about 600 lb. of material when full. Owing to the necessity of returning the lower two-thirds of the material deposited to be re-treated on the same buddle, also to the amount of labor required to operate it, this form of buddle is practically obsolete everywhere.

The round buddle works on the same principle as the box buddle, and could be described as a series of box buddles arranged radially, with their feed ends in the center.

To obviate channeling of the ore, arms are attached to the rotating feed gear, and on these arms are hung strips of brattice or coarse cloth, which gently drag over the deposit, and thereby keep the surface even. This type of buddle is in use both in the concave and the convex forms. In Cornwall it is very extensively used in tin dressing, also in Derbyshire, Isle of Man, and Cumberland for concentrating galena ore.

The Cornish buddle or building table, which was used almost exclusively as a concentrator for tin ores in Cornwall during the early days of tin mining in that district, was usually from 18 to 20 ft. in diameter and was built in the form of a circular tank with sides 18 in. deep, the bottom sloping gently from the center to the circumference. In the center of the tank or table was a mound with a conical top, the mound being from 5 to 6 ft. in diameter. Pivoted in the center of the mound was a vertical shaft which revolved. This shaft carried a pulp distributor and four wooden arms which extended out over the buddle, the arms being equipped with brushes made of cocoa matting frayed at the ends or wisps of brush which dragged lightly over the sand as it accumulated in the buddle. These brushes kept the surface of the charge trued up and hence assisted the work of concentration very materially by preventing channeling.

During the process of running a charge in a buddle the water and slime were drawn off at points on the circumference which were equipped with sliding doors perforated with auger holes. As soon as the buddle became filled, the foreman examined the contents, marking off with the point of

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his shovel four or five concentric circles on the surface of the charge, their relative position depending on the richness of the mineral within each. The buddle crew then shoveled the outer ring, which was mostly slime, into a launder which delivered it to a settling pond; the next ring was thrown into a waste tailing launder, the intermediate rings were shoveled into wheelbarrows and carried to other buddles for re-treatment, while the inner ring of all, containing about 25 per cent. black tin, was sent to the calciners.

The total result of this first buddling was that about one-quarter of the waste was removed from the pulp treated; another small part was sufficiently enriched to be sent to the calciners, while probably more than one-half of the whole had to be re-treated.

This method of buddle treatment has been supplanted in the more progressive plants by the use of Frue vanners and revolving round tables.

American milling practice has always been averse to the use of concentrating machines which operate intermittently, or machines which produce a large amount of middling material for retreatment, therefore, the revolving round table, which at once overcame so many of the difficulties of the operation of the stationary buddles, made rapid headway in American mills in displacing stationary buddles, kieves, and other intermittent machines.

### *The Round Table in Lake Superior Practice*

The milling practice in the Lake Superior district has been described in papers read before the Institute by Charles W. Rolker,[1] H. S. Munroe,[2] and F. G. Coggin.[3] As nearly as the writer can learn the first revolving round table was introduced in the Lake Superior region in 1873 and by 1883 was in general use in the district. The round table most largely used was the design of W. J. Evans, the distinctive feature of which was the "dead-head" at the center, from 6 to 8 ft. in diameter, one-half of which was used for distributing the slime and the other half for distributing clear water.

The part played by the revolving round table in the flow sheet of the Michigan mills was to treat the spigot discharges of the settling tanks receiving fine product from the stamp mills, the material coming to the tanks as overflow material of the hydraulic classifier.

Before the advent of the reciprocating table, the round tables were used to produce a finished concentrate; two decks on a shaft were generally used, the middling from the upper deck being re-treated on the lower deck or on a separate system of tables.

[1] *Trans.*, v, 584 to 606 (1876–77).
[2] *Trans.*, viii, 409 to 451 (1879–80).
[3] *Trans.*, xii, 64 to 68 (1883–84).

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The writer is informed by C. H. Benedict of the Calumet & Hecla company that the present practice at its mills is in no case to make a finished concentrate on the round tables, these tables being always followed by Wilfley tables, either two, three, or even six decks of the round tables being dressed up on one Wilfley table.

Round tables are in use at the C. & H., Tamarack, Ahmeek, Osceola, Isle Royal, and Mass mills. Ordinarily there are three decks on one shaft except at the C. & H. mills where there are four decks on a shaft. The tables make 1 rev. per minute, and in new installations cement deck surfaces are used to replace the wood deck surfaces of the older installations.

The slope of deck surface in general is 1.5 in. per foot although a slope of 1.25 in. per foot is sometimes used on fine material. The tonnage fed per deck ranges from 15 to 20 tons per 24 turns and the recovery made is ordinarily from 40 to 50 per cent. On low-grade ores the ratio of concentration may reach as high a figure as 80 into one and is never less than 30 into one. With classifiers now in use the maximum size of grain going to the round table is about 0.25 mm.

The revolving round table used at the present time by the Calumet & Hecla Mining Co. consists of four decks mounted on a shaft. The table is driven from the top by means of a crown gear and pinion and is timed to make 1 rev. per minute. The total height of the table from crown gear to floor line is 21 ft. and the spacing between the decks is 3 ft. 8 in. The decks have the usual umbrella frame, the supporting braces consisting of 2-in. angle irons. The frame is so designed that the supporting braces occupy very little of the space between the decks. The circumference of the decks is stiffened by 3-in. angle-iron rim segments bent to the proper curve. The deck surface used is cement laid on wood, the cement being about $\frac{1}{2}$ in. thick.

Each deck is equipped in the center with a cone with a 45° slope and a base diameter of 3 ft. The feed pulp is delivered on one side of the table only from a special form of feed box which delivers the pulp against the face of the 45° cone. The decks are in general given a slope of 1.5 in. in 1 ft.

*Early Use of Revolving Round Table in Butte District*

From C. W. Goodale's paper on the Concentration of Ores in the Butte District, Montana,[4] it would appear that the Colorado Smelting & Mining Co. was the first to install round tables in the Butte district. The plant started operations in 1882. Round tables were installed in the initial flow sheet of this mill, but were afterward discarded in favor of Frue vanners.

[4] *Trans.*, xxvi, 602 (1896).



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In the concentrator of the Montana Copper Co. the reverse is the case, as Frue vanners were first installed and later were replaced by round tables.

## Use of the Round Table at Great Falls for Sand-Slime Feed

Construction work on the first concentrator of the B. & M. C. C. & S. Mining Co. at Great Falls was started in the spring of 1891, the concentrator being put into operation in March, 1892. A second concentrator was completed and put into operation in the fall of 1900. Convex revolving round tables were installed for the treatment of the fine sand and slime which overflowed the Evans hydraulic classifiers.

The round-table practice at Great Falls, preceding the development of the table as a concentrator for through 0.07-mm. slime, did not differ greatly from the round-table practice in other Montana mills; usually the feed ranged in size from 50 mesh (0.36 mm.) to zero, with from 3 to 5 per cent. of coarser material due to the imperfections of the hydraulic classifiers employed, or to the overloading of these classifiers. No very close classification of feed was attempted, nor was any attempt made to produce a waste tailing from the round tables, the tables being guarded by the vanners. The use of the Wilfley table, under the conditions of round-table operation, rapidly superseded the use of the round table for the following reasons: (*a*) A cleaner grade of concentrate was produced which contained as high or higher a percentage of the copper fed to the table; (*b*) a large tonnage of fairly low-grade tailing could be produced from a feed no more closely classified than the feed sent to the round tables, thus eliminating the re-treatment of this material on vanners; (*c*) the Wilfley table acted as a classifier in that it made a separation between the sand and slime fed to it.

It is probable that good work could be obtained from round tables when fed with sand ranging in size from 60 mesh (0.25 mm.) to 200 mesh (0.07 mm.). Under these feed conditions, the very fine slime being eliminated from the feed, the tables should produce a large amount of low-grade tailing product.

## The Round Table for Fine Slime

The use of the revolving convex round table as a concentrator for a through 0.07-mm. slime feed was originated by J. M. Callow of the General Engineering Co., Salt Lake City, in an exhaustive series of tests made by him on the concentration of overflow slime at the Boston Montana concentrator, Great Falls, during the period October to December, inclusive, 1904.

This is believed to be the first time that so fine a feed had been treated alone on a round table; the writer has looked up all available

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literature on the subject but can find no record of a feed which did not contain a large percentage of coarser sand, the feeds usually varying from 0.35 mm. down.

The concentration of the slime was preceded by a dewatering or pulp-thickening operation, the slime as produced in the mill being unwatered in a peripheral overflow tank of Mr. Callow's design, since well known under the name of the Callow settling tank or cone.

By the use of this unwatering device, preceding the concentrator, the density of the slime was increased from about 70 g. of solids per gallon of pulp (2 per cent. solids) to about 350 g. of solids per gallon of pulp (8.75 per cent. solids) with a loss of only from 2.0 to 2.5 per cent. of solids and copper in the overflow of the tank. The capacity per Callow tank, to obtain the above results on B. & M. slime, was 25 gal. of slime feed per minute or 36,000 gal. per 24 hr. per tank, the tank being equipped with a goose-neck spigot discharge with an opening $\frac{3}{8}$ in. in diameter operating under a head of 24 in. A full description of Mr. Callow's tests will not be attempted in this paper, the general results only being given.

The machines at Mr. Callow's disposal for trying out the concentration of the thickened slime pulp from his tanks consisted of a smooth-belt Frue vanner, a Wilfley table, and a revolving convex round table.

The round table was not at first considered by Mr. Callow, the first four tests being made on a Frue vanner and the next 16 on a Wilfley table. The copper recoveries made by the Frue vanner and the Wilfley table were in both cases low, especially so in the case of the Frue vanner, which was not equipped with the necessary stroke adjustments for the treatment of so fine a feed, which it must be remembered was about 96 per cent. finer than 0.07 mm. (200 mesh). Mr. Callow then directed his efforts toward finding what could be accomplished by using the revolving round table as a concentrator for the slime and his first results were so satisfactory that the remainder of his tests were made on this table.

No mechanical adjustments were possible except the speed and this remained unchanged from the speed used for the regular sand-slime feed, 1 rev. in 2 min. The Wilfley table and vanner have been developed as slime concentrators since this period but the development of the round table has enabled it to maintain its supremacy in this particular branch of ore dressing.

The tests made by Mr. Callow included tests made on the individual slimes and on various combination of the slimes, the end results of which showed that the Callow tanks would handle from 25 to 30 gal. of slime pulp per minute, thickening this slimy water from a density of about 60 g. of solids per gallon of pulp to a density of about 350 g. (1.5 to 8.75 per cent. solids), overflowing about 90 per cent. of the water contained in the slime and delivering from 95 to 98 per cent. of the solids fed in the spigot discharge for treatment on the round tables.

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The revolving convex round table was shown to have a capacity for treating from 4 to 5 tons of the thickened slime per 24 hr., producing from ¾ to 1 ton of concentrate assaying from 5 to 6 per cent. copper and 71 to 75 per cent. insoluble, and containing about 50 per cent. of the copper in the feed, the ratio of concentration being from 5.5 to 6 tons into one.

It was recommended by Prof. R. H. Richards that an Embrey vanner be tried out as a concentrator for the thickened slime pulp from the Callow tanks, as, in his opinion, the end-shake vanner, to which type the Embrey vanner belongs, would prove to be a superior machine to the side-shake vanner for the concentration of the fine slime.

An Embrey vanner was, therefore, installed in the concentrator and thoroughly tested on the treatment of the slime during March and April, 1905. The results of the tests confirmed Professor Richards's statement in regard to the fact that the end-shake vanner would develop a higher efficiency than the side-shake vanner in the treatment of very fine slime, the recovery of copper being increased, but the recovery made by the Embrey vanner still fell short of the recovery made by the revolving convex round table and the capacity of the vanner was much lower. The main data received from tests made on the Embrey vanner follow (p. 1939).

Some results obtained from tests made in the Frue vanner when equipped with a corrugated rubber belt are also presented.

### *Construction of Slime Plant at Great Falls*

In order to try out Mr. Callow's flow sheet on a commercial scale it was decided to erect a slime plant with a capacity for treating about 1,000,000 gal. of slime per 24 hr., the pulp carrying about 80 tons of solids. This required an installation of 30 Callow tanks and four four-deck round tables. Construction was started on this plant during the summer of 1905 and the operation of the plant was started on Oct. 1, 1905.

The building was so designed, and the equipment so arranged, that the plant could be readily enlarged to the full capacity required should the results obtained justify this course.

The original dewatering and concentrating equipment of the slime plant consisted of 30 8-ft. Callow tanks and 16 17-ft. revolving round tables arranged as four four-deck tables. The round tables used were the two-deck tables removed from the concentrator at about this time, two double decks being set up one above the other and the shafts connected with a coupling so as to form a four-deck table.

Of the 16 decks installed, four were given a linoleum cover, four a cement surface laid on the original wood surface, and on the remaining eight the original wooden surface was maintained. The circular feed distributing apron with a 2 in. to the foot slope, previously mentioned,

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was allowed to remain on the tables. The slope of the deck surfaces was 1.25 in. per foot for the wood and linoleum surface decks and 1.5 in. per foot for the cement surface decks; the speed of all tables was 1 rev. in 100 sec. or 36 rev. per hour. The feed pulp was distributed over one-half of the table and washed with dressing water on the other half of the table, the tables being operated in practically the same manner as when treating a sand-slime feed in the concentrator.

The cement-surface decks installed in the slime plant were the first decks of this character installed at the Great Falls works; cement-surface decks were coming into general use in the Michigan ore-dressing plants at this time and probably had been used at other plants; in fact,

*Results of Tests Made on the Embrey Vanner*

| Date 1905 | Data on Feed to Vanner: No. of Tests Made | Rate per 24 Hr.: Gallons | Rate per 24 Hr.: Pounds | Density in Grams per Gal. | Assay Per cent. Cu | Concentrate: Assay Cu | Concentrate: Per cent. Insol. | Assay Per cent. Cu in Tailing | Ratio of Concentn. Tons to 1 | Recovery of Cu Per cent. |
|---|---|---|---|---|---|---|---|---|---|---|
| **Group No. 1** | | | | | | | | | | |
| Rev. per min. 225, 1-in. stroke, belt speed 6 ft. 8 in. per min., slope 4 in. in 12 ft. | | | | | | | | | | |
| Mch. 8 | 1 | 3,850 | 5,100 | 600 | 2.4 | 9.5 | 64.0 | 1.7 | 11.7 | 33.0 |
| **Group No. 2** | | | | | | | | | | |
| Rev. per min. 225, 1-in. stroke, belt speed 9 ft. 6 in. per min., slope 6 in. in 12 ft. | | | | | | | | | | |
| Mch. 9–14 | 6 | 4,210 | 3,400 | 364 | 2.7 | 6.8 | 73.0 | 2.13 | 8.3 | 30.5 |
| **Group No. 3** | | | | | | | | | | |
| Rev. per min. 225, 1-in. stroke, belt speed 10 ft. per min., slope 9½ in. in 12 ft. | | | | | | | | | | |
| Feb. 23–28 | 6 | 3,730 | 3,930 | 473 | 3.2 | 10.0 | 60.0 | 2.53 | 9.75 | 32.5 |
| **Group No. 4** | | | | | | | | | | |
| Rev. per min. 225, 1-in. stroke, belt speed 12 ft. 9 in. per min., slope 11½ in. in 12 ft. | | | | | | | | | | |
| Mch. 1–7 | 5 | 4,050 | 3,900 | 439 | 2.48 | 6.95 | 71.0 | 1.91 | 7.63 | 36.8 |
| **Group No. 5** | | | | | | | | | | |
| Rev. per min. 225, 1-in. stroke, belt speed 16 ft. per min., slope 18 in. in 12 ft. | | | | | | | | | | |
| Mch. 7–25 | 28 | 4,286 | 3,818 | 394 | 2.46 | 7.54 | 69.0 | 1.99 | 7.21 | 37.3 |
| **Group No. 6** | | | | | | | | | | |
| Rev. per min. 248, ¾-in. stroke, belt speed 16 ft. per min., slope 18 in. in 12 ft. | | | | | | | | | | |
| Apl. 14–22 | 11 | 5,829 | 5,199 | 425 | 2.3 | 7.20 | 68.0 | 1.7 | 9.4 | 37.8 |
| **Group No. 7** | | | | | | | | | | |
| Rev. per min. 247, ¾-in. stroke, belt speed 18 ft. per min., slope 18 in. in 12 ft. | | | | | | | | | | |
| Apl. 24–26 | 3 | 5,887 | 5,416 | 495 | 2.5 | 7.70 | 68.8 | 1.8 | 10.8 | 31.0 |

John A. Church, mentions having installed cement-deck round tables at Tombstone, Ariz., in 1883.[5]

Mr. Church states that the use of cement for round-table deck sur-

[5] *Trans.*, xv, 601 (1886–87).

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*Results of Tests Made on Frue Vanner, February to April, 1905*

| | | | | | | |
|---|---|---|---|---|---|---|
| Revolutions per minute | 196 | 196 | 196 | 196 | 196 | 196 |
| Length of stroke in inches | 1 | 1 | 1 | 1 | 1 | 1 |
| Belt speed in feet per minute | 5 | 4 | 2 | 6 | 9 | 10 |
| Slope of belt, in. in 12 ft | 6 | 7 | 9 | 5 | 8½ | 8½ |
| Number of tests made | 1 | 1 | 4 | 6 | 4 | 3 |
| Rate of feed, gallons per 24 hr | 4,620 | ..... | 4,350 | 5,127 | 5,977 | 5,745 |
| Rate of feed, dry solids, pounds, 24 hr. | 3,780 | 4,550 | 4,082 | 4,804 | 5,088 | 5,484 |
| Density of feed in grams per gallon | 380 | ..... | 406 | 391 | 398 | 427 |
| Assay per cent. copper in feed | 3.3 | 2.7 | 2.47 | 3.1 | 2.2 | 2.4 |
| Assay per cent. copper in concentrate | 8.9 | 7.2 | 6.7 | 7.2 | 5.1 | 6.5 |
| Assay per cent. insoluble in concentrate | 68.4 | 72.4 | 71.3 | 73.5 | 77.1 | 73.3 |
| Assay per cent. Cu in tailing | 2.7 | 2.2 | 2.0 | 2.4 | 1.6 | 1.8 |
| Ratio of concentration, tons into 1 | 11.1 | 8.65 | 9.37 | 6.8 | 6.2 | 8.7 |
| Per cent. Cu fed, recov'd in concentrate. | 24.4 | 30.7 | 29.9 | 33.5 | 35.7 | 33.3 |

faces was not original with him at that time, he having read of its use although he had not seen tables with this character of deck surface in operation. Mr. Church introduced the round tables in the Tombstone district after seeing the performance of the revolving round tables in the Lake Superior copper mills, where, at that time they were made entirely of wood. The deck slopes used by Mr. Church were as follows: 1.75 in. per foot for all but the finest slime, for the treatment of which the deck slope was reduced to 1.25 in. per foot. A novelty introduced by Mr. Church was the construction of some tables with iron framework, under the impression that the swelling of timbers in such a wet place as a mill might throw the surface out of true. In constructing the above-mentioned concrete-deck surfaces the body of the table surface was made of concrete with broken stones up to ⅜-in. ring and with the cement so scant, that the concrete was visibly (and coarsely) porous. A thin skin of pure cement was laid on this, of not much more than paper thickness. Mr. Church states that these tables kept their surface extremely well, better than some all-cement surfaces that he has seen; he thinks, however that the quality of the cement has more to do with the results than the mode of laying on.

### *Tests on Wood, Linoleum, and Cement Decks*

To return to the Great Falls slime plant. In January, 1900, the writer, who was placed in charge of the slime plant during the first year of its operation, carried out a series of tests with a view to determining the relative effectiveness of wood, linoleum, and cement deck surfaces on a round table, two series of tests being made each extending over a 10-day period. During the first test the total feed to the tables was distributed over 12 of the 16 decks, one four-deck table being shut down, while during the second test the 16 decks were used. The first test was called a high-tonnage test and the second a low-tonnage test. Three tables were tested, the deck surfaces being wood, linoleum, and cement, respectively.

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The average results of the high-tonnage tests only are given in detail in the following tabulation:

*High-Tonnage Test Results.*—The tonnage treated per 24 hr. during the above-mentioned tests was approximately 4.5 tons, which was called a high tonnage at the time of making the tests; the writer will show later how the tonnage treated per deck surface has been greatly increased. In considering the above feed it must be remembered that this feed contained a large percentage of colloidal slime in addition to the very fine sands, although at the time the tests were made it had been decided to subject the Callow tanks to an overload in order to secure a greater copper production from the plant, it being more profitable under the exist-

*Tests Made on Round Tables with Wood, Linoleum, and Cement Deck Surfaces*

| Data | Kind of Deck Surface | | |
|---|---|---|---|
| | Wood | Linoleum | Cement |
| Tons dry solids treated per 24 hr | 4.388 | 4.474 | 4.517 |
| Density of feed in grams per gallon | 310 | 280 | 296 |
| Assay per cent. copper in feed | 2.77 | 2.75 | 2.76 |
| Tons of concentrate produced per 24 hr | 1.422 | 1.149 | 0.954 |
| Assay per cent. copper in concentrate | 5.29 | 5.92 | 7.03 |
| Assay per cent. insoluble in concentrate | 76.13 | 73.4 | 69.3 |
| Per cent. of copper fed, recovered in concentrate | 62.14 | 55.28 | 53.81 |
| Ratio of concentration, tons into one | 3.086 | 3.894 | 4.735 |
| Ratio of enrichment (copper) | 1.90 | 2.153 | 2.547 |
| Tons feed-side tailing produced per 24 hr | 1.393 | 1.615 | 1.614 |
| Assay per cent. copper | 1.77 | 1.86 | 1.82 |
| Tons wash-side tailing produced per 24 hr | 1.573 | 1.709 | 1.969 |
| Assay per cent. copper | 1.35 | 1.45 | 1.44 |
| Tons total tailing produced per 24 hr | 2.966 | 3.324 | 3.563 |
| Assay per cent. copper | 1.55 | 1.65 | 1.61 |
| Gallons of fresh water used per 24 hr | 26,423 | 22,254 | 24,501 |
| Gallons of fresh water used per ton fed | 6,022 | 4,974 | 5,424 |
| Per cent. of original feed solids in concentrate | 32.41 | 25.69 | 21.12 |
| Per cent. of original feed solids in tailing | 67.59 | 74.31 | 78.88 |

ing conditions to operate the plant in this manner. On Jan. 1, 1906, the feed per Callow tank was, therefore, increased from 29.8 gal. per minute to 41.6 gal. per minute, which had the effect of raising the density of the Callow tank overflow from 3.5 to 13.5 g. of solids per gallon of pulp. As the increased overflow loss was made up largely of colloidal slime, the character of the spigot discharge of the tanks (round-table feed) was changed to some extent and in a manner which made it more susceptible to concentration on a round table due to the lesser percentage of colloids in the feed. This accounts for the fact that the table recoveries shown by the test results are from 4 to 12 per cent. higher than the results obtained by Mr. Callow.

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A reference to the above table shows that the wood deck surface recovered 6.86 per cent. more of the feed copper in concentrate than the linoleum-covered deck and 8.33 per cent. more than the cement deck surface, but also, that this additional recovery was made at the expense of a more siliceous concentrate and a concentrate with a lower copper value, the wood-deck concentrate assaying 0.63 per cent. less in copper than the linoleum-deck concentrate and 1.74 per cent. less than the cement-deck concentrate. From the above it follows that the wood deck produced a lower tonnage of a lower grade of tailing than the other decks and this is shown to be the case, the wood deck producing 0.358 ton less tailing than the linoleum deck and this tailing assaying 0.10 per cent. less in copper, and 0.597 ton less tailing than the cement deck assaying 0.06 per cent. less in copper.

It may be well to parallel additional data on the tailing losses from the tables and this is done in the following table:

*Data on Tailing Losses*

| | Kind of Deck Surface | | |
|---|---|---|---|
| | Wood | Linoleum | Cement |
| Per cent. of feed solids in feed-side tailing | 31.75 | 36.11 | 36.39 |
| Per cent. of feed copper in feed-side tailing | 20.16 | 24.39 | 24.10 |
| Assay per cent. Cu of feed-side tailing | 1.77 | 1.86 | 1.82 |
| Per cent. of feed solids lost in wash-side tailing | 35.84 | 38.20 | 42.49 |
| Per cent. of feed copper lost in wash-side tailing | 17.70 | 20.33 | 22.09 |
| Assay per cent. Cu of wash-side tailing | 1.35 | 1.45 | 1.44 |
| Per cent. of feed solids lost in total tailing | 67.59 | 74.31 | 78.88 |
| Per cent. of feed copper lost in total tailing | 37.86 | 44.72 | 46.19 |
| Assay per cent. Cu of total tailing | 1.55 | 1.65 | 1.61 |

The tailing made by the linoleum and cement decks does not show a notably higher assay copper percentage than the tailing produced by the wood deck, especially so in the case of the cement deck, the greater loss of copper in tailing being largely due to an increased production of tailing material. For this reason the writer favored the use of cement deck surfaces with a slightly flatter slope than the deck under test (this deck having a slope of 1.5 in. per foot), providing the cement was laid on a rigid framework so that the cement could not crack or get out of true in any way.

*Low-Tonnage Test Results.*—The reduction of the tonnage fed to the decks per 24 hr. from 4.5 to 3.5 tons, improved the copper recovery and reduced the tailing loss on all three decks but had the greatest effect on the work of the linoleum and cement decks, the recovery made by the latter decks approaching to within 3.0 per cent. and 1.4 per cent. respectively of the recovery made by the wood deck, this being partly due to a slight lowering of the grade of concentrate produced in each case.

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A comparison of the results obtained in the low-tonnage tests showed that the additional loss of copper in tailing from the linoleum and cement decks was due entirely to the production of a greater tonnage of tailing material than was produced by the wood deck, the linoleum deck discarding 3.64 per cent. more of the feed solids as tailing and losing 2.99 per cent. more of the feed copper than the wood deck, and the cement deck discarding 5.10 per cent. more of the feed solids as tailing and losing only 1.37 per cent. more of the feed copper. These results also favored the use of cement deck surfaces as tending to produce a cleaner grade of concentrate and a lower grade of tailing with a lower consumption of dressing water if rigidly constructed and given the right slope. During the running of above tests the writer noted the fact that no concentration took place on the feed aprons previously described, these aprons having a slope of 2 in. in 1 ft., and also noted that very little settlement took place on the first 8 or 10 in. of the working radius of the table, due to the velocity that the steep slope of the apron and the fall of about 1 in. from surface of apron to surface of table deck gave to the feed pulp.

### *Central Feed Apron Removed*

At the conclusion of the tests, February, 1906, the writer removed the feed apron from one of the wood decks, filled in the space with a cement mortar (3 sand, 1 cement), with a finishing coat of neat cement. A slope of but 1 in. in 1 ft. was given to this surface since it was considered that the initial velocity of the feed when discharged from the feed box was sufficient to warrant lesser slope than that given to the remainder of the deck surface.[6] The results of this change showed that a considerable deposit of very clean concentrate could be obtained within 12 in. of the edge of the wash-water cup, the working radius of the table being lengthened about 3 ft. in the direction of the center. Shortly afterward another of the wood decks was changed in the same manner and a new type of feed and wash-water cup was introduced with a view to getting a better delivery of feed on the table. These feed and wash-water cups, which for experimental purposes were made of galvanized iron, were so constructed that the feed and wash water overflowed the rims, falling down the sides on to an apron with 3-in. radius and 15° slope, the outer edge of the apron practically touching the deck surface. These feed and wash water cups, if kept perfectly level and given attention by the operator about twice per shift to remove any foreign material, gave satisfactory service, delivering the feed and wash water to the table with an even distribution.

Overflow feed and wash-water cups had been previously used on the

[6] This is the first time that a variable deck slope was tried at Great Falls.

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round tables, but on these the apron attached to the cup had a slope of 45° and the edge of the apron was usually from 6 to 8 in. above the surface of the distributing apron on the table. On all cement deck surfaces constructed since this time the central distributing apron has been dispensed with, the working radius of the table being extended to the feed and wash-water cup.

### *Re-treatment of Round-Table Tailing*

In June, 1907, the writer carried out a series of tests to determine to what extent the recovery of copper could be increased by re-treating the round-table tailing.

The feed and wash side tailings from six round tables (four wood and two linoleum deck surfaces) were fed into four Callow tanks, the plug discharges of the tanks in turn being fed on to a cement-deck round table, while the overflow went to the tail race. The cement-deck table made a low recovery of copper (21.29 per cent.) which when figured back to original slime gave an additional copper recovery of about 2.5 per cent. The concentrate produced was of a very low grade, being not much richer than the original slime.

A test was then made on the re-treatment of the feed-side tailing alone. A better grade of concentrate was obtained than from re-treating the mixture of feed and wash side tailings, but the recovery made was lower. The tests showed conclusively that the amount of original slime copper to be recovered by a re-treatment of the round-table tailing was small, probably no more than would be gained by a first treatment with tables properly designed for the treatment of a slime feed.

### *Determination of Conditions for Round-Table Treatment of Slimes*

Early in 1908 the writer had an experimental slime table constructed similar to the one described in Richards's *Ore Dressing*, vol. ii, p. 1155, with a view to carrying out a series of experiments to determine the proper slope of deck surface, and rate of feed, for round tables operating on a through 0.07-mm. feed. The table, which was 1 ft. wide and 6 ft. long, was made adjustable as to slope by an arrangement of steps which increased the slope by $\frac{1}{8}$ in. per foot at one time. A smooth cement surface was used on the table and the various slimes and slime mixtures were tested out under varying rates of feed, deck slopes, etc.

The conclusions reached from the results of the tests were as follows: The table should be 18 ft. in diameter; should have a cement deck surface sloping 1.25 in. in 1 ft.; and should make 1 rev. in 200 sec. A re-treatment of the feed-side tailings on a canvas deck surface was recommended.

During the summer of 1909 we carried out a series of tests in the concentrator, using an adjustable segment of a canvas-covered round

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table. The purpose was to determine the difference in behavior of the various concentrator slimes, as to both settling and concentration, and the proper treatment for each.

The experimental table as originally constructed was a segment of a round table, being the equivalent of one-tenth of an 18-ft. table. The slope of the table was adjusted by raising or lowering the feed end. The surface used on the table deck was No. 6 duck, the woof (cross threads) being laid approximately parallel to the direction of the slope. A large number of tests were made, both on the individual slimes and upon certain slime mixtures such as the slime plant was handling at the time.

The following general conclusions were arrived at: The primary slimes might be mixed, dewatered so as to recover about 98 per cent. of the solids, and then treated on a canvas-deck round table with a slope of 1.25 in. per foot. The table should make 1 rev. in 20 min. and produce a concentrate assaying 7.0 per cent. copper, and a tailing assaying 1.15 per cent. copper. For the lower-grade or secondary slime a slightly greater deck slope (1.375 in. per foot) was suggested, but it was considered probable that the 1.25 in. per foot slope would be found quite satisfactory for the normal mixture.

Toward the latter part of 1907, Dr. Robert H. Richards suggested the use of canvas-deck round tables for the slime plant, the tables to be given a speed of 1 rev. per hour and to be fed over nine-tenths of their entire surface, no dressing water to be used. Acting on this suggestion three round-table decks were covered with canvas (No. 6 duck) and the speed was reduced to 1 rev. per hour. It was expected that the concentrate would be so low in grade as to require a final dressing on another table, but such was not the case, as a very fair grade was produced.

The slime plant practice recommended by Dr. Richards was as follows: A low ratio of concentration, a low-grade concentrate and a high recovery on the round tables; the concentrate to be re-treated and enriched on a secondary table, from which the tailing would go back to the round table. In accordance with this suggestion, the round tables were recommended for use as roughing tables. Dr. Richards also recommended that the Embrey vanner be used as a finishing machine for cleaning up the rough concentrate from the round tables, but up to the present time this part of the plan has not been tried out, Deister, Craven, and James tables having been used as finishing machines.

### *Variation in Slope of Round-Table Deck*

The first round table with a variation in the slope of the deck surface, tried out at Great Falls, with the exception of the change made when removing the central feed aprons, was installed in the slime plant in June, 1910.

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J. H. Klepinger, Assistant Superintendent, suggested the use of a conoidal deck surface, and proposed to lay a cement surface of this character on one of the wooden decks. As the wooden decks in use were old tables, and as the cement deck would add greatly to the weight which the framework would have to support, it was thought that the structure would break down under the increased load. A compromise was therefore adopted, a cement deck surface being installed which was divided into two sections, at first of the same length radially, the feed apron previously referred to being left in the center, but of unequal areas, the upper deck being concentric to the lower deck. There was a drop of about 1.25 in. between the first and second deck sections. The first or upper deck section was given a slope of 1 in. in 1 ft. and the second or lower deck section a slope of 1.25 in. per foot. The steep feed apron with a slope of 2 in. per foot was afterward removed and the upper deck with its slope of 1 in. per foot was extended to the feed box.

This change gave the upper deck a radial length from the point where it received the feed, to its perimeter, of 5 ft. About 3 in. of radial length was occupied by the slope of the upper to the lower deck, leaving a radial length of 30 in. for the lower deck.

*Data From Five Tests on Cement Step Deck*

| | Date of Test Period | | | | |
|---|---|---|---|---|---|
| | June, 1910 | Sept., 1910 | April, 1911 | March, 1913 No. 1 | March, 1913 No. 2 |
| Time for table to make 1 rev., min. | 60 | 19 | 19 | 19 | 19 |
| Tons dry solids treated per 24 hr. | 5.84 | 11.3 | 5.4 | 6.184 | 4.825 |
| Density of feed in grams per gallon | 314 | 432 | 364 | 495 | 246 |
| Assay per cent. Cu in feed | 2.44 | 2.21 | 3.70 | 3.24 | 3.16 |
| Tons total concentrate produced per 24 hr. | 0.750 | 3.2 | 2.2 | 2.558 | 1.220 |
| Assay per cent. copper in concentrate | 7.75 | 4.83 | 7.19 | 5.98 | 8.64 |
| Assay per cent. insoluble in concentrate | 70.8 | 75.8 | 70.8 | 71.5 | 57.9 |
| Assay per cent. FeO in concentrate | 8.9 | 7.5 | 9.1 | 9.9 | 16.1 |
| Per cent. of copper fed, recov'd in conct. | 31.93 | 61.15 | 79.45 | 76.31 | 68.52 |
| Ratio of concentration, tons into one | 7.750 | 3.6 | 2.5 | 2.38 | 3.95 |
| Ratio of enrichment (copper) | 3.17 | 2.18 | 1.94 | 1.84 | 2.73 |
| Tons feed-side tailing prod. per 24 hr. | 2.51 | 4.02 | 0.9 | 1.388 | 2.044 |
| Assay per cent. copper | 1.58 | 1.08 | 1.03 | 1.58 | 1.22 |
| Tons wash-side tailing prod. per 24 hr. | 2.58 | 4.14 | 2.3 | 2.238 | 1.572 |
| Assay per cent. copper | 2.24 | 1.30 | 1.5 | 1.14 | 1.46 |
| Tons total tailing produced per 24 hr. | 5.1 | 8.2 | 3.2 | 3.626 | 3.616 |
| Assay per cent. copper | 1.9 | 1.19 | 1.28 | 1.31 | 1.33 |
| Gallons of fresh water used per 24 hr. | 8,382 | 12,347 | 9,417 | 15,659 | 16,783 |
| Gallons of fresh water used per ton fed. | 1,435 | 1,093 | 1,744 | 2,530 | 3,480 |
| Per cent. of original feed solids in conct. | 12.91 | 28.04 | 40.85 | 41.36 | 25.07 |
| Per cent. of original feed solids in tailing. | 87.09 | 71.96 | 59.15 | 58.64 | 74.93 |

This table was tested in June, 1910, being set to make 1 rev. per hour. The results obtained were not satisfactory, a heavy building up of solids occurring on the upper deck which caused a channeling of the feed and consequent heavy tailing loss. The speed of the table was then increased

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so that it made 1 rev. in 19 min. or practically 3 rev. per hour, the table being again tested in September, 1910, and April, 1911. In spite of the fact that the tonnage fed to the table in the September test was nearly double the tonnage fed in the June test, the work of the table showed great improvement, the channeling of the feed being nearly entirely eliminated. The April test was made on a lower tonnage, good results being obtained. The table was again tested in March, 1913, while the writer was carrying out some tests on conoidal deck tables, the steep feed apron having been removed in the meantime, two tests being made under different tonnage and feed-density conditions. The test results showed that the removal of the steep feed apron further improved the work of the table.

In the two 1910 tests and the April, 1911, test the table was operated as a roughing table, while in the March, 1913, tests the table was fed over about three-quarters of its area and washed with dressing water over about one-quarter of its area; this accounts for the smaller quantity of fresh water used in former tests.

In the April, 1911, test the central feed apron had been trued up so that a better distribution of feed was obtained.

In comparing the results of the two 1910 tests and the April, 1911, test it can be seen that the work of the table improved greatly with each succeeding test. In the June, 1910, test the slow speed of the table caused channeling even with a light feed. In the September, 1910, test the speed of the table and the tonnage were both increased, and although the tonnage was increased rather too much, the work done showed considerable improvement. In the June, 1910, test and the April, 1911, test the tonnages treated by the table were very nearly the same so that the effect of the increased speed can be better observed in these comparisons. The assay per cent. of copper of the feed in the April test was higher than in the June test, but the tailing produced was lower and the work done by the table showed great improvement. It was considered doubtful at the time, however, whether the work by this type of table would surpass the work done by a rigidly constructed cement-deck table with a regular slope of 1.25 in. per foot and so far these doubts have been justified.

### *The Steel-Frame Cement-Deck Round Table*

The construction of a four-deck steel-frame cement-deck round table (Fig. 1) was started early in 1911, the table being installed in the slime plant and put into commission on Apr. 17, 1911.

Of the four decks, the three upper decks were given a slightly rough finished cement surface while the lower deck was cemented in such a manner that canvas could be laid over the cement, the canvas being tacked to strips of wood laid radially in the cement. By this arrangement

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it was possible to change the deck readily from a canvas to a cement surface if the canvas proved unsatisfactory.

When the steel-frame table was designed it had already been proved that the rough canvas surface was satisfactory for a roughing table designed to produce, with high efficiency, a low-grade tailing and a low-grade concentrate. The disadvantages of the use of canvas were, the cost of maintenance and excessive amount of wash water required.

It was evident that if a cement surface of a character similar to canvas could be utilized, these disadvantages could be eliminated. It was for this reason that three of the decks of the new table were surfaced with cement at the start. The one canvas surface was soon replaced with cement.

The steel frame was a great improvement over the wood-frame construction; the cement decks being rigidly supported, a true conical surface could be maintained and the liability of cracks developing in the deck surfaces was eliminated. The cement decks were made of a mixture of one part cement to two parts concentrator-tailing sand.

The decks had a diameter of 18 ft. and a slope of 1.25 in. per foot, the distance between deck surfaces being 5 ft. The total height from floor to top of center shaft is 25 ft. 2 in. The diameter of the center shaft is 8 in., the shaft consisting of five cast-iron sections, a top and a bottom section and three intermediate sections, the intermediate sections being 5 ft. 1½ in. long.

The tables were equipped with movable sprays for washing off the concentrate. These were supposed to wash cleaner with less consumption of water than stationary sprays. The sprays had a throw of 9 in. transmitted by means of an eccentric and crank and made 4 strokes per minute; the use of movable sprays was afterward discontinued, since no especial advantage was shown over the stationary form.

### *Tests on Cement and Canvas Deck Surfaces*

One cement deck surface and the canvas-covered deck were thoroughly tested in April, 1911, the results of the tests being presented below.

It has been stated above that the decks had a slope of 1.25 in. per foot, but it was found from accurate measurements made on April 29, 1911, that the actual slope of the cement deck tested was 1.2 in. per foot and of the canvas deck 1.13 in. per foot. The chief data obtained from the tests are placed in parallel columns.

In the following tests the two tables were fed at the same rate per 24 hr., the feed for the canvas deck, however, being slightly higher in assay per cent. of copper than the feed for the cement deck.

The cement deck made a recovery of 82 per cent. of the copper fed in

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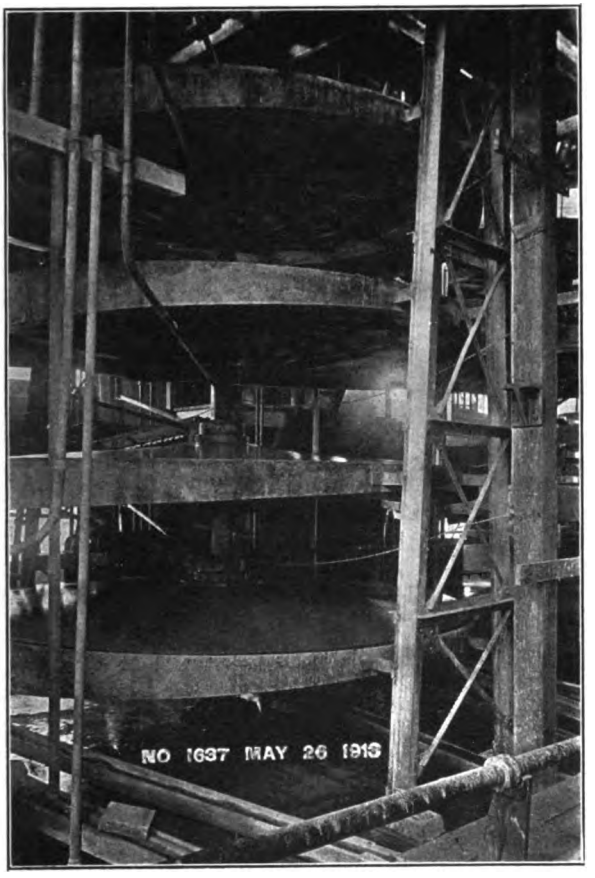
2.9 tons of concentrate, the ratio of concentration being 2.9 tons into one.

The canvas deck made a recovery of 87 per cent. of the copper fed in 4.3 tons of concentrate, the ratio of concentration being 1.9 tons into one.

If these concentrates were sent direct to the smelter in the condition




Deck surface, cement mortar (2 parts sand to 1 part cement). Slope of deck surface, 1.25 in. per foot. Diameter of decks, 18 ft. Decks make one revolution in 20 min.

FIG. 1.—FOUR-DECK STEEL-FRAME ROUND TABLE, GREAT FALLS.

in which they were produced by the round tables, the additional cost of fluxing the silica contained in the canvas-deck concentrate, together with the additional cost of loading, transporting, and unloading, would probably more than offset the advantage gained by the extra recovery of 5 per cent. of copper made by the deck.

As, however, the round tables were used as roughing tables, the concentrate produced being cleaned up on finishing tables, the canvas-deck table would appear to have some advantages in making a 5 per cent.

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higher recovery of copper than the cement deck. It is noted, however, that the assay value of the total tailing produced by the canvas deck is slightly higher than that of the similar product of the cement deck, the extra recovery of copper made by the canvas deck being traceable to a slightly richer feed.

The assay value of the feed-side tailing produced by the canvas deck is slightly higher than that of the cement deck, while for the wash-side tailing the copper content is lower in the case of the canvas deck.

*Test Results Obtained on Steel-Frame Cement and Canvas Deck Surfaces*

| | Kind of Deck Surface | |
|---|---|---|
| | Cement | Canvas |
| Time to make one revolution, minutes | 19 | 19 |
| Slope of table deck in inches per foot | 1.2 | 1.13 |
| Tons of dry solids fed per 24 hr. | 8.3 | 8.3 |
| Assay per cent. copper in feed | 3.36 | 3.46 |
| Density of feed in grams per gallon | 374 | 390 |
| Tons of concentrate produced per 24 hr. | 2.9 | 4.3 |
| Ratio of concentration, tons into one | 2.9 | 1.9 |
| Assay per cent. copper in concentrate | 7.84 | 5.79 |
| Assay per cent. insoluble in concentrate | 67.3 | 74.7 |
| Assay per cent. FeO in concentrate | 10.5 | 7.9 |
| Per cent. of copper fed, recovered in concentrate | 82.01 | 86.98 |
| Gallons of fresh water added per 24 hr. | 10,112 | 12,589 |
| Gallons of fresh water added per ton fed | 1,218 | 1,517 |
| Gallons of fresh water added per ton concentrate | 3,487 | 2,928 |
| Tons of north-side tailing produced per 24 hr. | 3.2 | 2.2 |
| Assay per cent. copper of above | 0.95 | 0.98 |
| Per cent. of feed solids in north-side tailing | 38.16 | 26.45 |
| Per cent. of feed copper in north-side tailing | 10.79 | 7.46 |
| Tons of south-side tailing produced per 24 hr. | 2.2 | 1.8 |
| Assay per cent. copper of above | 0.91 | 0.88 |
| Per cent. of feed solids in south-side tailing | 26.69 | 21.57 |
| Per cent. of feed copper in south-side tailing | 7.19 | 5.56 |
| Tons of total tailing produced per 24 hr. | 5.4 | 4.0 |
| Assay per cent. copper of above | 0.93 | 0.94 |
| Per cent. of feed solids in total tailing | 64.85 | 48.02 |
| Per cent. of feed copper in total tailing | 17.99 | 13.02 |

From the results obtained, it was the opinion of the writer that if the slope of the canvas deck had been the same as that of the cement deck, the advantage gained by the canvas deck in recovering an additional 5 per cent. of copper would have been wiped out in an increased tailing loss without very materially benefiting the grade of concentrate produced. It was, therefore, considered that the cement deck was the equal of the canvas deck in regard to its ability to recover copper mineral and its superior in producing a cleaner grade of concentrate, in using less water, and in possessing a longer life.

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Our experience with canvas deck surfaces has shown that the canvas has a comparatively short life, becoming rotten after about nine months, use. During September, 1910, a test had been made on a canvas-deck round table at the slime plant, the canvas being supported on a wooden deck surface with a slope of 1.25 in. per foot; in this case the canvas deck made a cleaner concentrate than the steel-frame canvas deck with a slope of 1.13 in. per foot, but the recovery of copper made was much lower. The comparative results of both tests are shown below.

*Data from Tests Made on Canvas-Deck Round Tables*

| | Steel-frame Canvas Deck | Wood-frame Canvas Deck |
|---|---|---|
| Slope of deck surface in inches per foot | 1.13 | 1.25 |
| Time to make one revolution, minutes | 19 | 19 |
| Tons dry solids treated per 24 hr | 8.3 | 7.6 |
| Assay per cent. copper in feed | 3.46 | 2.23 |
| Density of feed in grams per gallon | 390 | 409 |
| Tons of concentrate produced per 24 hr | 4.3 | 1.5 |
| Ratio of concentration, tons into one | 1.9 | 5.2 |
| Assay per cent. copper in concentrate | 5.79 | 7.33 |
| Assay per cent. insoluble in concentrate | 74.7 | 66.0 |
| Assay per cent. FeO in concentrate | 7.9 | 10.7 |
| Per cent. of copper fed, recovered in concentrate | 86.98 | 63.42 |
| Gallons of fresh water added per 24 hr | 12,589 | 21,531 |
| Gallons of fresh water added per ton fed | 1,517 | 2,833 |
| Gallons of fresh water added per ton of concentrate | 2,928 | 14,354 |
| Tons of north-side tailing produced per 24 hr | 2.2 | 2.97 |
| Assay per cent. copper of above | 0.98 | 0.94 |
| Per cent. of feed solids in north-side tailing | 26.45 | 39.75 |
| Per cent. of feed copper in north-side tailing | 7.46 | 16.92 |
| Tons of south-side tailing produced per 24 hr | 1.8 | 3.17 |
| Assay per cent. copper of above | 0.88 | 1.07 |
| Per cent. of feed solids in south-side tailing | 21.57 | 42.30 |
| Per cent. of feed copper in south-side tailing | 5.56 | 20.54 |
| Tons of total tailing produced per 24 hr | 4.0 | 6.1 |
| Assay per cent. copper of above | 0.94 | 1.01 |
| Per cent. of feed solids in total tailing | 48.02 | 82.05 |
| Per cent. of feed copper in total tailing | 13.02 | 37.46 |

Tailing products called north and south side tailings when the round tables are operated as roughing tables, as far as the side of the table from which they are delivered is concerned, correspond to feed and wash side tailings of finishing-table practice.

The foregoing comparison is given to show that a clean grade of concentrate can be produced from the canvas deck with an increase of the deck slope. The recovery of copper made by the wood-frame canvas



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deck, in producing a concentrate similar in character to the concentrate produced by the steel-frame cement deck, is 18.6 per cent. lower than the recovery made by the steel-frame cement deck. This great difference in recovery of copper is partly accounted for by the fact that the canvas deck treated a lower grade of feed than the cement deck. The canvas deck, however, in treating a slightly lower tonnage of a lower grade of feed, made a tailing which assayed higher in copper than the tailing produced by the cement deck.

Referring again to the test results obtained on the steel-frame cement deck and the steel-frame canvas deck. Eight sprays were used to wash off the concentrate from the canvas deck and two sprays on the cement deck; the spray nozzles on the canvas deck, however, had smaller diameters than those used on the cement deck.

A great deal more solid material settled out on the canvas deck than on the cement deck, during the test when a similar tonnage was treated on each kind of deck, the concentrate on the canvas deck amounting to 51.98 per cent. of the total solids fed, while the cement-deck concentrate amounted to only 35.15 per cent. of the total solids fed.

The results of the test indicate that given the same slope of deck surface, a higher grade of concentrate can be produced on a cement deck surface without any greater tailing loss than is incurred on a canvas surface.

In May, 1911, the speed of the tables was increased from 1 rev. in 19 min. to 1 rev. in 16 min., the object of the change chiefly concerning an improvement in the work of the canvas-deck table, it being expected that this change would enable this deck to produce a cleaner grade of concentrate with less tailing loss. The tables were tested under the new speed conditions but the results did not show that any advantage was gained.

The work of the steel-frame cement-deck table will be again referred to in connection with tests made on conoidal deck surfaces.

Some of the principal data obtained from the sizing tests made on the feed and products of the steel-frame wood and canvas decks in the above test are given below:

| Feed to tables: | Steel-frame Deck | |
|---|---|---|
| | Cement | Canvas |
| Per cent. of total solids on 200 mesh (0.07 mm.) | 4.34 | 6.54 |
| Assay per cent. copper of above | 0.81 | 0.85 |
| Per cent. of total copper on 200 mesh | 1.13 | 1.74 |
| Per cent. of total solids through 200 mesh | 95.66 | 93.46 |
| Assay per cent. copper of solids through 200 mesh | 2.98 | 2.89 |
| Per cent. of total copper through 200 mesh | 98.87 | 98.26 |
| Assay per cent. copper of total feed | 2.87 | 2.76 |

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| | Steel-frame Decks | |
|---|---|---|
| | Cement | Canvas |
| Concentrate produced: | | |
| Per cent. of total solids on 200 mesh | 2.05 | 3.97 |
| Assay per cent. copper of above | 1.35 | 2.10 |
| Per cent. of total copper on 200 mesh | 0.41 | 1.52 |
| Per cent. of total solids through 200 mesh | 97.95 | 96.03 |
| Assay per cent. copper of solids through 200 mesh | 6.95 | 4.90 |
| Per cent. of total copper through 200 mesh | 99.59 | 98.48 |
| Assay per cent. copper of total concentrate | 6.95 | 4.80 |
| North-side tailing: | | |
| Per cent. of total solids on 200 mesh | 5.36 | 9.13 |
| Assay per cent. copper of above | 0.28 | 0.33 |
| Per cent. of total copper on 200 mesh | 1.30 | 2.70 |
| Per cent. of total solids through 200 mesh | 94.64 | 90.87 |
| Assay per cent. copper of solids through 200 mesh | 1.20 | 1.12 |
| Per cent. of total copper through 200 mesh | 98.70 | 97.30 |
| Assay per cent. copper of total north-side tailing | 1.20 | 1.05 |
| South-side tailing: | | |
| Per cent. of total solids on 200 mesh | 8.43 | 12.26 |
| Assay per cent. copper of above | 0.36 | 0.66 |
| Per cent. of total copper on 200 mesh | 3.48 | 7.13 |
| Per cent. of total solids through 200 mesh | 91.57 | 87.74 |
| Assay per cent. copper of solids through 200 mesh | 1.02 | 1.00 |
| Per cent. of total copper through 200 mesh | 96.52 | 92.87 |
| Assay per cent. copper of total south-side tailing | 0.99 | 0.93 |

From the above figures it can be seen that the original feed to the round tables contained from 4.3 to 6.5 per cent. of material coarser than 200 mesh.

The figures given below show the grade of tailing made by the round tables from the material coarser than 200 mesh.

| Name of Tailing | Total Pounds per 24 Hour | Per Cent. on 200 Mesh | Pounds on 200 Mesh | Assay Per Cent. Copper | Pounds Copper on 200 Mesh |
|---|---|---|---|---|---|
| | | *Steel-Frame Cement-Deck Table* | | | |
| North | 4,941 | 5.36 | 265 | 0.28 | 0.74 |
| South | 3,693 | 8.43 | 311 | 0.36 | 1.12 |
| Total | 8,634 | 6.67 | 576 | 0.32 | 1.86 |
| | | *Steel-Frame Canvas-Deck Table* | | | |
| North | 3,209 | 9.13 | 293 | 0.33 | 0.97 |
| South | 5,765 | 12.26 | 707 | 0.66 | 4.67 |
| Total | 8,974 | 11.26 | 1,000 | 0.56 | 5.64 |

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The above figures show that with only 4.34 per cent. of on 200-mesh material contained in the feed to the table, the cement deck makes a very satisfactory tailing, assaying only 0.32 per cent. copper, this figure being comparable with a result of 0.49 per cent. copper obtained in a previous test when the feed to the deck contained 9.09 per cent. of on 200-mesh material. The on 200-mesh tailing produced by the canvas deck was somewhat higher, the amount of on 200-mesh material in the feed being also somewhat higher.

From the results of the above and previous sifting tests the writer concluded that the amount of on 200-mesh material in slime-plant feed should not exceed 4 per cent., and that the work of the tables could be improved if this amount were still further reduced.

### *Conoidal Deck Round Tables*

In May, 1912, the question of the use of conoidal deck round tables for slimes again came up for discussion, the subject being brought up by Prof. Henry S. Munroe of Columbia University.

Professor Munroe stated that his object in making the round-table deck conoidal instead of conical, was to make the movement of the pulp on the deck, and the concentration conditions, uniform, in spite of the divergence of the water on the deck as it flows from the center toward the circumference; he, therefore, adopted the plan of gradually increasing the slope of the table from the center to the circumference. Referring to the fact that as the stream of pulp on the table spreads out, the film becomes shallower and the transporting power becomes less, Professor Munroe stated that the most advantageous way to overcome this was to change the deck surface as described above. He stated that the frame and steel top of the round table experimented with at Columbia had a uniform slope of 4°.

The surface of the cement, however, was rounded so that the inclination varied from 3° at the center to 5° at the circumference. The table had a diameter of 18 ft. 5 in., so that the change of inclination was very gradual, something less than a quarter of a degree to the foot, and at the middle point of the profile the curved surface was only about ½ in. above a straight line. This table was designed for the treatment of medium-fine slime, through about 0.15 mm.; for the very fine slime treated at Great Falls, Professor Munroe recommended a slightly steeper slope.

The first conoidal deck surfaces tried out at Great Falls were installed by laying a conoidal cement surface on three of the wooden round-table decks which were still operating in section No. 5 of the concentrator. The tables, therefore, received the regular sand-slime feed of this section, which,as was mentioned earlier in this paper, was very poorly classified, containing grains ranging in size from 0.5 mm. to the very finest slime.

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A rough finish and a smooth finish deck surface were tested out under high, medium, and low tonnage conditions. The conoidal deck surface was laid on so that the chord of the arc made an angle of 4° 30′ (1 in. per foot) with the horizontal.

A large number of tests were made and the tables did good work considering the class of feed treated; a good recovery of copper was made in a fair grade of concentrate but no waste tailing could be made by the tables when handling this class of feed. As the flow sheet for future use in this section of the concentrator had already been decided upon, and as this practice could hardly be improved by the use of round tables unless a very close sorting of the feed was resorted to, it was determined to try out the conoidal deck surfaces in the slime plant, where the field was still open for any suggested improvements in round-table practice.

### *Results with Sand-Slime Feed on Conoidal Deck*

Before presenting data on the use of conoidal round-table deck surfaces in the slime plant, a few of the test results obtained from the operation of the tables in the sand-slime section of the concentrator will be presented to illustrate the character of the work done by the tables.

*Test Results from Conoidal Deck Round Tables*

| | Rough Finish Deck Surface | | |
|---|---|---|---|
| Character of feed treated (size) | Medium | Fine | Fine |
| Rate of feed | High | Medium | Low |
| Rate of feed in tons per 24 hr | 31.500 | 12.349 | 4.826 |
| Density of feed in grams per gallon | 693 | 358 | 259 |
| Density of feed, per cent. of solids in pulp | 16.48 | 9.0 | 7.14 |
| Assay per cent. copper in feed | 4.25 | 3.37 | 4.23 |
| Tons of concentrate produced per 24 hr | 3.823 | 1.554 | 0.675 |
| Density of concentrate in grams per gallon | 126 | 45 | 20 |
| Density of concentrate, per cent. solids in pulp | 3.25 | 1.20 | 0.55 |
| Assay, per cent. copper in concentrate | 18.90 | 16.64 | 19.06 |
| Assay, per cent. insoluble in concentrate | 18.4 | 18.4 | 31.8 |
| Assay, per cent. FeO in concentrate | 31.2 | 29.4 | 20.4 |
| Per cent. of copper fed, recovered in concentrate | 54.02 | 62.14 | 62.99 |
| Ratio of concentration, tons into one | 8.24 | 7.95 | 7.15 |
| Feed-side tailing solids, per cent. of total solids fed | 63.57 | 14.11 | 27.73 |
| Feed-side tailing, assay per cent. copper | 2.09 | 2.17 | 2.14 |
| Feed-side tailing, copper per cent. of total copper fed | 31.29 | 9.13 | 13.97 |
| Wash-side tailing solids, per cent. of total solids fed | 24.29 | 73.31 | 58.28 |
| Wash-side tailing, assay per cent. copper | 2.57 | 1.32 | 1.67 |

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| | | | |
|---|---|---|---|
| Wash-side tailing, copper, per cent. of total copper fed | 14.69 | 28.73 | 23.04 |
| Gallons of dressing water used per 24 hr | 13,887 | 25,133 | 18,077 |
| Gallons of dressing water used per ton of feed | 440 | 2,035 | 3,746 |
| Gallons of spray water used per 24 hr | 25,640 | 30,624 | 29,334 |
| Gallons of spray water used per ton of feed | 814 | 2,480 | 6,078 |
| Pounds of feed water per minute per foot of total circumference | 4.15 | 2.88 | 1.70 |
| Pounds of dressing water per minute per foot of total circumference | 1.39 | 2.60 | 1.82 |
| Pounds of spray water per minute per foot of total circumference | 2.86 | 3.32 | 3.18 |
| Revolutions of table per minute | 2 | 2 | 2 |
| Number of sprays operating | 8 | 8 | 8 |

An explanation of the "Character of feed treated" in the preceding tests is as follows: Medium feed is a mixture of all spigot discharges of the feed tank; fine feed is the product of the last three spigot discharges of the feed tank. The spray water used for washing off the concentrate was afterward greatly reduced by using five spray nozzles of smaller diameter instead of eight.

The high-tonnage test above mentioned was run to produce a slightly poorer grade of concentrate than we had been getting in previous tests on high-tonnage feeds and in this respect the test was successful, the change in the concentrates being practically the only variable with the exception of a slight drop in the ratė of feed. By increasing the insoluble content of the concentrate 3 per cent. the recovery of copper in concentrate was raised 10 per cent.

In the medium-tonnage test made on a fine feed a very clean grade of concentrate was produced and a fair recovery of copper made, the recovery being as good and the grade of concentrate better than in the low-tonnage test on the same class of feed.

| | Smooth Finish Deck Surface | | |
|---|---|---|---|
| Character of feed treated | Medium | Medium | Fine |
| Rate of feed | High | Medium | Low |
| Rate of feed in tons per 24 hr | 20.197 | 14.039 | 6.261 |
| Density of feed in grams per gallon | 452 | 271 | 292 |
| Density of feed, per cent. of solids in pulp | 11.93 | 7.15 | 7.71 |
| Assay per cent. copper in feed | 4.86 | 3.23 | 2.63 |
| Tons of concentrate produced per 24 hr | 3.493 | 1.661 | 0.788 |
| Density of concentrate in grams per gallon | 1.39 | 90.0 | 44.0 |
| Density of concentrate, per cent. solids in pulp | 3.64 | 2.36 | 1.19 |
| Assay per cent. copper in concentrate | 17.0 | 16.08 | 13.9 |
| Assay per cent. insoluble in concentrate | 29.6 | 26.0 | 35.3 |
| Assay per cent. FeO in concentrate | 26.4 | 27.50 | 25.6 |
| Per cent. of copper fed, recovered in concentrate | 60.52 | 58.81 | 66.56 |
| Ratio of concentration, tons into one | 5.78 | 8.45 | 7.95 |

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| | | | |
|---|---|---|---|
| Feed-side tailing solids, per cent. of total solids fed | 42.81 | 56.11 | 8.29 |
| Feed-side tailing, assay per cent. copper | 2.2 | 1.43 | 1.93 |
| Feed-side tailing copper, per cent. of total copper fed | 19.36 | 24.78 | 6.08 |
| Wash-side tailing solids, per cent. of total solids fed | 39.90 | 32.06 | 79.12 |
| Wash-side tailing, assay per cent. copper | 2.45 | 1.65 | 0.91 |
| Wash-side tailing copper, per cent. of total copper fed | 20.12 | 16.41 | 27.36 |
| Gallons of dressing water used per 24 hr | 13,150 | 19,992 | 21,151 |
| Gallons of dressing water used per ton of feed | 651 | 1,424 | 3,378 |
| Gallons of spray water used per 24 hr | 22,106 | 16,447 | 15,598 |
| Gallons of spray water used per ton of feed. | 1,094 | 1,171 | 2,491 |
| Pounds of feed water per minute per foot of total circumference | 3.88 | 4.74 | 1.95 |
| Pounds of dressing water per minute per foot of total circumference | 1.34 | 2.15 | 2.11 |
| Pounds of spray water per minute per foot of total circumference | 2.40 | 1.79 | 1.69 |
| Revolutions of table per minute | 2 | 2 | 2 |
| Number of sprays operating | 8 | 8 | 8 |

The high-tonnage test results showed a fair recovery of copper in a fair grade of concentrate, the recovery of copper being higher than in the medium-tonnage test immediately following although the grade of concentrate is not so good. The lower copper recovery of the medium-tonnage test is due to the fact that a lower grade of feed was treated, and the cleaner concentrate to the greater quantity of dressing water used. In the low-tonnage test a higher recovery of copper was made, but the grade of concentrate produced was much poorer, the grade of the feed treated being much lower.

In closing a report on the above tests the writer suggested that the conoidal deck round table be given a trial in the slime plant as it presented possibilities of development into a useful table for the treatment of very fine slime.

In the slime-plant practice prevailing at this time, the conical deck round tables were being used as roughing tables, the tables being fed over their entire surface, with the exception of the part reserved for washing off concentrate, and discharging tailing material from the total perimeter with the above exception. The round-table decks used had a slope of 1.25 in. per foot. This slope, with a favorable density of feed, had been proved to give a maximum recovery of free-mineral grains, the coarser grains being recovered on the upper part of the deck and the very minute grains toward the perimeter of the deck. In recovering the very minute mineral grains, however, the tables also recovered a large amount of gangue material, this having been determined by qualitative experiments carried out at the suggestion of C. W. Goodale, in which samples

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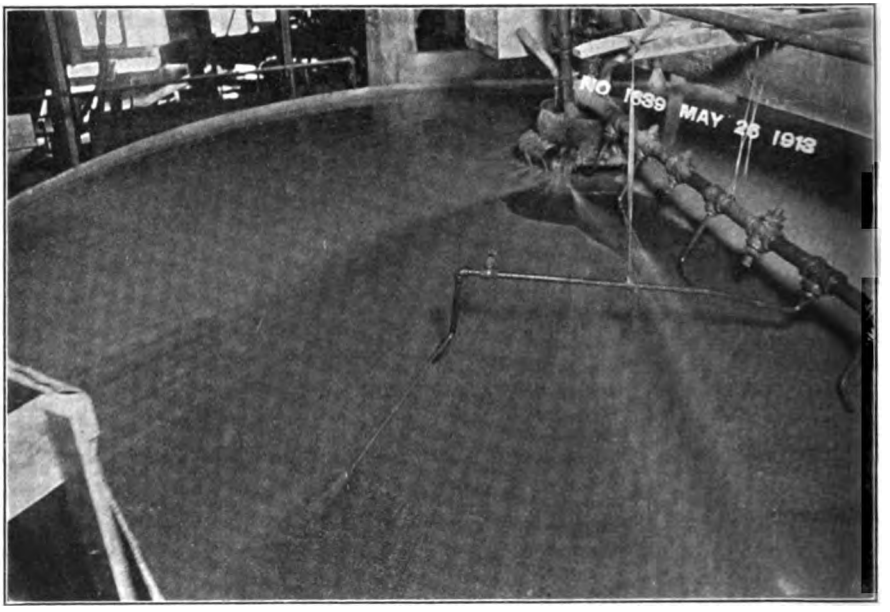
of the concentrate on the deck surface were taken for each foot from the center of the table to the perimeter. These experiments, which will be described later, developed the fact that the concentrate deposited on the lower half of the deck was of a very low grade, and within 12 in. of the perimeter of the deck was practically a tailing material. In the opinion of the writer, the conoidal deck with its very gradual increase in slope presented possibilities for overcoming the above defect. The first Munroe conoidal deck round table with cement surface (Fig. 2) was installed in the slime plant and ready for operation on Dec. 3, 1912. The




Deck surface, cement mortar (2 parts sand to 1 part cement). Slope of chord of arc = 1.25 in. per foot. Diameter of deck, 17 ft. Table operating to produce a finished concentrate, a middling for re-treatment, and a low-grade tailing product. One revolution in 16 min.

FIG. 2.—CONOIDAL DECK ROUND TABLE, GREAT FALLS. CONOIDAL DECK NO. 1.

table was installed with a total slope of 6° (1.25 in. per foot), as, from our preliminary experiments made in section No. 5, this slope seemed to be the most satisfactory. Hence, the chord of the arc of the conoidal deck made the same angle with the horizontal as the surface of the conical deck.

The cement mortar used for the deck surface consisted of two parts of sand to one of cement, the surface being given a rough finish similar to the surface of the steel-frame conical decks. The conoidal deck was laid on one of the wooden decks in the slime plant, the following figures representing the cost of the work and the weight of materials used:

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| Cost | | Weight of Material Used | Pounds. |
|---|---|---|---|
| Sand, 2,890 lb. @ 17c. per ton | $0.25 | Cement, 14 sacks @ 93 lb. per sack | 1,302 |
| Cement, 14 sacks @ 70c. per sack | 9.80 | Sand at 3,000 lb. per cubic yard | 2,890 |
| Laborers, 4 days @ $3.25 per day | 13.00 | | |
| Drayage | 2.00 | Total | 4,192 |
| Total | $25.05 | | |

The total slope of 6° which was given to the conoidal deck surface gave the following slope values for each foot of deck surface from the center of the table to the perimeter.

| | Degrees and Minutes to Nearest Minute | Inches per Foot to Nearest $\frac{1}{8}$ in. |
|---|---|---|
| First foot | 4° 30′ to 4° 53′ | $\frac{1}{8}$ to 1 |
| Second foot | 4° 53′ to 5° 15′ | 1 to $1\frac{1}{8}$ |
| Third foot | 5° 15′ to 5° 38′ | $1\frac{1}{8}$ to $1\frac{1}{4}$ |
| Fourth foot | 5° 38′ to 6° 0′ | $1\frac{1}{4}$ to $1\frac{1}{2}$ |
| Fifth foot | 6° 0′ to 6° 23′ | $1\frac{1}{2}$ to $1\frac{5}{8}$ |
| Sixth foot | 6° 23′ to 6° 45′ | $1\frac{5}{8}$ to $1\frac{3}{4}$ |
| Seventh foot | 6° 45′ to 7° 8′ | $1\frac{3}{4}$ to $1\frac{7}{8}$ |
| Eighth foot | 7° 8′ to 7° 30′ | $1\frac{7}{8}$ to 2 |

The deck had a diameter of 17 ft. and an available radial width of working surface of 7 ft. 3 in. The table was timed to make 1 rev. in 20.5 min., or 70.24 rev. per 24 hr.

This table was tested against a steel-frame cement-deck table of the following specifications:

The table tested was the upper deck of the steel-frame four-deck conical table. This deck had a slope of 1.25 in. per foot and the cement surface of the deck had a similar finish to that of the conoidal deck. The deck had a diameter of 18 ft. and an available radial width of working surface of 7 ft. 9 in. The table made 1 rev. in 16 min., or 90 rev. per 24 hr.

During the months of December, 1912, and January, 1913, a large number of comparative tests were carried out on the conoidal and the conical decks, the details of which, although very interesting, the writer is obliged to omit in order to keep the length of this paper within reasonable limits. During this period the tables were operated both as roughing tables producing a waste tailing product and a rough concentrate for re-treatment on Deister tables, and as finishing tables producing a waste tailing, a middling for re-treatment on Deister tables, and a finished concentrate, both systems of operation being carried out under various feed-density conditions and various tonnage conditions.

The operation of the tables to produce a finished concentrate, a middling for re-treatment, and a waste tailing resulted from a consideration of

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the assay results obtained on samples of rough concentrate taken at regular intervals between the center of the deck and its perimeter. From these results it was evident that a considerable percentage of the rough concentrate could be recovered in a concentrate which would be equal in grade to the finished concentrate turned out by the plant after re-treating the total round-table concentrate on Deister tables.

The adoption of this latter system would mean a reduction of the load on the Deister tables and a lowering of the re-treatment loss.

The middling for re-treatment was washed off the lower 3 ft. 6 in. to 4 ft. 6 in. of the deck, by a jet of water, into a separate launder which immediately preceded the finished concentrate launder. The nozzle used for directing the jet of water was attached to a flexible connection so that the amount of material produced for re-treatment could be varied as desired.

Another method used for producing a clean concentrate and a middling for re-treatment was as follows:

The corner of the feed box nearest to and preceding the concentrate wash-off jets was partitioned off and used as a small wash-water box, fresh water being added at this point and allowed to flow over the concentrate just before it reached the wash-off sprays; the material washed off the deck in this manner was sent to the Deister tables for re-treatment. Either method gives very satisfactory results.

*Conclusions from Preliminary Tests on Conoidal and Conical Decks*

It was evident from the results obtained both in the roughing tests and in the finishing tests that a conoidal deck with an overall slope of 1.25 in. per foot and a conical deck with an overall slope of 1.25 in. per foot will not yield similar end results under similar feed conditions, the conoidal deck requiring a denser feed pulp.

The tabular comparison (p. 1961) illustrates very clearly that if we feed a dense pulp to the conoidal deck and a more dilute pulp to the conical deck, practically similar end results may be expected; this refers to the tables as installed at this time. To produce similar end results it was found that the feed to the conoidal deck must be thickened to a density about 75 per cent. greater than that of the feed to the conical deck.

Since it might not be found advisable to thicken the table feed to a density of 500 g. of solids per gallon of pulp (12.2 per cent. solids) the next best thing to do would be to flatten the conoidal deck; that is, to reduce the overall slope from 1.25 in. per foot to 1.125 or 1 in. per foot, the latter slope, in the opinion of the writer, being the extreme limit in this direction for the class of feed which was being treated in the slime plant at this time.

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*Comparative Results of Roughing Tests on Conoidal and Conical Decks*

| | Name of Deck | | | | |
|---|---|---|---|---|---|
| | Conoidal | | Conical | | |
| | A | H | B | C | D |
| Tons dry solids treated per 24 hr..... | 7.979 | 6.586 | 6.146 | 6.913 | 8.649 |
| Density of feed in grams per gallon.... | 469 | 501 | 287 | 295 | 280 |
| Assay per cent. copper in feed..... | 2.69 | 2.50 | 2.34 | 2.44 | 2.20 |
| Tons concentrate produced per 24 hr.. | 2.243 | 2.180 | 1.711 | 1.890 | 2.438 |
| Assay per cent. copper in concentrate.. | 7.11 | 5.53 | 6.05 | 6.88 | 5.84 |
| Assay per cent. insoluble in concentrate. | 67.0 | 73.1 | 71.7 | 64.5 | 71.2 |
| Assay per cent. FeO in concentrate.... | 11.8 | 9.2 | 9.6 | 12.8 | 10.4 |
| Per cent. of copper fed, recovered in concentrate..... | 74.36 | 73.03 | 71.87 | 76.92 | 74.80 |
| Ratio of concentration, tons into one.. | 3.557 | 3.021 | 3.592 | 3.658 | 3.547 |
| Ratio of enrichment (copper)..... | 2.64 | 1.37 | 2.59 | 2.8 | 2.65 |
| Tons north-side tailing produced per 24 hr..... | 2.071 | 2.086 | 2.496 | 2.062 | 3.183 |
| Assay per cent. copper..... | 1.01 | 0.98 | 0.96 | 0.75 | 0.77 |
| Tons south-side tailing produced per 24 hr..... | 3.664 | 2.320 | 1.938 | 2.960 | 3.029 |
| Assay per cent. copper..... | 0.93 | 1.03 | 0.85 | 0.79 | 0.78 |
| Tons total tailing produced per 24 hr.. | 5.735 | 4.406 | 4.434 | 5.022 | 6.212 |
| Assay per cent. copper..... | 0.95 | 1.01 | 0.91 | 0.78 | 0.77 |
| Per cent. of copper fed, lost in north-side tailing..... | 9.79 | 12.42 | 16.67 | 9.17 | 12.86 |
| Per cent. of copper fed, lost in south-side tailing..... | 15.85 | 14.55 | 11.46 | 13.91 | 12.34 |
| Per cent. of copper fed, lost in total tailing..... | 25.64 | 26.97 | 28.13 | 23.08 | 25.20 |
| Gallons of fresh water used per 24 hr.. | 8,590 | 10,025 | 13,821 | 13,122 | 11,891 |
| Gallons of fresh water used per ton fed | 1,076 | 1,522 | 2,249 | 1,898 | 1,375 |

NOTE.—North-side tailing = tailing usually produced from feed side. South-side tailing = tailing usually produced from wash side.

Under the "deck slope" conditions of the tests, with feed conditions similar, *the tables operating to produce a rough concentrate and a waste tailing*, the following end results are obtained:

(*a*) The conoidal deck will deliver from 17 to 25 per cent. less of the feed solids in the concentrate than the conical deck.

(*b*) The concentrate produced by the conoidal deck will assay from 1.5 to 3.0 per cent. higher in copper, 6 to 12 per cent. less in insoluble and 3 to 5 per cent. higher in iron than the concentrate produced by the conical deck.

(*c*) The concentrate produced by the conoidal deck will contain from 6.5 to 8.0 per cent. less of the copper fed to the table than the concentrate produced by the conical deck.

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The same relative conditions maintain for any other method of operating the tables, hence, if we wish to operate conoidal and conical decks in parallel under similar feed conditions to produce similar end results: For any given density of feed the conoidal decks must be installed with a less overall slope than the conical decks.

It was considered possible that by flattening the chord of the conoidal deck by $\frac{1}{16}$ in. per foot at a time, using an experimental deck section, a slope might be found which, while giving a recovery equal to that made by the conical deck, would produce a better grade of concentrate than is produced by the conical deck. The writer considered that the building of a conoidal deck for regular operation with an overall slope of 1.125 in. per foot was justifiable and this was done later.

A conoidal deck with an overall slope of 1.0 in. per foot was first constructed, this table doing very satisfactory work if the table feed did not get above medium density and did not contain more than 2 or 3 per cent. of fine sand. A preliminary test was made on this table on Feb. 28, 1913, when the feed to the table had a density of 468 g. of solids per gallon of pulp; the density of feed was too high, the table making an extraction of 85.6 per cent. of the copper fed, in a concentrate assaying 4.12 per cent. copper. Other tests were made later on this table using less dense feed with better results, as presented in the following pages.

In a general way it would seem that it makes very little difference which type of deck is used as long as we use a slope on each type of deck which is most suitable for the average density of feed handled at the slime plant. This conclusion is subject, however, to results to be obtained by using a flatter conoidal deck, as stated on a preceding page, as it was considered possible that the recovery made by the conical deck might be reached in a better grade of concentrate.

In tests *A* and *H*, when a high density of feed was used, averaging 480 g. of solids per gallon of pulp, the writer considered that the work of the conoidal deck was superior to that of the conical deck in spite of the fact that the former made a recovery of copper lower by 6.82 per cent. than the other, 73.78 against 80.70 per cent. (averages of tests *A* and *H*).

This was on account of the fact that the conoidal deck sent to tailings 20.4 per cent. more of the feed solids than the conical deck, the tailings having similar assay values. This means that the conoidal deck produced much less material for re-treatment, the additional loss from this deck being entirely due to additional tailing production and this tailing product being of a lower grade than the tailing made in the re-treatment system. An examination of the following figures will make this point clear.

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| Test | Pounds Total Tailing Produced per 24 Hr. | Per Cent. of Solids in Feed | Assay Per Cent. Copper | Pounds Copper per 24 Hr. | Per Cent. of Copper in Feed |
|---|---|---|---|---|---|
| | | *Conoidal Deck* | | | |
| *A* | 11,472 | 71.89 | 0.96 | 110 | 25.64 |
| *H* | 8,812 | 66.90 | 1.01 | 89 | 26.97 |
| Total and Average .... | 20,284 | 69.63 | 0.98 | 199 | 26.22 |
| | | *Conical Deck* | | | |
| *A* | 7,067 | 48.10 | 1.05 | 74 | 19.07 |
| *H* | 7,133 | 50.41 | 0.98 | 70 | 19.55 |
| Total and Average .... | 14,200 | 49.23 | 1.01 | 144 | 19.30 |

The results obtained from both the conoidal and the conical deck indicated that considerable improvement could be made in the round-table practice if the slimes were roughly classified preceding table treatment.

The main difference in the work of the two tables under discussion, both having an overall slope of 1.25 in. per foot, may be explained as follows:

The conical deck, as installed, is capable of retaining upon its surface a considerable percentage of the very finest mineral grains contained in the slime, but in doing so also retains a large percentage of the fine sand, the result being that a concentrate high in insoluble material is produced and a high recovery of copper made.

The conoidal deck, as installed, cannot retain upon its surface a high percentage of the very finest mineral grains, but in losing these fine mineral grains the deck also gets rid of a high percentage of the fine sand, the result being that a clean grade of concentrate is produced, but the recovery of copper made is lower than that made by the conical deck.

From the behavior of the two types of table deck under varying feed densities it would at first seem that the work of the two types of tables under constant feed conditions could be made similar either by increasing the slope of the conical deck or by flattening the conoidal deck. The test data on the second conoidal deck installed in the slime plant will threw some further light on this subject.

Professor Munroe's idea in adopting the conoidal deck surface, as previously stated, was to make the movement of the pulp and the concentration conditions uniform in spite of the divergence of the water as it flows from the center toward the circumference, which was accomplished by gradually increasing the slope of the table and, therefore, the speed of the film of water. This condition allows the use of a flatter slope than would be practical on a conical deck.

A heavy deposition occurs at the center of the table, but with a slope

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at this point of 4° 30′ (½ in. per foot), which is the slope of the table tested, this deposition has been very clean mineral. The writer considers that it would be safe to make the slope at the center of the table 4° or possibly a shade less if the density of the feed is kept at about 350 g. of solids per gallon of pulp (9.25 per cent. solids). With a 3° slope at the center of the table, as suggested by Professor Munroe, a good deal of fine sand is deposited with the mineral if the slime contains 2 or 3 per cent. of on 200-mesh material, and it is difficult to clean this concentrate with dressing water.

The second conoidal deck installed at the slime plant, known as conoidal deck No. 2, was given an overall slope of 4° 45′, or 1 in. per foot, this giving a deck surface inclination varying from 3° 30′ at the center to 6° 15′ at the circumference, or very nearly the slope which is recommended by Professor Munroe.

The above deck was tested during the period April 26 to 29, inclusive, 1913, in parallel with conoidal deck No. 1, the steel-frame conical deck, and the step-deck previously described in this paper.

During the test an endeavor was made to keep the tonnage fed to each table, and the density of the feed uniform.

The comparative results obtained from the tests made on the four tables are shown below:

*Comparison of Data Obtained on Four Round Tables*

| | Conoidal Deck No. 1 | Conoidal Deck No. 2 | Step Deck | Conical Deck |
|---|---|---|---|---|
| Feed: | | | | |
| Gallons of feed pulp per 24 hr | 18,272 | 18,362 | 17,705 | 17,106 |
| Tons dry solids in feed per 24 hr | 9.909 | 9.736 | 9.700 | 9.296 |
| Density of feed in grams per gallon | 492 | 481 | 497 | 493 |
| Density of feed, per cent. solids | 12.0 | 11.5 | 12.1 | 12.0 |
| Assay per cent. copper in feed | 2.91 | 2.94 | 2.74 | 2.71 |
| Pounds of copper fed per 24 hr | 576 | 572 | 532 | 503 |
| Concentrate No. 1: | | | | |
| Gallons per 24 hr | 9,177 | 11,562 | 7,957 | 14,694 |
| Tons dry solids per 24 hr | 1.379 | 2.940 | 1.604 | 1.403 |
| Density in grams per gallon | 136 | 231 | 183 | 87 |
| Density in per cent. solids | 2.9 | 5.3 | 4.3 | 2.2 |
| Per cent. of solids in feed | 13.91 | 30.19 | 16.53 | 15.09 |
| Assay per cent. copper | 13.46 | 7.26 | 8.42 | 10.87 |
| Assay per cent. insoluble | 37.8 | 64.9 | 59.3 | 48.7 |
| Assay per cent. FeO | 23.6 | 12.4 | 14.9 | 18.8 |
| Pounds of copper per 24 hr | 371 | 427 | 270 | 305 |
| Per cent. of copper in feed | 64.41 | 74.65 | 50.75 | 60.64 |
| Concentrate No. 2 (middling): | | | | |
| Gallons per 24 hr | 2,048 | 4,543 | 10,274 | 6,205 |
| Tons dry solids per 24 hr | 0.150 | 0.680 | 0.637 | 0.952 |
| Density in grams per gallon | 66 | 136 | 56 | 139 |

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| | Conoidal Deck No. 1 | Conoidal Deck No. 2 | Step Deck | Conical Deck |
|---|---|---|---|---|
| Concentrate No. 2 (middling)—*Continued:* | | | | |
| Density in per cent. solids (approximate) | 1.7 | 3.5 | 1.6 | 3.6 |
| Per cent. of solids in feed | 1.51 | 6.99 | 6.55 | 10.23 |
| Assay per cent. copper | 4.35 | 1.91 | 7.16 | 2.99 |
| Assay per cent. insoluble | 76.6 | 88.1 | 65.3 | 84.0 |
| Assay per cent. FeO | 7.0 | 2.8 | 11.8 | 4.3 |
| Pounds of copper per 24 hr | 13 | 26 | 91 | 57 |
| Per cent. of copper in feed | 2.26 | 4.55 | 17.11 | 11.33 |
| Total concentrate: | | | | |
| Gallons per 24 hr | 11,225 | 16,105 | 18,231 | 20,899 |
| Tons dry solids per 24 hr | 1.528 | 3.620 | 2.239 | 2.354 |
| Density in grams per gallon | 123 | 204 | 111 | 102 |
| Density in per cent. solids (approximate) | 3.20 | 5.1 | 2.9 | 2.7 |
| Per cent. of solids in feed | 15.42 | 37.18 | 23.08 | 25.32 |
| Assay per cent copper | 12.56 | 6.26 | 8.06 | 7.69 |
| Assay per cent. insoluble | 41.6 | 69.3 | 61.0 | 63.0 |
| Assay per cent. FeO | 22.0 | 10.6 | 14.0 | 12.9 |
| Pounds of copper per 24 hr | 384 | 453 | 361 | 362 |
| Per cent. of copper in feed | 66.67 | 79.20 | 67.86 | 71.97 |
| North-side tailing: | | | | |
| Gallons per 24 hr | 10,208 | 8,995 | 9,212 | 11,502 |
| Tons dry solids per 24 hr | 4.056 | 2.726 | 2.536 | 3.616 |
| Density in grams per gallon | 360 | 275 | 250 | 285 |
| Density in per cent. solids (approximate) | 9.0 | 7.0 | 6.35 | 7.3 |
| Per cent. of solids in feed | 40.92 | 28.0 | 26.14 | 38.90 |
| Assay per cent. copper | 1.08 | 1.08 | 1.16 | 1.05 |
| Pounds of copper per 24 hr | 88 | 59 | 59 | 76 |
| Per cent. of copper in feed | 15.28 | 10.31 | 11.09 | 15.11 |
| South-side tailing: | | | | |
| Gallons per 24 hr | 10,801 | 9,009 | 13,838 | 9,606 |
| Tons dry solids per 24 hr | 4.326 | 3.390 | 4.925 | 3.326 |
| Density in grams per gallon | 363 | 341 | 323 | 314 |
| Density in per cent. solids (approximate) | 9.0 | 8.5 | 8.1 | 7.8 |
| Per cent. of solids in feed | 43.66 | 34.82 | 50.78 | 35.78 |
| Assay per cent. copper | 1.20 | 0.88 | 1.14 | 0.98 |
| Pounds of copper per 24 hr | 104 | 60 | 112 | 65 |
| Per cent. of copper in feed | 18.05 | 10.49 | 21.05 | 12.92 |
| Total tailing: | | | | |
| Gallons per 24 hr | 21,009 | 18,004 | 23,050 | 21,108 |
| Tons dry solids per 24 hr | 8.382 | 6.116 | 7.461 | 6.942 |
| Density in grams per gallon | 362 | 308 | 294 | 298 |
| Density in per cent. solids (approximate) | 9.0 | 7.8 | 7.5 | 7.5 |
| Per cent. of solids in feed | 84.58 | 62.82 | 76.92 | 74.68 |
| Assay per cent. copper | 1.14 | 0.97 | 1.15 | 1.01 |
| Pounds of copper per 24 hr | 192 | 119 | 171 | 141 |
| Per cent. of copper in feed | 33.33 | 20.80 | 32.14 | 28.03 |

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*Discussion of Results of Four-Day Test on Conical and Conoidal Decks*

The four-day test results place the decks in the following order in regard to efficiency.

| In Regard to Copper Recovery | Based on Total Concentrate Per Cent. Recovery | In Regard to Grade of Concentrate |
|---|---|---|
| Conoidal deck No. 2 | 79.20 | Conoidal deck No. 1, 12.6 per cent. Cu; 41.6 per cent. insol.; 22.0 per cent. FeO. |
| Conical deck | 71.97 | Step deck, 8.1 per cent. Cu; 61.0 per cent. insol.; 12.9 per cent. FeO. |
| Step deck | 67.86 | Conical deck, 7.7 per cent. Cu; 63.0 per cent. insol.; 12.9 per cent. FeO. |
| Conoidal deck No. 1 | 66.67 | Conoidal deck No. 2, 6.3 per cent. Cu; 69.3 per cent. insol.; 10.6 per cent. FeO. |

| In Regard to Copper Recovery | Based on No. 1 Concentrate Per Cent. Recovery | In Regard to Grade of Concentrate |
|---|---|---|
| Conoidal deck No. 2 | 74.65 | Conoidal deck No. 1, 13.5 per cent. Cu; 37.8 per cent. insol.; 23.6 per cent. FeO. |
| Conoidal deck No. 1 | 64.41 | Conical deck, 10.9 per cent. Cu; 47.8 per cent. insol.; 18.8 per cent. FeO. |
| Conical deck | 60.64 | Step deck, 8.4 per cent. Cu; 59.3 per cent. insol.; 14.9 per cent. FeO. |
| Step deck | 50.75 | Conoidal deck No. 2, 7.3 per cent. Cu; 64.9 per cent., insol.; 12.4 per cent. FeO. |

Before discussing the above results some figures will be given showing the fresh water added on the tables during the test period.

| | Conoidal Deck No. 1 | Conoidal Deck No. 2 | Step Deck | Conical Deck |
|---|---|---|---|---|
| Gallons of frerh water used per 24 hr..... | 13,960 | 15,750 | 23,580 | 24,900 |
| Gallons of fresh water used per ton fed..... | 1,410 | 1,620 | 2,430 | 2,680 |
| Gallons of No. 1 concentrate produced..... | 9,180 | 11,560 | 7,960 | 14,690 |
| Gallons of No. 2 concentrate produced..... | 2,050 | 4,540 | 10,270 | 6,210 |
| Per cent. of added water to wash off No. 1 concentrate..... | 66.0 | 73.0 | 34.0 | 59.0 |
| Per cent. of added water used as dressing water. | 34.0 | 27.0 | 66.0 | 41.0 |

Most of the water used as dressing water finds its way into No. 2 concentrate (middling), some of it, however, going to south-side tailing and some to concentrate No. 1.

The order of the decks as regards efficiency in copper recovery made in concentrate and as regards grade of concentrate is about what should be expected. Had less dressing water been used per ton fed to the conical deck the copper recovery made by this deck would undoubtedly have been

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raised to the level of the recovery made by conoidal deck No. 2, but the grade of the concentrate would have been correspondingly lowered. The gallons of dressing water used per ton fed and some further comparative data are shown by the following figures.

| | Conoidal Deck No. 1 | Conoidal Deck No. 2 | Step Deck | Conical Deck |
|---|---|---|---|---|
| Gallons of dressing water used | 4,780 | 4,190 | 15,620 | 10,210 |
| Tons fed | 9.909 | 9.735 | 9.699 | 9.296 |
| Gallons of dressing water used per ton | 483 | 430 | 1,610 | 958 |
| Ratio of concentration, tons into one | 6.485 | 2.689 | 4.334 | 3.949 |
| Ratio of enrichment (copper) | 4.316 | 2.129 | 2.942 | 2.838 |
| Copper in concentrate No. 1, per cent. of total concentrate, Cu | 96.6 | 94.2 | 74.8 | 84.2 |

It should be noted that the total concentrate produced by conoidal deck No. 1 (chord of arc on 1.25-in. slope) is of a much better grade than the No. 1 concentrate produced by any of the other decks, while the recovery of 66.7 per cent. of the copper made by this deck in total concentrate is greater than the recovery made by the other decks in No. 1 concentrate, with the exception of conoidal deck No. 2, which on account of its flat slope at the head of the deck, makes a 7 per cent. higher recovery in No. 1 concentrate than is made by conoidal deck No. 1 in total concentrate, the concentrate being, of course, of very much lower grade.

The total recovery made by conoidal deck No. 1 and the grade of concentrate produced will now be compared with the recovery in concentrate No. 1 made by conoidal deck No. 2 and the conical deck. As the step deck made a concentrate No. 2 of high enough grade to be considered in the finished-concentrate class, its recovery in total concentrate and grade of concentrate will also be placed in comparison.

| | Conoidal Deck No. 1 Total Concentrates | Step Deck | Conoidal No. 2 No. 1 | Conical Concentrates |
|---|---|---|---|---|
| Per cent. of copper fed, recovered. | 66.67 | 67.86 | 74.65 | 60.64 |
| Assay per cent. copper in concentrates | 12.56 | 6.26 | 7.26 | 10.87 |
| Assay per cent. insoluble in concentrates | 41.6 | 69.3 | 64.9 | 48.7 |
| Assay per cent. FeO in concentrates | 22.0 | 10.6 | 12.4 | 18.8 |

The No. 2 concentrate or middling made by conoidal deck No. 1 and by the step deck are the only ones which can be considered as concentrate, the concentrate No. 2 from conoidal deck No. 2 being of much lower grade than the feed, and the middling from the conical deck being of about the same grade as the feed.



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In May, 1913, conoidal deck No. 3 was installed in the slime plant. The deck was given a cement surface of similar finish to that on the other two conoidal decks installed in the slime plant.

This deck varied from the first two conoidal decks installed in that the chord of the arc was laid on a slope of 1.125 in. per foot instead of 1.25 in. per foot for conoidal deck No. 1, or 1 in. per foot for conoidal deck No. 2. This was the only variable.

The three conoidal decks and the steel-frame conical deck were tested in parallel during the eight-day period May 17 to 24, inclusive, 1913, the results obtained from these tests being presented below. The conoidal-deck results have been placed in the order of decreasing slope of deck surface. This arrangement brings out more clearly the differences in the work of the conoidal decks, and also places the results of conoidal deck No. 2 by the side of those of the conical deck, the work done by these tables being very similar.

*Comparative Results on Four Round Tables*

| | Conoidal | | | Conical |
|---|---|---|---|---|
| | No. 1 | No. 3 | No. 2 | |
| Slope of chord of arc in inches per foot | 1.25 | 1.125 | 1.00 | ..... |
| Feed: | | | | |
| Gallons of feed pulp per 24 hr | 14,546 | 15,427 | 15,280 | 16,341 |
| Tons of dry solids in feed per 24 hr | 6.526 | 6.887 | 6.882 | 7.007 |
| Density of feed in grams per gallon | 407 | 405 | 405 | 389 |
| Assay per cent. copper in feed | 2.67 | 2.66 | 2.46 | 2.37 |
| Pounds of copper fed per 24 hr | 349 | 367 | 336 | 332 |
| Concentrate No. 1: | | | | |
| Gallons per 24 hr | 10,522 | 11,672 | 10,523 | 13,562 |
| Tons dry solids per 24 hr | 0.821 | 1.280 | 1.643 | 1.489 |
| Density in grams per gallon | 71 | 99 | 142 | 100 |
| Per cent. of solids in feed | 12.58 | 18.59 | 24.08 | 21.24 |
| Assay per cent. copper | 13.03 | 9.84 | 7.03 | 7.56 |
| Assay per cent. insoluble | 39.6 | 53.4 | 65.5 | 63.3 |
| Assay per cent. FeO | 22.9 | 17.4 | 12.3 | 13.3 |
| Pounds of copper per 24 hr | 214 | 252 | 231 | 225 |
| Per cent. of copper in feed | 61.32 | 68.66 | 68.75 | 67.77 |
| Concentrate No. 2 (middling): | | | | |
| Gallons per 24 hr | 2,005 | 2,206 | 4,107 | 4,587 |
| Tons dry solids per 24 hr | 0.101 | 0.193 | 0.442 | 0.588 |
| Density in grams per gallon | 45 | 79 | 98 | 116 |
| Per cent. of solids in feed | 1.54 | 2.80 | 6.49 | 8.39 |
| Assay per cent. copper | 4.97 | 2.59 | 1.81 | 1.79 |
| Assay per cent. insoluble | 74.6 | 84.7 | 88.1 | 88.3 |
| Assay per cent. FeO | 8.0 | 4.1 | 2.8 | 2.7 |
| Pounds of copper per 24 hr | 10 | 10 | 16 | 21 |
| Per cent. of copper in feed | 2.86 | 2.73 | 4.76 | 6.33 |

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| | Conoidal | | | Conical |
|---|---|---|---|---|
| | No. 1 | No. 3 | No. 2 | |
| Total concentrate: | | | | |
| Gallons per 24 hr | 12,527 | 13,878 | 14,630 | 18,149 |
| Tons dry solids per 24 hr | 0.922 | 1.473 | 2.085 | 2.077 |
| Density in grams per gallon | 67 | 96 | 129 | 104 |
| Per cent. of solids in feed | 14.12 | 21.39 | 30.57 | 29.63 |
| Assay per cent. copper | 12.15 | 8.89 | 5.92 | 5.92 |
| Assay per cent. insoluble | 43.4 | 57.5 | 70.3 | 70.4 |
| Assay per cent. FeO | 21.3 | 15.6 | 10.3 | 10.4 |
| Pounds of copper per 24 hr | 224 | 262 | 247 | 246 |
| Per cent. of copper in feed | 64.18 | 71.39 | 73.51 | 74.10 |
| North-side tailing: | | | | |
| Gallons per 24 hr | 8,519 | 9,718 | 8,555 | 9,634 |
| Tons dry solids per 24 hr | 2.288 | 2.602 | 2.037 | 2.261 |
| Density in grams per gallon | 244 | 243 | 216 | 232 |
| Per cent. of solids in feed | 35.05 | 37.78 | 29.85 | 35.12 |
| Assay per cent. copper | 0.98 | 0.96 | 1.08 | 0.93 |
| Pounds of copper per 24 hr | 45 | 50 | 44 | 46 |
| Per cent. of copper in feed | 12.89 | 13.62 | 13.10 | 13.85 |
| South-side tailing: | | | | |
| Gallons per 24 hr | 10,852 | 9,289 | 10,593 | 9,351 |
| Tons dry solids per 24 hr | 3.317 | 2.812 | 2.700 | 2.470 |
| Density in grams per gallon | 277 | 275 | 231 | 240 |
| Per cent. of solids in feed | 50.83 | 40.83 | 39.58 | 35.25 |
| Assay per cent. copper | 1.21 | 0.98 | 0.83 | 0.81 |
| Pounds of copper per 24 hr | 80 | 55 | 45 | 40 |
| Per cent. of copper in feed | 22.93 | 14.99 | 13.39 | 12.05 |
| Total tailing: | | | | |
| Gallons per 24 hr | 19,371 | 19,007 | 19,148 | 18,985 |
| Tons dry solids per 24 hr | 5.605 | 5.414 | 4.737 | 4.931 |
| Density in grams per gallon | 262 | 258 | 224 | 236 |
| Per cent. of solids in feed | 85.88 | 78.61 | 69.43 | 70.37 |
| Assay per cent. copper | 1.11 | 0.97 | 0.94 | 0.87 |
| Pounds of copper per 24 hr | 125 | 105 | 89 | 86 |
| Per cent. of copper in feed | 35.82 | 28.61 | 26.49 | 25.90 |
| Ratio of concentration (No. 1 concentrate) | 7.9 | 5.4 | 4.2 | 4.7 |
| Ratio of concentration (total concentrate) | 7.1 | 4.7 | 3.3 | 3.4 |
| Ratio of enrichment, copper (No. 1 concentrate) | 4.9 | 3.7 | 2.8 | 3.2 |
| Ratio of enrichment, copper (total concentrate) | 4.6 | 3.3 | 2.4 | 2.5 |
| Gallons of fresh water used per 24 hr | 17,352 | 17,458 | 18,498 | 20,793 |
| Gallons of fresh water used per ton fed | 2,659 | 2,535 | 2,711 | 2,967 |

*Discussion of Results of Eight-Day Test*

The following tabulation shows that the test results arrange the different decks in just the order which should be expected both in regard to copper recovery and grade of concentrate produced.

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*Order of Deck Efficiency Based on Total Concentrate*

| In Regard to Copper Recovery | | In Regard to Grade of Concentrate | | | |
|---|---|---|---|---|---|
| | Recovery, Per Cent. | | Cu | Insol. | FeO |
| Conical deck, | 74.10 | Conoidal deck No. 1 | 12.15 | 43.14 | 21.3 |
| Conoidal deck No. 2 | 73.51 | Conoidal deck No. 3 | 8.89 | 57.5 | 15.6 |
| Conoidal deck No. 3 | 71.39 | Conoidal deck No. 2 | 5.92 | 70.3 | 10.3 |
| Conoidal deck No. 1 | 64.18 | Conical deck | 5.92 | 70.4 | 10.4 |

The work of the conical deck is practically identical with the work of the conoidal deck No. 2. The conoidal deck results illustrate very clearly the effect of the slope on the work of the table, the grade of concentrate produced being lowered by a decrease in slope while the recovery of copper is raised. Conoidal deck No. 3 with the chord of its arc laid on a slope of 1.125 in. per foot was designed to produce a cleaner grade of concentrate than the conical deck with its slope of 1.25 in. per foot, at the same time making practically the same recovery of copper. The tests showed that the results desired were very closely approximated, conoidal deck No. 3 recovering 71.39 per cent. of the copper fed in 21.39 per cent. of the total feed solids while the conical deck recovered 74.10 per cent. of the copper fed in 29.63 per cent. of the feed solids. In considering these results it must be remembered that the conical deck had a radial width 6 in. greater than the radial width of the conoidal deck and that, therefore, in order to make the results strictly comparable, the deposit on the lower 6 in. of the conical deck should have been washed into the tailing launder.

*Order of Deck Efficiency Based on Concentrate No. 1*

| In Regard to Copper Recovery | | In Regard to Grade of Concentrate | | | |
|---|---|---|---|---|---|
| | Recovery, Per Cent. | | Cu | Insol. | FeO |
| Conoidal deck No. 2 | 68.75 | Conoidal deck No. 1 | 13.03 | 39.6 | 22.9 |
| Conoidal deck No. 3 | 68.66 | Conoidal deck No. 3 | 9.84 | 53.4 | 17.4 |
| Conical deck | 67.77 | Conical deck | 7.56 | 63.3 | 13.3 |
| Conoidal deck No. 1 | 61.32 | Conoidal deck No. 2 | 7.03 | 65.5 | 12.3 |

The above tabulation shows that the conical deck and the conoidal deck No. 2 did very similar work in producing a finished concentrate,

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the conical deck making a slightly cleaner grade of concentrate and a slightly lower recovery of copper.

The tabulation also shows that conoidal deck No. 3 made a higher recovery in a much cleaner grade of finished concentrate than the conical deck, recovering 68.66 per cent. of the copper fed, contained in 18.59 per cent. of the original feed solids, while the conical deck recovered 67.77 per cent. of the copper fed, contained in 21.24 per cent. of the original feed solids.

After making a few tests on the first conoidal deck installed in the slime plant, the writer became impressed with the fact that the conoidal deck surface presented possibilities for producing a cleaner grade of concentrate than is produced by a conical deck with a slope of 1.25 in. per foot while at the same time making as high a recovery of copper, and, that a conoidal deck could be installed which would yield just such results as are recorded above for conoidal deck No. 3. The recovery made by a conical deck can, of course, be increased by a slight flattening of the slope, the increase in recovery being made at the cost of lowering the grade of the concentrate, or the grade of concentrate may be enriched by a slight increase of the slope, the enrichment being made at the cost of a lower recovery of copper. Thus, at first glance, it might appear that by merely juggling with deck slopes or feed densities, the feed density being fully as important as the slope, the results obtained upon any conoidal deck could be duplicated upon any conical deck, or *vice versa.* While the writer is not yet prepared to state emphatically that such is not the case, it may be stated that a conoidal deck can be installed which will make a cleaner grade of concentrate and as high a recovery of copper as a conical deck with a slope of 1.25 in. per foot. It may be possible that further experiment may develop the fact that a combination of the two types of deck surfaces will be beneficial, making the deck semi-conoidal, semi-conical, the upper part of the deck being conical.

### *Qualitative Sampling of Concentrates Across Conical Deck*

Early in August, 1912, the writer, at the suggestion of Mr. Goodale, obtained qualitative samples of the rough concentrate on the deck of the steel-frame conical table for each foot of deck surface from the center to the perimeter. The assay results on these samples showed that the concentrate deposit at the head of the deck was considerably richer than the finished concentrate produced by the slime plant, while the deposit nearer the perimeter of the deck was of very low grade. The table was tested on August 11, 12, and 13, being sampled quantitatively preceding the qualitative sampling of the concentrate, two or three revolutions of the table being allowed to elapse before the qualitative samples were taken.

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The density of the feed was increased with each day's sampling, the light density of the first day making a lighter tonnage of feed to the table than was the case on the two following days. The quantitative results of the tests will first be presented:

| | Date of Test | | |
|---|---|---|---|
| | Aug. 11 | Aug. 12 | Aug. 13 |
| Rate of feed, tons per 24 hr | 9.10 | 11.50 | 11.05 |
| Density of feed in grams per gallon | 386 | 454 | 482 |
| Assay, per cent. copper in feed | 2.00 | 2.50 | 2.44 |
| Tons of concentrate produced per 24 hr | 1.687 | 3.112 | 4.151 |
| Assay per cent. copper in concentrate | 6.05 | 6.45 | 4.70 |
| Assay per cent. insoluble in concentrate | 71.0 | 71.6 | 77.2 |
| Assay per cent. FeO in concentrate | 8.7 | 8.7 | 7.8 |
| Per cent. of copper fed, recovered in concentrate | 56.04 | 69.86 | 72.36 |
| Ratio of concentration, tons into one | 5.4 | 3.7 | 2.7 |
| Assay per cent. copper north tailing | 1.05 | 1.05 | 1.05 |
| Assay per cent. copper south tailing | 1.10 | 1.00 | 1.10 |
| Assay per cent. copper total tailing | 1.08 | 1.03 | 1.08 |
| Gallons of fresh water used per 24 hr | 12,803 | 10,800 | 12,126 |
| Gallons of fresh water used per tons fed | 1,407 | 939 | 1,097 |
| Gallons of fresh water used per ton concentrate | 7,589 | 3,470 | 2,921 |

On August 11, the table was receiving a low grade of feed and, therefore, making a lower percentage recovery of copper than would be made with a richer feed. The feed was also more dilute than on the two following days which may have helped the tailing loss to some extent. On August 12, a slightly richer and denser feed was treated and an increase made in the recovery of copper in concentrate, the grade of the concentrate being slightly better as regards copper value than that made on the preceding day.

On August 13, the table feed had about the same assay value as on the preceding day, but had a greater density. The percentage of copper in feed recovered in concentrate showed a further increase, chiefly due to the lower grade of concentrate produced. (In a test made on the above deck in May, 1911, a 73 per cent. recovery of copper was obtained in a concentrate assaying 7.03 per cent. copper, 69.1 per cent. insoluble, and 9.4 per cent. FeO. This recovery was made on a feed assaying 2.91 per cent. copper, and with a rate of feed of 6.2 tons per 24 hr., as against 11.05 tons per 24 hr. in the present test.)

The following tabulation shows the results obtained on the qualitative samples of rough concentrate.

During the test of August 13, the end of the feed box was partitioned off to prevent new feed from coming on the table at the head of the sampling line, therefore, in this case a uniform increase in copper and iron percentages and a uniform decrease in insoluble content of the concentrate was obtained from the perimeter to the feed box. The samples

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were taken along a line on the table immediately preceding the wash-off spray, so that the samples may be said to represent the rough concentrate produced by the table.

*Analyses of Rough Concentrate from Steel-Frame Conical Table*

| Sample. | Copper | | | Insoluble | | | Ferrous Oxide | | |
|---|---|---|---|---|---|---|---|---|---|
| | Aug. 11 | Aug. 12 | Aug. 13 | Aug. 11 | Aug. 12 | Aug. 13 | Aug. 11 | Aug. 12 | Aug. 13 |
| No. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| 1..... | 3.75 | 2.25 | 1.15 | 80.8 | 87.2 | 91.6 | 4.9 | 2.9 | 2.2 |
| 2..... | 5.5 | 2.9 | 2.15 | 74.0 | 85.2 | 87.6 | 7.5 | 3.3 | 3.6 |
| 3..... | 7.1 | 3.35 | 2.2 | 67.6 | 83.8 | 87.4 | 9.8 | 4.2 | 3.2 |
| 4..... | 8.1 | 6.95 | 2.9 | 63.0 | 70.6 | 84.4 | 11.6 | 9.1 | 4.5 |
| 5..... | 9.35 | 7.55 | 4.2 | 57.6 | 68.4 | 79.8 | 13.9 | 9.8 | 6.9 |
| 6..... | 10.2 | 12.3 | 6.0 | 54.4 | 48.4 | 72.8 | 14.7 | 17.8 | 8.6 |
| 7..... | 9.2[a] | 12.1[a] | 10.9 | 57.4[b] | 49.4[b] | 52.2 | 13.8[a] | 17.4[a] | 17.1 |
| 8..... | 7.35[a] | 11.45[a] | 13.2 | 64.8[b] | 52.6[b] | 40.8 | 11.3[a] | 16.3[a] | 22.0 |
| Total Conct. | 6.05 | 6.45 | 4.7 | 71.0 | 71.6 | 77.2 | 8.7 | 8.7 | 7.8 |

Sample No. 1 taken from first foot above perimeter of table; sample No. 8 taken from first foot below feed box.

[a] The lowering of the copper and FeO percentages at this point was caused by new feed coming on the table.

[b] The increase in insoluble content at this point was caused by new feed coming on the table.

The results obtained on August 13 are shown graphically in Fig. 3. The curves illustrate the results obtained when the new feed is excluded from the two upper samples.

The results showed that in using the conical round tables as roughing tables, which was the regular slime-plant practice at the time the samples were taken, a grade of concentrate was being produced on the upper part of the deck which was of a better grade than the usual run of finished concentrate, and that in removing this concentrate from the deck it was remixed with a large quantity of very siliceous material from the lower part of the deck.

Another set of qualitative samples was obtained during a two-day test made on August 26 and 28, 1912, the material on the deck surface being sampled just preceding the wash-off spray and also on a radial line diametrically opposite the wash-off spray. In the tabulation (p. 1975) "*A*" samples are those taken immediately preceding wash-off spray, and "*B*" samples those taken on a radius diametrically opposite the wash-off spray. Plus and minus signs indicate that results on "*B*" samples are greater or less than results on "*A*" samples.

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Another set of qualitative samples was taken on June 17, 1913, including samples taken on each of the three conoidal decks which were in operation at this time and on one of the steel-frame conical decks.

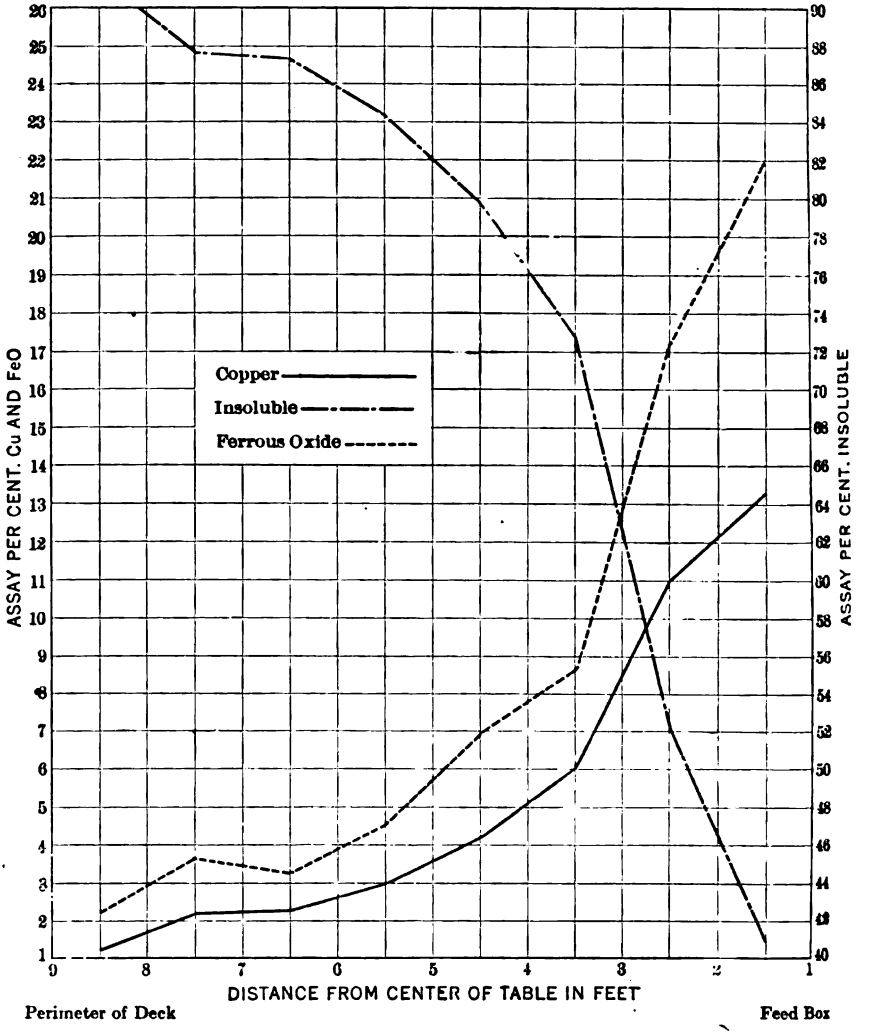


FIG. 3.—CURVES SHOWING GRADE OF CONCENTRATE FOR EACH FOOT OF DECK SURFACE OF STEEL-FRAME CEMENT-DECK ROUND TABLE

Insoluble curve, one-half scale of Cu and FeO curves.

It was at first intended to make a more extended test in this direction but it was afterward decided that enough experimental data had been secured to answer all practical purposes.

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*Comparison of Average Results of "A" and "B" Samples*

| Sample No. | Copper | | | Insoluble | | | Ferrous Oxide | | | Sulphur | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | A Per Cent. | B Per Cent. | Difference Per Cent. | A Per Cent. | B Per Cent. | Difference Per Cent. | A Per Cent. | B Per Cent. | Difference Per Cent. | A Per Cent. | B Per Cent. | Difference Per Cent. |
| 1 | 1.80 | 1.33 | −0.47 | 89.2 | 90.3 | +0.11 | 2.7 | 2.2 | −0.5 | 2.8 | 1.9 | −0.9 |
| 2 | 3.17 | 2.16 | −1.01 | 83.7 | 87.7 | +4.0 | 4.6 | 3.4 | −1.2 | 5.3 | 3.4 | −1.9 |
| 3 | 3.94 | 2.99 | −0.95 | 80.5 | 84.4 | +3.9 | 6.0 | 4.3 | −1.7 | 6.7 | 4.8 | −1.9 |
| 4 | 4.37 | 4.36 | −0.01 | 78.9 | 78.9 | ..... | 6.5 | 6.2 | −0.3 | 7.5 | 7.3 | −0.2 |
| 5 | 6.97 | 6.72 | −0.25 | 68.2 | 69.1 | +0.9 | 10.7 | 10.2 | −0.5 | 12.1 | 12.0 | −0.1 |
| 6 | 9.60 | 8.85 | −0.75 | 57.5 | 60.2 | +2.7 | 14.7 | 13.6 | −1.1 | 17.0 | 15.9 | −0.1 |
| 7 | 11.51 | 10.48 | −1.03 | 48.4 | 52.2 | +3.8 | 18.4 | 16.8 | −1.6 | 21.3 | 19.7 | −1.6 |
| 8 | 11.15 | 8.85 | −2.30 | 48.3 | 58.3 | +10.0 | 18.6 | 15.1 | −3.5 | 21.6 | 17.2 | −4.4 |

Samples are numbered from the perimeter of the deck to the feed box.

The following results, while not absolutely uniform, serve to illustrate the difference in the work of the tables:

| Sample No. | Conoidal Decks | | | | | | | | | Conical Deck | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | No. 1, slope 1.25 in. | | | No. 3, slope 1.125 in. | | | No. 2, slope 1.00 in. | | | Slope 1.25 in. | | |
| | Assay Per Cent. | | | | | | | | | | | |
| | Cu | Insol. | FeO | Cu | Insol. | FeO | Cu | Insol. | FeO | Cu | Insol. | FeO |
| 1 | 12.1 | 44.5 | 20.0 | 3.62 | 80.7 | 4.9 | 2.5 | 85.1 | 3.9 | 2.23 | 86.8 | 3.1 |
| 2 | 12.1 | 45.7 | 19.1 | 5.65 | 73.9 | 7.4 | 4.6 | 76.2 | 7.6 | 2.05 | 87.1 | 2.9 |
| 3 | 12.9 | 41.8 | 21.0 | 6.53 | 69.7 | 9.1 | 4.1 | 78.5 | 6.7 | 2.01 | 87.6 | 2.7 |
| 4 | 10.9 | 50.0 | 18.0 | 5.50 | 73.2 | 8.6 | 4.8 | 74.9 | 7.9 | 2.95 | 83.8 | 4.4 |
| 5 | 9.1 | 58.0 | 14.7 | 4.9 | 76.0 | 7.4 | 4.7 | 75.9 | 7.8 | 3.56 | 81.2 | 5.4 |
| 6 | 9.5 | 56.6 | 15.3 | 5.1 | 74.4 | 8.5 | 4.9 | 76.7 | 7.6 | 6.03 | 70.0 | 10.3 |
| 7 | 10.6 | 50.2 | 18.1 | 6.2 | 69.3 | 10.6 | 5.6 | 71.8 | 9.7 | 9.22 | 54.9 | 17.0 |
| 8 | 14.4 | 33.4 | 25.9 | 9.4 | 54.3 | 17.5 | 7.4 | 62.8 | 13.6 | 12.59 | 40.2 | 23.3 |

Samples are numbered from perimeter to center of table. At the time the first of above samples was taken, and for some two years previous to this time, all of the slime-plant round tables, with the exception of one table which was used for clearing up finishing-table, middling, were operated as roughing tables, the rough concentrate being cleaned on Deister No. 3 slime tables. This roughing system, or two-stage system, was installed under the supervision of G. M. Bates, slime-plant foreman, along the lines recommended by Dr. Richards some time previously, with the exception that the tables were set to make 3 rev. an hour instead of 1 rev.

This system gave very satisfactory results, but, of course, necessitated

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the re-treatment of a large amount of material and the continuous circulation of a large quantity of middling.

On a conical deck round table, operating as a roughing table, taking an average of all decks in operation, about 70 per cent. of the original feed copper is concentrated in about 30 per cent. of the original feed solids, and about 30 per cent. of the copper and 70 per cent. of the feed solids are sloughed off as tailing. Hence, under above conditions, about 30 per cent. of the original feed to the round tables requires re-treatment on finishing tables before a satisfactory concentrate can be obtained.

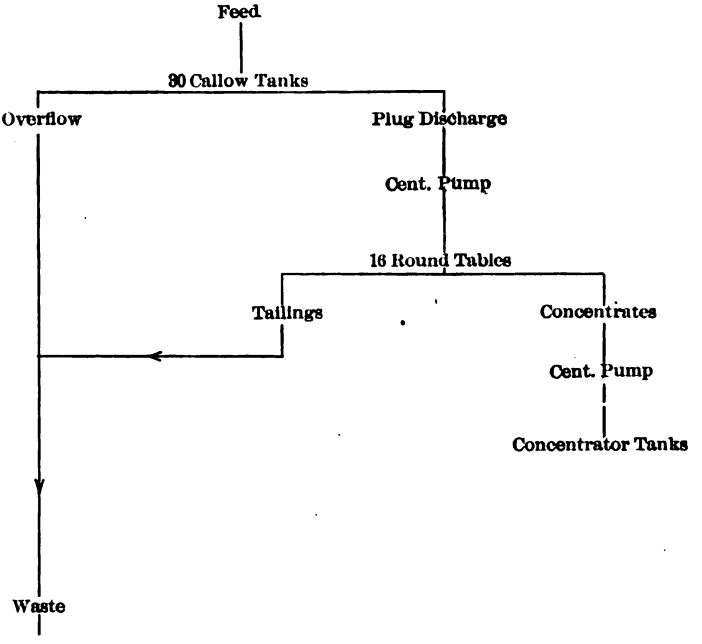


FIG. 4.—ORIGINAL SLIME-PLANT FLOW-SHEET.

After some further investigation of the grade of the rough concentrate on various parts of the deck surface of the round tables, it became evident to the writer that it was unnecessary to re-treat the whole of the rough concentrate, as by far the greater proportion of the concentrate copper was contained in a concentrate which was clean enough for a finished product.

Arrangements were therefore made to operate the round tables in the slime plant to make two products in addition to the tailing product, a finished concentrate and a rough concentrate or middling, the latter product only being subject to r-etreatment.

The necessary changes for operating the plant as noted above were completed under the supervision of Mr. Bates and the plant put into operation on Mar. 1, 1913.

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*Summary of Slime-Plant Practice*

The following summary, submitted by George M. Bates, foreman of the slime plant, illustrates the progress in slime-plant practice as the result of the application of the knowledge gained by sampling. Four successive

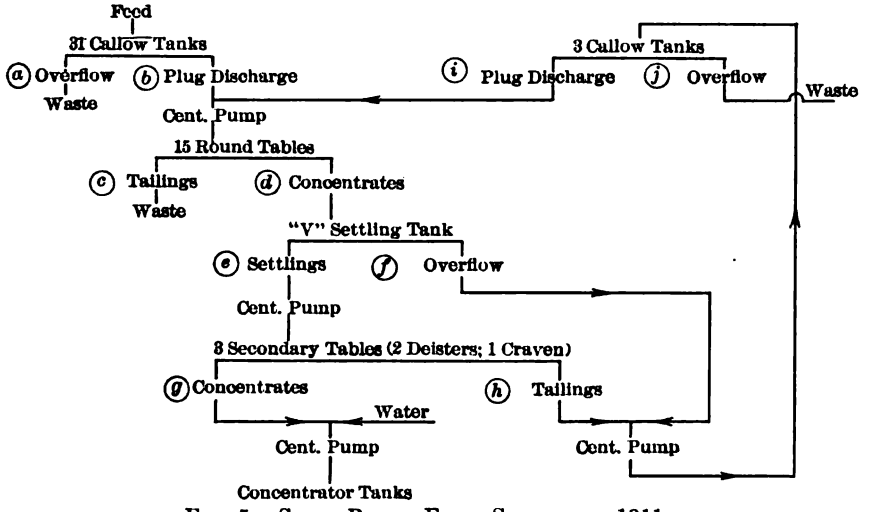


FIG. 5.—SLIME-PLANT FLOW SHEET FOR 1911.

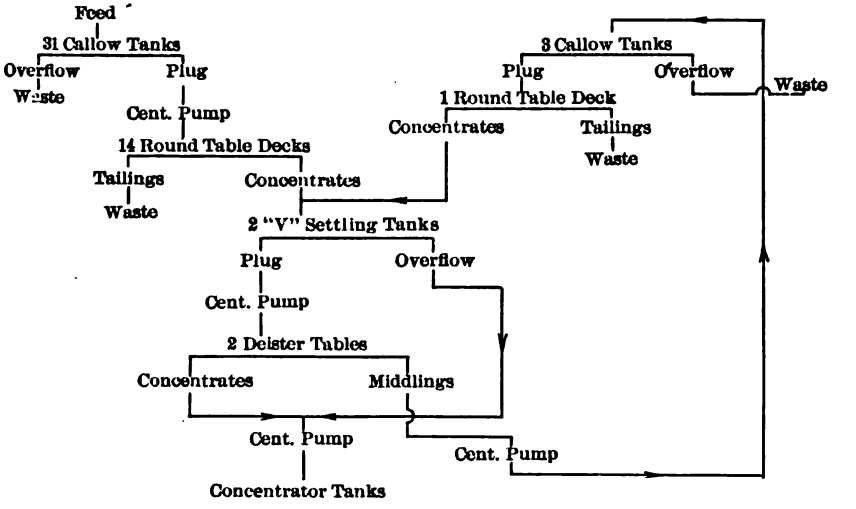


FIG. 6.—SLIME-PLANT FLOW SHEET FOR 1912.

flow sheets are presented, followed by a tabulation of figures to show the improvement at each step forward.

"Fig. 4 illustrates the original slime plant, as operated from 1905 to

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1910. Three kinds of deck surface were in use, viz., plain wood, linoleum, and smooth cement. All, however, were operated for two finished products, concentrate and tailing.

"Fig. 5 illustrates the so-called roughing system. The round-table decks were canvas covered and operated primarily for finished tailing. The rough concentrate was re-dressed on Deister tables producing finished concentrate and a middling which was dewatered and returned to the round-table system.

"In this connection the writer wishes to call attention to the fact that this roughing system is more than its name implies. Previous to this time the round tables had probably operated as true film-sizing tables. But, with the adoption of the roughing system, the tables ceased

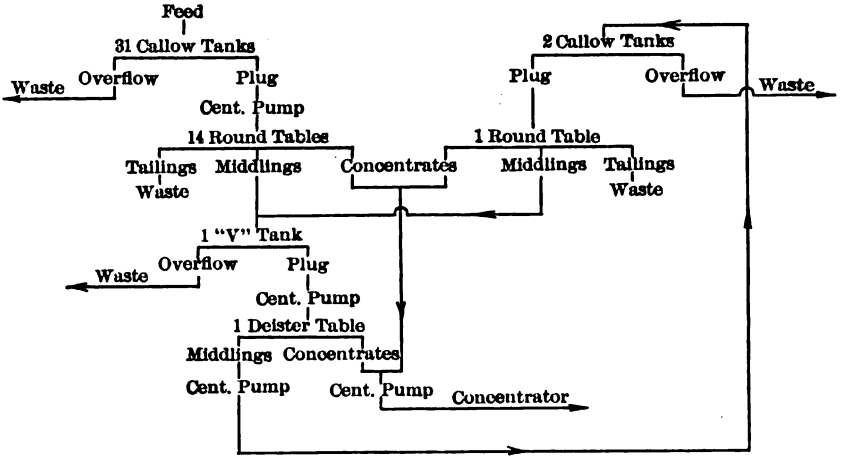


FIG. 7.—SLIME-PLANT FLOW SHEET ADOPTED MARCH, 1913.

to be film sizers, and became building tables. The feed began to bed on the tables, sometimes as much as $\frac{3}{8}$ in. in thickness. The rough canvas surface undoubtedly assisted in the rapid formation of this bed. After this bed was once established, however, the character of the deck surface at that point would seem to be of small moment, except in so far as it tended to hold back quartz grains and thus rendered washing more difficult. Whatever enrichment took place in the bed itself, as it was subjected to the action of fresh feed, must have been due to replacement of quartz grains by mineral grains.

"Fig. 6 illustrates the revised roughing system or building-table system, in which the return middling, instead of being returned to the round-table system, was subjected to an individual treatment on a single round table. The equipment at this time included several rough cement decks. The cement surface was adopted, not for any particular virtue in itself, but as an economical substitute for canvas. The cost of mainte-

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nance of the canvas surface was high. It was evident that if a cement surface of similar character could be produced it would be much cheaper to maintain. Upon trial it was found that the cement surface had the added advantage of requiring less wash water for the removal of the concentrate.

"Fig. 7 illustrates the latest slime-plant practice and a distinct step forward. The space samples taken at regular intervals from the center to the perimeter of the round table (described by Mr. Crowfoot) showed that the round table would produce a certain amount of high-grade concentrate directly. The operation of the plant was immediately changed to conform to this knowledge. The rough surface deck was retained (either canvas or cement) but the tables were equipped to make three products, viz., a finished tailing, a certain amount of finished concentrate, and a considerable amount of middling, the latter then enriched on a secondary table (Deister or James). The return middling was re-treated on a round table as before. While Mr. Crowfoot's tests indicated that a very high-grade concentrate, namely from 10 to 14 per cent. copper content, was possible under this system, owing to certain considerations of recovery and smelting costs, the production of a concentrate richer than 8 per cent. copper content has not been attempted.

"The following figures, corresponding to the flow sheets just presented, show that the progress in slime concentration has been real."

*Average Monthly Figures of Slime Plant*

| Flow sheet number | 1 | 2 | 3 | 4 |
|---|---|---|---|---|
| Year | 1908 | 1911 | 1912 | 1913[a] |
| Average number round tables | 15.5 | 14.0 | 14.5 | 14.9 |
| Table feed: | | | | |
| Tons per month | 2,091 | 2,273 | 2,474 | 3,010 |
| Tons per table per 24 hr | 4.70 | 5.68 | 6.16 | 7.28 |
| Assay per cent. copper | 2.56 | 2.63 | 2.63 | 2.78 |
| Density in grams per gallon | 385 | 372 | 386 | 499 |
| Density in per cent. solids | 9.5 | 9.2 | 9.5 | 12.0 |
| Plant concentrate: | | | | |
| Assay per cent. copper | 5.14 | 9.87[b] | 9.27[b] | 8.02[c] |
| Copper content, pounds | 56,160 | 64,180 | 47,420 | 114,680 |
| Tailing: | | | | |
| Assay per cent. copper | 1.65 | 1.38 | 1.33 | 1.15 |
| Percentage copper recovery: | | | | |
| Plant | 40.4 | 39.8 | 43.0 | 50.7 |
| Tables | 52.4 | 53.6 | 57.2 | 68.6 |

[a] April to September.

[b] Final concentrate after being finished on one or two Deisters.

[c] Final concentrate = round-table concentrate plus secondary table concentrate.

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*Suggested Flow Sheet for Slime Classification and Treatment*

While testing out the conoidal decks in the slime plant the writer took up the question of classifying the through 0.07-mm. slime preceding round-table treatment with a view to obtaining a further increase in slime-plant efficiency, having in mind the following flow sheet for the treatment of slime.

(1) The slime overflowing the hydraulic classifiers in the mill to be sent to V-settling tanks equipped with transverse or longitudinal baffles set at an angle of 45° to insure the overflowing of colloidal or semicolloidal material only, the fine crystalloid material to be discharged with a certain amount of the colloidal material through the spigot.

(2) The spigot discharges of the V-tanks, a comparatively small volume of slime, to be sent to cone-shaped hydraulic classifiers, designed to effect a separation between the colloidal and the crystalline material. The spigot discharge of the cone classifiers to be fed to fine-sand tables of the Wilfley or James type or to round conoidal deck tables having a comparatively flat slope. The overflow of the classifiers to join the overflow of the V-tanks.

(3) The combined overflow product of the V-tanks and cone classifiers to be dewatered in Callow tanks equipped with conical baffles set at an angle of 45° (about seven baffles per tank). Some of the very finest of the colloidal material to be sloughed off as waste in the overflow of the Callow tanks, probably about 10 per cent. of the total solids fed. The overflow of the Callow tanks to go to waste, or to be pumped for re-use in mill; the spigot discharges of the Callow tanks to be sent to free-settling classifiers, or long tanks equipped with longitudinal baffles set at an angle of 45°. The cross-sectional area of the tanks to be large enough to permit the slime to pass through the tanks at a very slow rate. The speed of the current at all points in the cross section to be kept uniform by the use of a feed sole and the longitudinal baffles spaced 3 in. apart. The overflow of the tanks, if any, to go to waste. The spigot discharge of the tanks to be combined in three products representing head end, center and tail end of tanks. The combinations of the spigot discharges of the tanks to be distributed as follows: Spigots from head end of tank to go to revolving round tables with conical or conoidal deck surfaces and with about the same overall slope as conoidal deck No. 2, previously mentioned (flat slope). Spigots from center of tank to go to similar tables with about the same overall slope as conoidal deck No. 3, previously mentioned (medium slope). Spigots from tail end of tank to go to similar tables with about the same overall slope as conoidal deck No. 1, previously mentioned (steep slope).

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All round tables to be operated to produce a finished concentrate, a middling for re-treatment, and a waste tailing product.

The re-treatment middling to be pumped back to head of Callow tank system, to go over again. No finishing tables other than the round tables to be employed.

In connection with the classification of through 0.07-mm. slime, the writer carried out a laboratory test on the classification of slime by free settling, and on the subsequent concentration of the settled products, the results of which will be presented in another paper. The wet mechanical concentration of the through 0.07-mm. slime resulting from the milling of the copper and iron sulphide ores of the Butte district of Montana is still in process of development, although rapid progress has been made in increasing the efficiency of the practice since Mr. Callow introduced the use of the round table as a concentrating machine for the slime in 1904.

## Conclusion

As stated earlier in this paper, the round table has been able to maintain its supremacy over all other types of machines on this particular class of work, and in the opinion of the writer it will continue to do so as long as wet mechanical concentration continues to be the most satisfactory method of extracting the valuable minerals from this class of material.

In making the above statement the writer does not mean to convey the impression that the revolving convex round-table practice described in this paper represents the last word in round-table practice on fine slime; it is possible that with further experimenting we may find that decks of a lesser diameter can be used, or, that a combination of the conical and the conoidal deck may prove effective. In the opinion of the writer, the work of the conical steel-frame cement deck would be greatly improved if a smoother finish were given to the deck surface than has so far been used. The comparatively rough cement deck surface used is an excellent surface for saving slime mineral grains but offsets this advantage to a large extent by retaining a large amount of fine gangue material, especially upon the lower radial half of the deck surface, which includes by far the greater part of the deck area. On this area, the transporting power of the water film having been lessened, a considerable amount of fine gangue material settles out which should be kept in a state of semi-suspension if a clean grade of concentrate is to be produced. On the upper radial half of the deck surface a rough finished surface is permissible because a heavy deposition of free-mineral grains occurs on this area which fills up the minute pits in the deck surface and tends to displace any of the lighter grains of gangue material which may settle out on this area.

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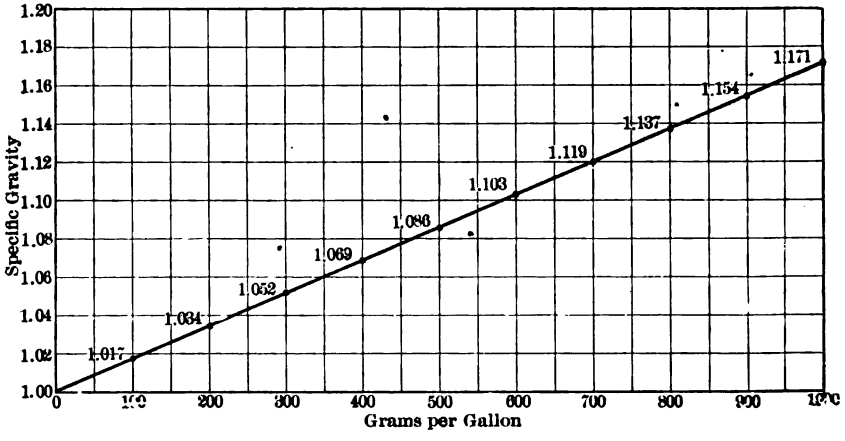
Therefore, a cement deck surface might be constructed with the upper radial half rough finished and the lower radial half very smoothly finished. It is probable, however, that the result obtained would closely approximate the results obtained from a conoidal deck having the chord of the arc laid on a slope of 1.125 in. per foot and having a rough finished cement surface.

In concluding this paper the writer wishes to acknowledge his indebtedness to the following gentlemen for data furnished, the source of which is not fully acknowledged in the preceding pages of this paper.

Henry Fisher and C. H. Benedict of the Calumet & Hecla Mining Co., John A. Church of New York, Gordon S. Duncan of the Mines Management Co. of New York and London, Eng., and H. Foster Bain of the *Mining and Scientific Press*. The writer also wishes to acknowledge the courtesy of C. W. Goodale, A. E. Wheeler, and M. W. Krejci of the management of the Anaconda Copper Mining Co., Boston & Montana Reduction Department, in allowing access to the company's files bearing on the subject and to thank Mr. Goodale for valuable suggestions resulting from his reading and criticism of the manuscript.

NOTE.—As the expressing of the density of a pulp in grams of solids per gallon of pulp, which is the practice at Great Falls, is probably unique (the usual practice being to express the density in per cent. of solids), three charts, Figs. 8, 9 and 10, are presented with this paper which may be used as follows: Fig. 8: To convert grams per gallon to specific gravity. Fig. 9: To convert grams per gallon to per cent. solids. Fig. 10: To convert grams per gallon to degrees Baumé.




$$\text{Formula: Sp. Gr.} = \frac{3{,}784 - \frac{x}{2.85} + x}{3{,}784}$$

$x$ = Grams per gallon.
3,784 = Weight 1 gal. water in grams.
2.85 = Sp. Gr. of dry slime.

FIG. 8.—SLIME DENSITY. GRAMS PER GALLON TO SPECIFIC GRAVITY

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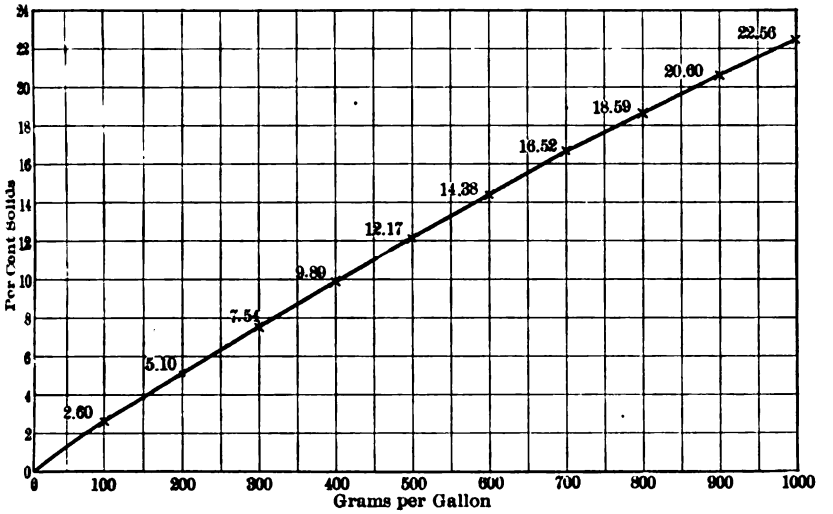
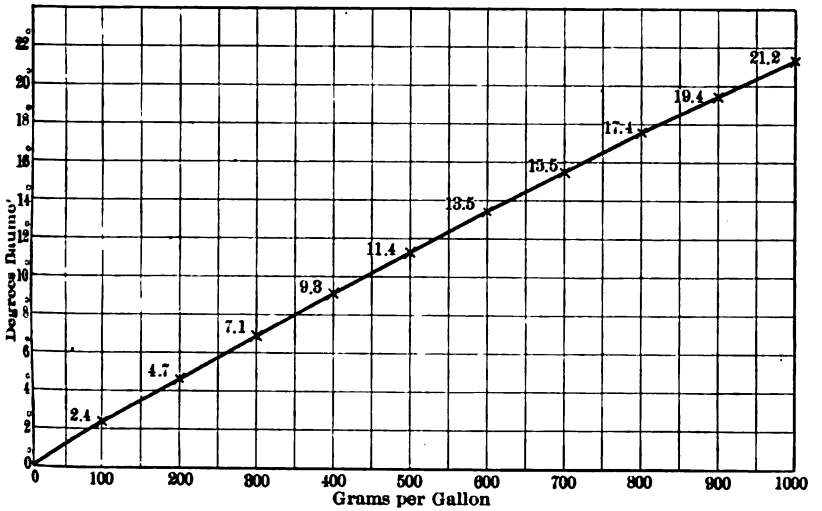



Formula: Per cent. solids. = $\dfrac{x}{3{,}784 - \dfrac{x}{2.85} + x}$

$x$ = Grams per gallon.
3,784 = Weight 1 gal. water in grams
2.85 = Sp. Gr. of dry slime

FIG. 9.—SLIME DENSITY. GRAMS PER GALLON TO PER CENT. SOLIDS.




Formula: $B° = 144 - \dfrac{144}{\text{Sp. Gr.}}$

FIG. 10.—SLIME DENSITY. GRAMS PER GALLON TO DEGREES BAUMÉ.



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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# An Amendment to Sales's Theory of Ore Deposition

BY FREDERICK W. BACORN, BUTTE, MONT.

(Salt Lake Meeting, August, 1914)

THE paper of Reno H. Sales on Ore Deposits at Butte, Mont.,[1] is a careful and painstaking work, an important contribution to the literature of the subject. As is almost inevitable in a work of such magnitude, a few misstatements occur; and some of these I shall endeavor to point out. Mr. Sales has also fallen into error, as I think, in the application of his theory of ore deposition; and while the modification which I venture to suggest is a minor one geologically, it is of great economic importance. I refer to Mr. Sales's idea of an outward migration of mineralizing solutions from a supposed central copper zone. In order that I may make myself plain, I must first epitomize the theory as a whole, which I understand to be this:

All of the Butte ore deposits (save those along the Continental fault, not herein considered) are derived from solutions of magmatic origin, which, at the beginning of their ascent from a great depth, were acid in reaction, hot, and were carriers of copper, zinc, manganese and other metals. As they departed from their point of origin they lost heat, and, because of reactions with the wall rock, became more and more alkaline. The metal deposited from these solutions in their hot condition was principally copper; and, as they cooled, they deposited less and less copper and more and more manganese and zinc, until the copper practically disappeared.

With this theory in general I have no fault to find, nor have I, it must be stated, the qualifications necessary to enable me to pass upon it critically; but it seems to me to be reasonable and probable, and is one which I am quite willing to accept.

But Mr. Sales, for what seem to me to be insufficient reasons, has assumed that there is one deep central source of copper, relatively rich and small, from which all the solutions emanated. From this central source the solutions rose, through one channel or through a group of channels, to the surface, at and within a small area called the central copper zone,

[1] *Bulletin* No. 80, August, 1913, pp. 1523 to 1626.

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shown in Fig. 7 of Mr. Sales's paper, p. 1575. From this central copper zone, via channels relatively near the surface, the solutions migrated concentrically.

In the veins within the first ring, called the intermediate zone, was deposited copper more or less intermixed with manganese, zinc, etc. Further out, in the peripheral zone, was deposited little copper with much manganese and zinc. Still further, beyond the peripheral zone, the depleted solutions made no noteworthy deposits.

I have spoken of the channels through which the solutions migrated outwardly as being relatively near the surface. Mr. Sales is not clear on this point; but I think his idea is that of a main trunk channel rising from the central source of copper, and tapped near the surface for horizontal distribution. If his idea were that of a radial distribution from the central source of copper, he would have said so; and, besides, a radial distribution, or any distribution from a point relatively near the central source, would not have produced the concentric arrangement which Mr. Sales conceives, within the small area considered in his paper. A circle 10,000 ft. in diameter takes in practically everything shown in the map on p. 1575, yet if the central source were 20,000 ft. deep, no point in the circle would be more than 615 ft. further away from the central source than any other point, and 615 ft. would be relatively insignificant. There is no radial arrangement of the veins themselves, either on strike or on dip.

I think that I have stated Mr. Sales's views correctly, though perhaps not with technical accuracy. His views on this point, as I so understand them, I believe to be erroneous.

In the first place, I do not think that there is any "concentric zonal arrangement" of the district. There may be a sort of concentric arrangement about the area which Mr. Sales calls the "central copper zone;" but this concentric arrangement, in a view of the district as a whole, is a local phenomenon, and it may be one of a plurality of such.

Within this central copper zone are the great copper mines of the district. The veins, as explained by Mr. Sales, carry little or no manganese or zinc. Ordinarily they do not outcrop, and their apices, when exposed by surface cuts, show only iron-stained, crushed, altered granite, with stringers of iron-stained quartz.

West and northwest of this central zone is the "silver district," so called because here were the first mines of Butte, wherein mining operations were for silver only. North of the central copper zone is an eastward projection, in the shape of a wedge, of the silver district. The veins within the silver district ordinarily outcrop strongly, the outcrops being principally quartz, stained black by manganese. In depth the veins carry manganese and zinc; and until recently they were supposed to carry little or no copper.

From the central copper zone into this silver district, there is a pro-

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gressive change from veins of the characteristic "copper" type to veins of the characteristic "silver" type; but eastward and southward from the central zone, this progression, if it exists at all, is by no means so obvious; and discoveries in these directions are coming so rapidly that it is safest to say that conditions there are unknown.

Furthermore, on going northward, across the wedge-shaped projection of the silver district, the order of progression is reversed; and one comes again into a region of "copper" veins; that is to say, of veins which ordinarily do not outcrop, which do not show the prominent black outcrops of typical "silver" veins, and which show little or no manganese or zinc; but which, on the other hand, show the typical crushed and altered granite and iron-stained quartz of the "copper" veins.

Here is a large section within which veins of the "copper" type predominate; and for specific illustration there may be pointed out at Mountain View Junction, only a little over a mile from the great Badger copper mine, many large, strong veins, which completely fulfill Mr. Sales's description of "copper" veins, and which, so far as developed, show no evidence whatsoever of inferiority.

The point of all this is, that while Mr. Sales is careful to state that the limits of pay ore are as yet unknown, his theory of the outward migration of the mineralizing solutions from the central copper zone implies to the outlying and undeveloped territory some degree of inferiority. His attention being not unnaturally focused on the ground which comes under his daily observation, he has unconsciously assumed that the present absence of operating copper mines in the outlying sections is proof of the absence of copper; and on this assumption he has predicated the idea of the concentric arrangement of the district.

As a matter of fact, Butte is peculiar among all the mining districts of the world in that development and growth have extended outward from the point where the first discovery of profitable copper ore was made; not, as elsewhere when bonanzas are discovered, by leaps and bounds, but slowly and haltingly, usually from vein to vein and from claim to claim. The reasons for this have been sociological, and have had nothing to do with geology.

But to continue the discussion, not only does it appear that any concentric arrangement, if it exist, is a local phenomenon, but there seems to be no other good reason to support the theory of outward migration.

The deep central source of copper is, of course, purely hypothetical. Nor is it necessary thus to account for the copper in the Butte mines, as Mr. Sales suggests (p. 1601). Hundreds of analyses of Butte granite, made in the course of mining litigation, have disclosed the presence of copper. One specimen taken from a point without the central copper zone and particularly studied by Mr. Weed[2] carried copper to the extent

[2] *Professional Paper No. 74, U. S. Geological Survey*, pp. 93, 94 (1912).

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of 0.006 per cent. A rough calculation shows that, at this rate, every 3 or 4 cubic miles of rock contains as much copper as has ever been mined in Butte; and it cannot be doubted that very many cubic miles of rock were drawn upon in the process of the mineralization of the district.

Much importance is attached by Mr. Sales to the porphyry dikes, although he does not clearly specify the part which he believes them to have taken. I think it will be found that these dikes are much more common phenomena than now believed to be, for on the surface they are ordinarily not conspicuous, and nobody has hunted for them; but even assuming that none exist save those described by Mr. Sales, their connection with the ore bodies is not obvious. They appear without as well as within the central zone; copper is found near them and thousands of feet distant; they are not permeable and were not channels for the solutions; they do not carry copper; and the veins are not enriched where they cut through the dikes. The only evidence of their relation to the ore deposits is in the fact that one of the series of ore-bearing fissures roughly parallels them. Standing alone, as it does, this fact would seem to mean only that the rock stresses which formed the porphyry-filled fissures persisted until after the Anaconda vein fissures were formed. Later, different rock stresses caused another series of fissures, not parallel with the dikes, but forming with them an angle of 45°, and these fissures, like the Anaconda fissures, are ore bearing, so that the evidentiary value of the parallelism between the dikes and the Anaconda fissures is greatly weakened.

From this review, it would seem that the evidence of the outward migration of the mineralizing solutions is very slight; and, on the other hand, there are certain facts which are incompatible with that theory.

The Jessie and the Edith May veins run outwardly from the central zone into the intermediate and peripheral zones; and hence, if there has been an outward migration, these veins must have been among those channels through which the flow took place. In such cases, we should expect to find, as we depart from the central zone, that the ore deposits would partake more and more of the characteristics of the silver veins, or, for short, that they would be more and more zincky. At any given point, the character of the deposit must be more zincky than at any point nearer the central zone, because the solutions, once cooled, would not regain heat, nor, once released from pressure, enter channels under greater pressure. What we do find is stated by Mr. Sales on p. 1581; that is, a zonal arrangement on a small scale. "High-grade chalcocite-enargite-bornite ores form the central part of the ore shoot with but little sphalerite and chalcopyrite and shade almost imperceptibly into zincky ore with chalcopyrite toward the ends of the shoots."

Now this is exactly what we should expect to find if the mineralizing solutions had come up through the open portions of the veins, there depos-

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iting the copper minerals, and in the thinner places cooling off and depositing zincky minerals.

Again, the Badger vein, at the surface and for the first 1,000 ft. in depth, is what would be called a silver vein. At about the 1,000-ft. level copper begins to come in; and from the 1,300-ft. level to the present bottom of the mine is one of the finest bodies of copper ore in the district.

No part of this vein is within about 4,000 ft. of the central zone; and it is a practical impossibility to figure out any connection between it and the central zone which would account for the facts on the theory of the outward migration. This vein, unlike the Jessie and the Edith May, does not run from the central zone outward, but strikes eastward and westward. It does not approach the central zone on its dip; but is approximately vertical, whereas the principal veins in the central zone dip to the south; that is, away from this vein.

Again, as in the case of the Jessie and Edith May veins, the facts observed in the Badger vein are compatible only with the hypothesis that it was mineralized by solutions rising through the vein itself.

Finally are the veins at Mountain Vein Junction above referred to. In these the granite filling has been altered, as in the copper veins elsewhere, and at one point there is a considerable showing of copper carbonates, within the walls of and unmistakably a part of one of the veins. In addition to this, nearby in the railroad cut is an exposure, for 500 ft., of altered granite similar to or identical with the altered granite of the central copper zone.

What produced this alteration and what deposited this copper? Meteoric waters do not make such alterations (Sales, p. 1553). This place is 2 miles distant (northeasterly) from the central copper zone; and between are great areas of unaltered granite, zinc mines, silver veins, and all the phenomena of the "peripheral zone." There is no north-and-south fissure which could have carried the mineralizing solutions from the central zone. There is no flat, blanket-like fissure, known or suspected, which might have acted as a channel; and if, for the sake of this discussion, we assume the existence of such a blanket fissure, then we are confronted with the difficulty that where tapped by the Black Rock vein, a mile from the central zone, the solutions rising through the Black Rock vein made deposits characteristic of "silver" veins; and where tapped in the neighborhood of Mountain View Junction, 2 miles from the central copper zone, the solutions have effected results characteristic of the "copper" veins.

We are, therefore, driven to the conclusion in this case, as in the cases of the Badger, the Jessie, and the Edith May veins, that the solutions came up through the veins themselves, from whatever may have been their source.

There are a great many veins in the outlying territory which show no

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variation (material to this discussion) from the veins mentioned: and as the process of reasoning which has been applied to a few veins is applicable to each of the others, it must be that all veins which were strong enough and open enough, acted as channels through which the mineralizing solutions found their way from great depths to the surface.

There is nothing unreasonable about this view. The Boulder batholith, of which the Butte district is a part, was a mass of lately molten granite (so Mr. Sales thinks) thousands of square miles in area and of great depth. There was a solid crust overlying a solidifying mass. Within the comparatively small area of the Butte district the crust was much broken and fissured. Steam and water were being forced to the surface. All openings must have acted as channels; and the fact that many fissures now show the effects of the hot water ("solutions") which flowed through them, is exactly what we might expect.

As will have been inferred from the foregoing discussion, the amendment to Mr. Sales's theory or hypothesis which I venture to propose, is this: that instead of there having been an outward migration of the mineralizing solutions, during which outward migration the cooling and the chemical changes took place, the flow of the solutions was generally upward, and the cooling and the chemical changes took place during the upward flow.

I see no difficulties in the way of accepting this modification. I do not see why the entire series of changes in the chemical character of the solutions, as well as the loss of heat, described by Mr. Sales, may not have taken place in each vein. I believe it possible to explain on this hypothesis all the phenomena described by Mr. Sales. This I can myself attempt only in a very general way; and it is to be hoped that Mr. Sales and other geologists will give the matter further attention.

I think the phenomenon of the central copper zone may be explained in the following manner:

Here are many veins, including a number belonging to the Anaconda system. The Anaconda fissures were the earliest, and the solutions rising through them, because of the long period during which they acted, or because of their greater concentration, or perhaps of their greater pressure and heat, or perhaps because of all these conditions, altered the granite to a greater extent than has yet been observed elsewhere in the district.

There is little zinc or manganese in the central zone. If the deposition of these metals depended on the lowering of the temperature of the solutions to a certain critical point, then we have only to assume that the temperature of the solutions in this region remained above that critical point until they reached the surface, or until they rose higher than the present surface. The larger the volume of solution, or the faster its flow, the further up would its temperature be maintained.

In the case of the Emily vein, in the Badger mine, we have another

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condition, and we may say that most of the copper contained in the solutions was deposited by the time they had reached the 1,300-ft. level, in the form of chalcocite and enargite. A little copper was left, and this was deposited as a fringe, perhaps 300 ft. deep, above the enargite and chalcocite, in the form of bornite and chalcopyrite. A similar fringe, in a number of other cases, is reported by Mr. Sales. Above the copper, the temperature of the solutions was lowered to a point permitting the deposition of the minerals characteristic of a "silver" vein. Now if the critical point for the deposition of the "silver vein" minerals had been a few hundred feet deeper than it actually was, or if the Emily vein had never been explored to a depth greater than 1,000 ft., then the Emily would now be regarded as a typical "silver" vein. Quite likely all "silver" veins are copper veins in depth, though it may often be the case that the point at which copper comes in is too deep for commercial exploitation.

In some veins are large copper ore shoots which do not extend to the surface, above which copper there are no notable deposits of "silver vein" minerals. Perhaps these cases can be explained on the theory that the copper in the solutions was exhausted at the tops of the copper ore bodies; and that the temperature of the solutions, at these points, had not fallen to the critical point for the deposition of the "silver vein" minerals. From the top of the copper, up to the points where the temperature so fell, which points might be above the present surface, the veins would be practically barren, showing only an alteration of the granite.

These are matters which must be worked out in detail by qualified men; and it is greatly to be hoped that not only will Mr. Sales himself be able to give further attention to the work which he has so well begun, but that others will enter the field, where much remains to be done.

People coming to Butte from the outside may well be surprised to discover how much does remain. In view of the many years during which Butte has been known as a "bonanza camp," in view of the enormous production of copper, which continues without diminution, it is indeed strange that almost all knowledge of the district is confined to the small area, perhaps 2,000 acres, which Mr. Sales has described in detail.

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# A Comparison of the Huntington-Heberlein and Dwight-Lloyd Processes

BY W. W. NORTON, MURRAY, UTAH

(Salt Lake Meeting, August, 1914)

THE gradually increasing proportion of sulphide ores which lead smelters of to-day are called upon to handle has caused the roasting problem to become one of ever greater importance.

We may look back a score of years or so and recall the old hand roasters, some of which turned out a finely divided roasted product, and others fashioned with a "fuse-box" wherein the roasted ore was slagged. Then came mechanical furnaces of several types, the Brückner cylinder, the Brown-O'Harra, the Ropp, and others, all designed to cut out the cumbersome hand labor of the old reverberatories. Roasting costs were thereupon reduced and tonnages stimulated to a gratifying extent. However, it seems to have been early recognized that this very feature of a greater amount of roasted sulphide ore as compared with oxide ore brought with it a train of difficulties at the blast furnaces. The mechanically roasted ore was fine physically, the blast furnaces were choked and the troubles of the smelterman were not at an end. Then the Huntington-Heberlein pot system of roasting appeared upon the scene, and later on the Dwight-Lloyd machines were invented, and inasmuch as both of these processes were designed not only to roast the ore but to agglomerate it as well, it was apparent that distinct steps in advance has been achieved.

At the Murray plant, modern up-to-date roasting practice is fully exemplified and there are now in successful operation roasting furnaces or devices of several sorts: namely, Godfrey revolving-hearth furnaces, Wedge multiple-hearth mechanical roasters, Dwight-Lloyd sintering machines, and Huntington-Heberlein pots. Godfrey and Wedge furnaces will properly handle material high in sulphur, say ores with 25, 30, and 35 per cent. of that element; D. & L. machines and H. & H. pots will positively not treat efficiently ores or mixtures containing anywhere near the sulphur content mentioned, but are confined to charges containing in the neighborhood of 15 or 18 per cent. In passing, it may also be explained that, so far as the knowledge of the writer goes, Godfrey and Wedge furnaces do not economically eliminate sulphur to an extent

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sufficiently low for lead-smelter practice. With these simple facts in mind, it will be perfectly clear to all that the metallurgist in charge may elect to treat sulphide ores in either one of two ways: he may pre-roast in Godfrey and Wedge furnaces and subject the partly roasted product to a final treatment on D. & L. machines and H. & H. pots, or he may dilute the average sulphur in the raw ore to 15 or 18 per cent. by means of an admixture of the requisite quantity of non-sulphur fines and send the mixture thus obtained to D. & L. and H. & H. machines. The Murray plant does both. A certain flexibility is thus afforded for a segregation of the various classes of sulphide ores; moreover, in the matter of oxide fines, one can limit screening operations to a point deemed best metallurgically.

Godfrey and Wedge furnaces are essentially pre-roasters, D. & L. machines and H. & H. pots are final roasters. At Murray all final roast is either D. & L. or H. & H. We turn now to the primary object of this paper, namely, a brief discussion of the comparative merits of these two widely distinct methods of final roasting.

### *Cost of Installation*

The Murray plant is equipped with two D. & L. machines, the total daily capacity of which may be stated at 220 tons, and 23 H. & H. pots, with capacity of 400 tons. It would, of course, be manifestly unfair to directly compare the total costs of these two installations, but it seems quite safe to say that for almost any given tonnage capacity a D. & L. plant can be built for considerably less than an H. £ H. plant, it being understood that by H. & H. is meant the converting-pot portion of an installation only, with no reference to Godfrey furnaces. In the case of the H. & H. one must have heavy cast-iron pots for handling ore in comparatively large units, expensive overhead traveling crane, substantial cooling floor, and, finally, a crusher which the D. & L. does not require at all. The cost of the installation item must be put down in favor of the D. & L.

### *Cost of Roasting*

Any discussion of roasting costs should of course be based on units of sulphur eliminated. In a general way, our experience has shown that the D. & L. will reduce an initial sulphur of about 15 or 16 per cent. to about 4 per cent. in the roasted product, while the H. & H. is capable of handling a slightly higher initial sulphur, say 17 to 18 per cent., with resultant 5 per cent. in product. During a very recent period of 47 consecutive days, it is known that units of sulphur eliminated per ton of charge by the D. & L. practically equaled units of sulphur eliminated per ton of H. & H., and it is probable that an exhaustive examination of Murray-plant roasting

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records would show about the same amount of sulphur per ton of charge driven off as between the two sorts of roasters now under review. It follows that figures representing costs of roasting are truly comparable.

The limitations of this paper will not permit of a detailed review of roasting costs, but it may be stated that during the entire year 1913 the H. & H. made the better showing to the extent of about 5c. per ton roasted, and for the first three months of 1914 the H. & H. also had an advantage of about 3c. per ton. Murray experience, everything considered, indicates slightly lower costs for H. & H. as compared with D. & L., but the fact that all calculations are based on operations at an H. & H. plant having twice the capacity of a D. & L. plant must not be overlooked.

### *Adaptability to Wide Range of Charge Ingredients*

Any intelligent discussion of analysis of raw charge to roasters should have the fundamental thought in mind that the metallurgist must treat what comes to the plant. He cannot always be favored with the proportions of silica, iron, and lead which would give the best results, consequently the adaptability of any given roasting device to a variety of materials will be accepted as an item of far-reaching importance.

Some two or three years ago, in connection with a visit to three or four custom lead smelters newly equipped with D. & L. machines, the writer was somewhat impressed with the limitations placed on the charge the machines were capable of handling. Inquiry brought forth the information that certain sorts of materials could be attempted only by resort to a special layer of fine limestone or other infusible material carried next to the grates; any percentage of raw matte at all seemed out of the question; zinc was naturally side-stepped as highly deleterious; much stress was placed upon the proportion of silica to the iron, and nearly all the enthusiasts demanded a goodly percentage of lead provided a choice quality of sinter was to be in evidence. Of late, however, the staff at Murray have found that a wide range of mixtures may be efficiently handled over the D. & L., and have no doubt that equally good progress has been accomplished at other works. Pre-roasted ore, any kind of raw sulphide ore or concentrates, flue dust, pre-roasted matte, or even raw matte may be combined in certain proportions and successfully sintered over these machines. A sufficient quantity of non-sulphur diluent to bring the average of the mixture down to 16 per cent. sulphur must always be added and of course the details of operation must be cared for.

However, equally pleasing results have been attained with H. & H. pots. With reference to chemical make-up of charge, the H. & H. will also handle "any old thing."

Turning now to physical character of the raw ore, it is of course recognized that the air currents are required to permeate a thin layer of

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charge in case of D. & L. treatment, whereas the pot roasters are committed to a very much thicker layer; however, a physically fine charge will restrict tonnage on D. & L. just as surely as it will on H. & H. machines, although the D. & L. process is able to successfully treat slimes or other fine material which it would be wholly useless to attempt to treat on the H. & H.

By way of summing up, it may be stated that the D. & L. process possesses a slight advantage over the H. & H. in the matter of flexibility or range of charge because the D. & L. permits more delicate application of operating details which are essential to success, and also because extremely fine materials find no proper place in the H. & H. charge.

### *Lead Losses*

We have certain data at hand showing a very moderate lead loss on D. & L. machines, these data being based on standard operating conditions during which the resultant gases and fumes were sampled and analyzed. No data are available as covering losses with H. & H. pots. The expense and difficulties in connection with accurately sampling an H. & H. output of 400 tons per day need not be pointed out and gas measurements and samples taken from the combined gases of 23 pots on two different main flues might eventuate in metal-recovery data not wholly dependable.

At the Murray plant there have been periods of time when either the D. & L. or the H. & H. machines have been wholly or partly out of commission, and the metal losses of the whole smelter during these periods have been observed. It is regarded as doubtful if the D. & L. process is productive of any lower metal losses than is the H. & H.

### *Physical Condition of Product*

Final roasting treatment results in a sintered or agglomerated product, and incidentally material of a desirable physical character is passed along to the blast furnaces. The D. & L. sinter is usually of a porous or cellular structure; the H. & H. tends to greater density or firmness. Published and unpublished opinions of metallurgists have sought to show that the peculiarly open or coke-like structure of the D. & L. sinter carried with it certain extraordinarily favorable properties when subjected to the smelting process in the blast furnace, and have even claimed appreciable saving in the coke percentage used for smelting. Rather exaggerated ideas concerning the efficiency of an exposure of porous surfaces to contact with reducing gases have been advanced and intimate mixtures (possibly intimately combined silica and lead) have been proclaimed as "predigested" and therefore more easily reduced. The writer believes

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that a partly fused or "predigested" combination may tend to poor results rather than to good results when smelted, for the reason that such substances fuse at too low a temperature in the furnace. Certain writers have gone so far as to examine the cell structures of the D. & L. product microscopically and have declared that glazed or unglazed surfaces have a bearing upon the readiness with which the products were later reduced in furnaces.

With all due respect to the theories above set forth, it was considered that more dependable conclusions could be drawn by means of actual operating tests and accordingly the Murray plant furnaces during five days of August, 1912, were run on two charges, the one containing no D. & L. roast at all, the other containing a rather large amount of this material. It was believed that any peculiar virtue existing in D. & L. product would have abundant opportunity to make itself manifest. The exact charges used are given below, together with the average lead in resultant slag and matte.

| | Furnaces 1, 3, 7 and 8 (No D. & L.) | Furnace 5 (D. & L.) |
|---|---|---|
| Coke, 920 (11½ per cent.) | ..... | ..... |
| Bed 36, bin 7 | 2,970 | 320 |
| H. & H. roast | 2,000 | ..... |
| D. & L. roast | ..... | 4,800 |
| Hand-roasted matte | 600 | 400 |
| Iron ore | 690 | 540 |
| Limestone | 1,640 | 1,840 |
| Scrap iron | 100 | 100 |
| | 8,000 | 8,000 |
| | Per Cent. | Per Cent. |
| Average lead in slag for the run | 0.63 | 0.91 |
| Average lead in matte for the run | 10.7 | 14.96 |

Great pains were used to make the experimental run one of value. The D. & L. roasted product was of a typically honeycombed character. No. 5 furnace was in excellent condition, its operations were closely watched by the metallurgist in charge of the furnaces and by the writer, yet absolutely no strengthening of reduction appeared. On the contrary, No. 5 did worse than the other furnaces.

General blast-furnace experience covering a wide range of charges and a considerable period of time indicates that no particular effect, either good or bad, can be claimed for D. & L. sinter as relating to strength of reduction during the smelting process, and exactly the same remark will apply to H. & H. agglomerated material. (Of course the D. & L. sintered cakes must be broken to the proper size and the H. & H. material

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must be crushed suitably small, or distinctly bad reduction will ensue.) That both of these products of modern roasting development help the speed of furnaces enormously is certainly a fact. The final roasters of modern smelters, in supplanting the old hand roasters and fine-ore producing mechanical furnaces, have very naturally served to increase blast-furnace tonnages to a remarkable extent.

As to which product is the better physically, that is to say, which will produce the heavier tonnage at blast furnaces, a first-class D. & L. sinter does not excel a first-class H. & H. agglomerated product. Moreover, given a proposition of inferior quality of both, it would seem that the admittedly cellular or at times fragile D. & L. can hardly equal the more firm and stable H. & H. Here again, however, real experience at blast furnaces may outweigh mere conjecture or theorizing, so the following data are submitted with the idea of showing that in this instance at least the physical character of the D. & L. produced no better tonnage at blast furnaces than did the physical character of the H. & H.

On Aug. 12 and 13, 1912, the following two charges were smelted side by side with the same coke percentage, the same blast pressure, and as near like conditions in other respects as it was possible to obtain.

| | Furnaces 1, 3 and 5 (H. & H.) | Furnaces 7 and 8 (D. & L.) |
|---|---|---|
| Coke, 920 (11½ per cent.) | ..... | ..... |
| Bed 37, bin 4 | 1,400 | 2,060 |
| H. & H. roast | 3,000 | ..... |
| D. & L. roast | ..... | 3,000 |
| Hand-roasted matte | 400 | 400 |
| Iron ore | 1,140 | 580 |
| Limestone | 1,960 | 1,860 |
| Scrap iron | 100 | 100 |
| | 8,000 | 8,000 |
| Average tons per furnace per day | 294 | 287 |
| | Per Cent. | Per Cent. |
| Average lead in slag | 0.81 | 1.03 |
| Average lead in matte | 13.47 | 13.0 |

*Recapitulation*

The D. & L. and the H. & H. plants are operated conjointly in some respects; indeed, both are supplied with ore from the same cylindrical mixing bins and from the same belt conveying system. One metallurgist and one and the same set of foremen guide the destinies of the two types, abundant opportunity is offered for comparison, and absolutely no circumstances exist which would swerve opinion either one way or the other. It

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is believed that a fair summary of the actual experience set forth in this paper would be as follows:

| | Advantage in Favor of |
|---|---|
| Cost of installation | D. & L. |
| Cost of roasting | H. & H. |
| Adaptability of charge | D. & L. |
| Metal losses | Doubtful |
| Physical condition of product | H. & H. |

*Conclusions*

The D. & L. is a good modern machine now brought to a state of efficiency where it is splendidly adapted to lead-smelter work. The H. & H. process is still operated with a good measure of success. A good-sized, compact H. & H. plant at Murray, well designed and operated in a painstaking manner, permits of possibly an unusually good expression being recorded in favor of the H. & H.

The low first cost of D. & L. units and the ease and flexibility with which successive units may be added to any given lead smelter are perhaps the reasons why a greater number of 400-ton H. & H. plants have not received thorough trial.

Actual experience with D. & L. machines at Murray has caused them to gradually grow in favor from year to year. We are highly pleased with the flexibility of charge and other important items. The beautifully sintered product is so exceedingly good as to make it unfortunate that exaggerated claims for its virtues should have been published.

This article is unable to point out any overwhelming advantage of the D. & L. over the H. & H. system, although continued progress may upset the balance at any time. If history repeats itself some new roasting system will take rank over both within a few years.



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# The Bag House in Lead Smelting

BY H. H. ALEXANDER, MAURER, N. J.

(Salt Lake Meeting, August, 1914)

IN the early part of the last century textile fabric was used for the filtration of products of combustion and lampblack was obtained by passing smoke through a series of canvas bags. Natural draft was used to draw and force the smoke through the bags. About 1850 bag filtration was used for collecting zinc oxide. Around 1876 it was introduced for collecting the fume from lead-ore hearth smelting in Missouri. Shortly afterward it was used, first at Portland, Me., and later at Cañon City, Colo., for the collection and production of zinc-lead pigment in treating mixed zinc-lead sulphides. In 1890 the Globe Smelting & Refining Co., at Denver, Colo., installed a bag house containing 1,458 bags for the recovery of fume from silver-lead blast furnaces. This installation was too small to handle all of the gases and was increased to approximately 2,300 bags. In 1900 the number of bags was again increased, making a total of approximately 2,800.

The usual difficulties incident to new installations were encountered, but they were gradually overcome and the bag house became accepted as standard practice for this class of smelting. As all of the bag houses built since have followed the general lines of the Globe installation, a short description may be of interest.

The general construction of the Globe bag house is shown in Figs. 1, 2, and 3.

The building is 149 ft. 8 in. long by 67 ft. 4 in. wide and 40 ft. 2 in. high from the basement floor to the eaves of the roof, having brick walls 26 in. thick from the foundation up to the thimble floor, from there an 18-in. wall for 16 ft. and a 13-in. wall from this point to the top, a distance of 14 ft. 2 in. Additional stiffening is given to the walls above the basement by pilasters located every 16 ft. The end-wall construction is similar to that of the side walls with the exception of the spacing of the pilasters, which is somewhat greater. The roof is made of corrugated iron laid on 1-in. boards and supported by a timber frame work spaced 16 ft. apart resting on the basement partition walls. On top is a louver running the

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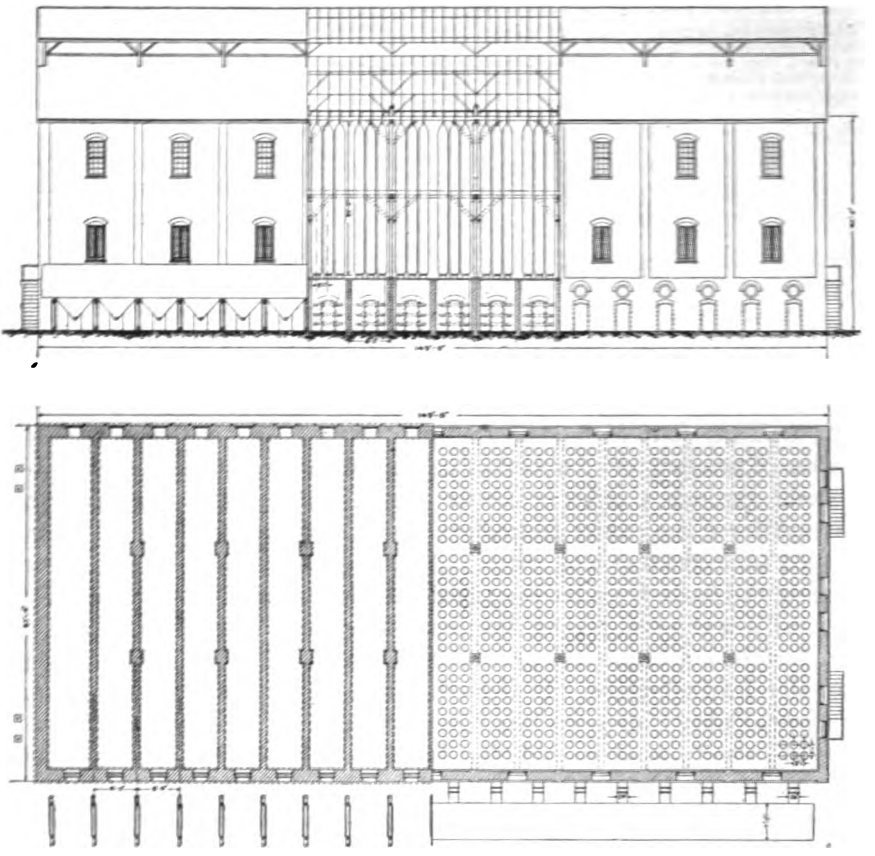

FIG. 1.—ELEVATION AND PLAN OF BAG HOUSE OF GLOBE SMELTING & REFINING CO., DENVER, COLO.

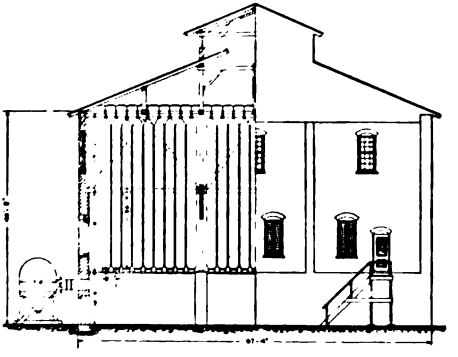

FIG. 2.—SECTIONAL END ELEVATION OF BAG HOUSE.

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length of the building to allow the gas to escape. The basement is 10 ft. high and divided into compartments by brick walls 13 in. thick spaced 8 ft. apart, made tight to prevent the gases from interfering with the cleaning out of the fume in any one compartment while the others are in operation. These walls also carry the thimble floor above, which is made of No. 10 gauge sheet-iron plates riveted together as nearly gas tight as possible, so that all gas entering the basement will be forced through the thimble floor into the bags above. On the thimble floor are fastened the thimbles, which are 17 in. in diameter and 10 in. high, made of No. 14 gauge sheet iron, with a head at the top to attach the bag and a flange at the bottom for riveting to the floor plate. They are spaced on 2 ft. 1 in. centers and arranged as shown on the floor plan. To these thimbles one end of the bag, which is 31 ft. long, is fastened by means of a soft iron or

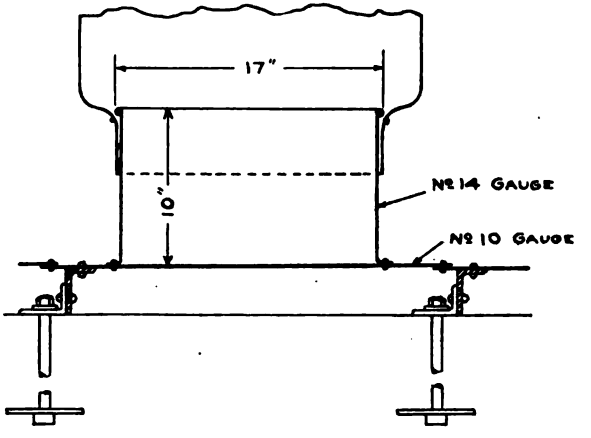


FIG. 3.—DETAIL OF THIMBLE CONSTRUCTION.

copper wire; the other end is hung by means of a similar wire fastened to the bag and then given several turns around a 2 by 12 in. plank overhead which runs parallel with and directly over each row of thimbles. These planks are carried on cross timbers resting on the frame work which supports the roof. The bags usually begin to deteriorate around the thimbles. As this takes place the lower end of the bag is cut off and the hanging wire at the top, which has been left sufficiently long for this purpose, is lengthened out; in this way the bag may still be used, although slightly decreased in filtering area. The gas is drawn from the furnaces through the flue by means of a fan and discharged into a flue passing along the side of the bag house. Short connecting flues 24 in. in diameter, equipped with gas-tight dampers, lead the gas from the main flue into each separate compartment in the basement. The thimble floor being gas tight, the fumes are forced up into the bags, the gas filtering through the

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fabric and passing out through the louver on top of the roof. The fume retained in the bags is dislodged, at regular intervals, by shaking, and deposited in the basement, and is removed through the basement doors after first shutting off the gas from that compartment.

The ideal filtering material is a thin layer of absorbent cotton, but, owing to the difficulty of cleaning and recovering the fume without destroying the material for filtering purposes, it is impractical. The bags used are either cotton or woolen, and while various other substances have been experimented with, they have not been successful, either on account of excessive cost, or, lacking in nap, the material would act as a screen and fail to abstract the solid particles from the gases. Some 20 years ago the writer treated woolen cloth with titanium chloride. This treated material filtered as well as untreated cloth and resisted the corrosive effect of sulphuric acid to a point where the condensed acid fully saturated the bag. The material of the saturated bag then either clogged up and no amount of shaking would dislodge the fume and all filtering ceased, or, if the gases were low in fume and high in acid, the nap of the cloth would fold up on the strands and the fumes would pass through without filtering.

Cotton bags do not offer as great a resistance to the corrosive action of acid and will not stand as high temperatures as woolen. It is difficult to determine from the analysis of the gas which material should be used, as the acid contents and temperatures of the gases, particularly from an oxidizing furnace, are liable to vary greatly. It is better to use woolen bags whenever there is doubt. As woolen bags cost from three to four times as much as cotton, they must last correspondingly longer.

The writer recalls seeing all the bags in a building destroyed within 30 min. This happened with gases from a converter working on lead-bearing copper mattes and occurred on the finishing blow to blister, when the lead content of the converter charge was low—under 2 per cent. The converter delivered the gases into a brick flue, the opening of which made a snug joint with the converter snout. The temperature in the brick flue back of the converter ranged from 800° to 1,300° F., and at the discharge end of this brick flue the temperature ranged between 400° and 700° F. In this range, giving the proper temperature—from 800° to 1,200° F.—for conversion, by contact, of sulphur dioxide to sulphur trioxide, the amount of sulphur trioxide formed was in excess of the lead oxide carried by the gases; the surplus acid rapidly destroyed the bags. This was overcome by enlarging the opening in the flue, also an opening with a damper was made in the side of the flue close to the point where the converter discharged into it; the fan was speeded up, thus diluting and cooling the gases. A recording thermometer was placed in the flue 25 ft. from the converter, and 600° F. was the maximum reading permitted on this thermometer. The dilution of the gas was also regulated by analyses,

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never allowing the sulphur dioxide content of the gases to get over 4 per cent. It had formerly run as high as 9 per cent. on the finishing blow. After these precautions were taken a set of cotton bags lasted over a year.

The same action may occur on concentrating and cupelling furnaces where the only sulphur present comes from the fuel. With the temperature of the flue as noted above, the conversion of sulphur dioxide into sulphur trioxide proceeds rapidly and destroys the bags. Fortunately, in a majority of these cases there is sufficient lead oxide in the gases to combine with the sulphur trioxide formed, rendering it harmless.

Another cause for the corrosion of bags is the presence of selenium. The selenium being volatilized and passing off with the gases as an oxide, upon coming into contact with the sulphur dioxide converts the latter into sulphur trioxide with the production of selenium, and unless there are sufficient bases to combine with the sulphur trioxide produced the bags are attacked.

The difficulties of an accurate determination for small amounts of sulphur trioxide in the presence of sulphur dioxide are well known, and while a large number of determinations were made, they were used for comparative purposes only.

Exhaustive tests were made to determine the critical temperature of cotton and woolen bags. The fabrics used in these tests were cut from the cotton and woolen bags in use in the bag house. The test pieces were cut to 3 by 4 in. and were pulled, always against the warp, in an Olsen testing machine. In making a test for any given temperature and period of time, a piece of cloth 8 by 15 in. was cut into 10 rectangular pieces, as shown below.

| | | | | |
|---|---|---|---|---|
| 1 | 2 | 3 | 4 | 5 |
| 10 | 9 | 8 | 7 | 6 |

The pieces bearing the odd numbers were laid aside and those bearing the even numbers were placed in a Freas electric oven, which was kept at the proper temperature for the desired period of time, at the expiration of which they were removed and all 10 pieces broken in the Olsen machine. In most cases, especially at or near the critical points, two or more, and sometimes as many as five, independent tests were made. All told, something more than 1,000 test pieces were pulled. Individual tests varied, but sufficient work was done to make the results given below, which are averages, correct.

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*Percentage Loss in Tensile Strength Due to Heating*

Cotton Cloth

| Temperature of Oven, Deg. F. | Time Pieces were Kept in Oven | | | | |
|---|---|---|---|---|---|
| | 1 hr. | 24 hr. | 48 hr. | 96 hr. | 144 hr. |
| | Per cent. | Per cent. | Per cent. | Per cent. | Per cent. |
| 210 | ..... | 0.0 | ..... | ..... | 0.0 |
| 215 | ..... | 0.0 | 0.9 | 6.1 | 9.4 |
| 220 | ..... | 4.2 | ..... | ..... | ..... |
| 225 | ..... | 5.25 | ..... | ..... | ..... |
| 230 | ..... | 7.3 | ..... | ..... | ..... |
| 235 | ..... | 12.1 | ..... | ..... | ..... |
| 240 | ..... | 17.1 | ..... | ..... | ..... |
| 245 | ..... | 20.7 | ..... | ..... | ..... |
| 260 | 0.3 | ..... | ..... | ..... | ..... |
| 265 | 3.5 | ..... | ..... | ..... | ..... |
| 270 | 5.3 | ..... | ..... | ..... | ..... |
| 280 | 9.6 | ..... | ..... | ..... | ..... |

Woolen Cloth

| Temperature of Oven, Deg. F. | Time Pieces were Kept in Oven | | | | |
|---|---|---|---|---|---|
| | 1 hr. | 24 hr. | 48 hr. | 96 hr. | 144 hr. |
| | Per cent. | Per cent. | Per cent. | Per cent. | Per cent. |
| 270 | ..... | 0.0 | 0.0 | 0.0 | 0.0 |
| 275 | ..... | ..... | ..... | 0.3 | 3.2 |
| 280 | ..... | ..... | 0.0 | 5.3 | 6.9 |
| 285 | ..... | ..... | 0.0 | ..... | ..... |
| 290 | ..... | 0.0 | 2.8 | ..... | ..... |
| 295 | ..... | 4.0 | ..... | ..... | ..... |
| 300 | ..... | 10.7 | ..... | ..... | ..... |
| 355 | 0.0 | ..... | ..... | ..... | ..... |
| 365 | 0.0 | ..... | ..... | ..... | ..... |
| 370 | 2.3 | ..... | ..... | ..... | ..... |
| 375 | 4.4 | ..... | ..... | ..... | ..... |
| 385 | 6.7 | ..... | ..... | ..... | ..... |

Each plant has its favorite brand of cloth, which is required to have a definite number of strands; in cotton cloth these vary from 30 by 30 to 48 by 48, depending upon the character of the fume to be filtered; with woolen cloth, owing to the longer nap, a coarsr weave may be used, and the number of strands is usually in the twenties. The woolen cloth should contain the natural grease, but, as the manufacturers object to this, the

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wool is usually scoured and manufactured into cloth and the grease added. After numerous working tests, checked by the laboratory, the following specifications were formulated for a satisfactory woolen cloth:

**The weight is to average 12 oz. per yard and the tensile strength is to be not less than 21.5 lb. per lineal inch. The test pieces are to be square, 3½ in. on a side, and the pull is to be against the warp. The fabric is to contain not less than 85 per cent. wool fiber, estimated by taking the difference between 100 per cent. and the sum of grease, dirt, moisture, burrs, and cotton fiber, and to be practically free from vegetable matter of all kinds. The weave is to be 22 ends by 20 picks per inch.**

Yarn is used for sewing woolen material. Linen thread should be used for cotton bags, usually No. 40 Barbour's Irish Linen, with a double lap seam, and lock stitch is specified.

There is a diversity of opinion as to the necessity of ventilation around the bags, but the consensus of opinion is that good ventilation lengthens the life of a bag. Some plants have gone to the extent of drawing the gases from around the bags with a fan and discharging them into a stack. An iron stack 4 ft. 6 in. in diameter and 68 ft. above the roof has proved satisfactory. This stack takes care of 15,000 cu. ft. of gas per minute and maintains a draft of about 0.05 in. of water. The temperature of the gases entering this stack varies from 110° to 130° F. Diffusion stacks after the Wislicenus type were tried for converter gases. They worked nicely, but the rain beating through the openings caused rapid deterioration and they were abandoned.

The number of bags in a compartment which can be closed off from the main current varies greatly. The original Globe bag house contained 81 bags. Some smelting works have as many as a thousand bags in a compartment. As most bag houses are run continuously, one or two compartments are usually cut out for cleaning, shaking, etc., and it is therefore better that a unit be of such size as not to have too large a percentage of the filtering area cut out when one or more compartments are closed off.

The filtering area or bag surface necessary to handle a given amount of gas is entirely dependent upon conditions, and what is ample in one case may be insufficient in another. The area necessary is not only dependent upon the volume of gases, but upon the amount of solids per cubic foot, the stronger or weaker adhesion of the fume to the filtering material, and the number of times the bags are shaken in 24 hr.

Few smelters accurately measure the volume of gases being handled, but use the manufacturer's rating of the fan at the different speeds. As this depends upon character, resistance, temperature, etc., of the flue to the fan, and the same variable factors on the discharge end of the fan, the rated amount is not always delivered. Comparing performances of bag houses with fan rating, one bag house could be cited in which each bag is filtering 130 cu. ft. of gas per minute and recovering 5 lb. of fume

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per bag per day, with bags shaken once in 24 hr., and another in which each bag filters 70 cu. ft. of gas per minute and recovers 27 lb. of fume per day, with bags shaken eight times during that period.

In dislodging the adhering fume from the bags, hand shaking, when done properly, gives the best results, but it is a slow and disagreeable task and is being replaced by various mechanical shakers. This mechanical shaking is accomplished by striking the inflated bag lengthwise; quickly jerking the deflated bag up and down; swinging back and forth; or a combination of the two motions. Another method is to reverse the flow of the gas through the bags by means of an individual fan or by a second connection between each compartment and the suction side of the main fan. The best results were obtained with the arrangement shown in Fig. 4. In this arrangement we have the combination of the

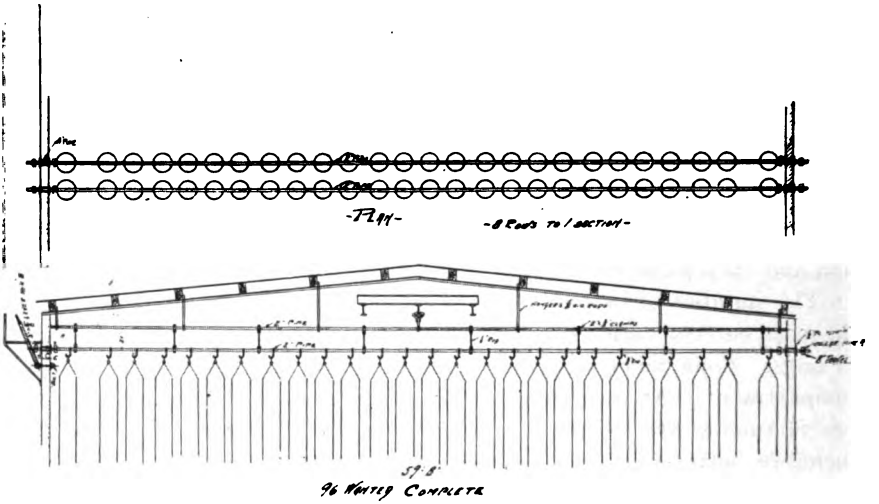

FIG. 4.—BAG-SHAKING DEVICE.

up-and-down and the back-and-forth motions. The bags in each row are hung from a 2-in. pipe, which in turn is supported from above by hangers spaced about 8 ft. apart and 21 in. long. The ends of the pipe pass through cast-iron spools placed in the walls, with holes in them sufficiently large to allow for the up-and-down motion caused by the swinging around the 21-in. radius. On the ends of this pipe are placed collars, one on each side of the cast-iron spool, and set for 8-in. stroke. A lever is fastened to one end of the pipe, which, when pulled quickly back and forth, gives the bags an up-and-down as well as a back-and-forth motion, and on coming up sharp against the collars also produces a sudden jar which assists in dislodging the fume from the bags. This device when handled properly will lower the pressure in the bags to 0.15

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in. of water. Good hand shaking will lower the pressure to 0.10 in. or slightly less.

It is important that the bags are so hung that when inflated they will stand straight; otherwise, upon shaking, the fune will collect around the thimble top, distorting the bottom of the bag and thus obstructing the flow of the gases into the bag, at the same time putting an undue strain upon the bag at this point.

Fume from blast furnaces carries enough sulphides and finely divided carbon to burn. The burned material, after moistening thoroughly, can be handled safely. Sluicing the fume from the compartment promises to be the quickest and most sanitary method, but filter pressing and the large amount of moisture left in the cake are objectionable. In any case, cleanliness should be demanded from all employees and no scattering of fume should be permitted around a bag house.

Bag filtration, with its high percentage recovery of fume, diffusion of gases, and simplicity of operation, is very satisfactory, and the bag house has well proved its worth to the lead smelter. Credit for its introduction into this branch of metallurgy is due Dennis Sheedy, Manager, and Dr. M. W. Iles, Superintendent, of the Globe Smelting & Refining Co.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Ajo Copper-Mining District

BY IRA B. JORALEMON, WARREN, ARIZ.

(Salt Lake Meeting, August, 1914)

THE Ajo copper district is in the heart of the Arizona desert, near the western boundary of Pima county. Gila Bend, the nearest railroad point, is 43 miles north of the camp, and the little Mexican border town of Sonoita is 30 miles south. Between Gila Bend and Sonoita, Ajo is the only settlement save for one or two small cattle ranches and the uncertain villages of nomadic Papago families.

The Little Ajo mountains rise a few hundred feet above wide desert valleys, beyond which are lava mesas. The camp itself lies at an elevation of about 1,900 ft. above sea level, in a little basin on the east side of the range, separated from the open desert by low hills. In the center of the basin the brilliantly iron and copper stained rocks of Copper mountain rise 150 ft. above the village. Many varieties of cactus and low desert bushes cover the hills, while in the large valleys there are clusters of mesquite, paloverde, and ironwood trees which furnish limited amounts of fairly good fuel. A scanty supply of water is obtained from wells in the desert and from prospect pits in the camp. The nearest abundant source of water is the Gila river, 50 miles north of Ajo. (See Fig. 1.)

## HISTORY OF THE AJO DISTRICT

The Ajo, next to Santa Rita, New Mexico, is claimed to have been the first copper district in the Southwest worked by Americans. In the '60's or even earlier high-grade native copper and cuprite ore was mined from shallow surface workings and hauled by bull team 400 miles across the desert to San Diego. From there it was carried in sailing ships to Swansea, Wales. Later, the ore was hauled to Yuma, floated down the Colorado river to the Gulf of California, and shipped to Swansea. Bat-infested old workings and the massive axles of the bull carts are shown to prove the truth of the stories.

From this early period until the beginning of the present century, the Ajo district was worked only in a casual and intermittent way. With the increased interest in copper mining during the past decade, the brilliant

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surface showing at Ajo furnished an ideal basis for stock companies. A succession of reorganizations and new stock issues resulted in little underground development, and the greatest depth reached in the two or three years of activity was hardly more than 100 ft. Rich bornite and chalcocite ore was shipped from several small veins, and stamp mills with concentrating tables were installed to treat lower-grade ores. But the long dry haul to the railroad made the shipment even of rich ore and con-

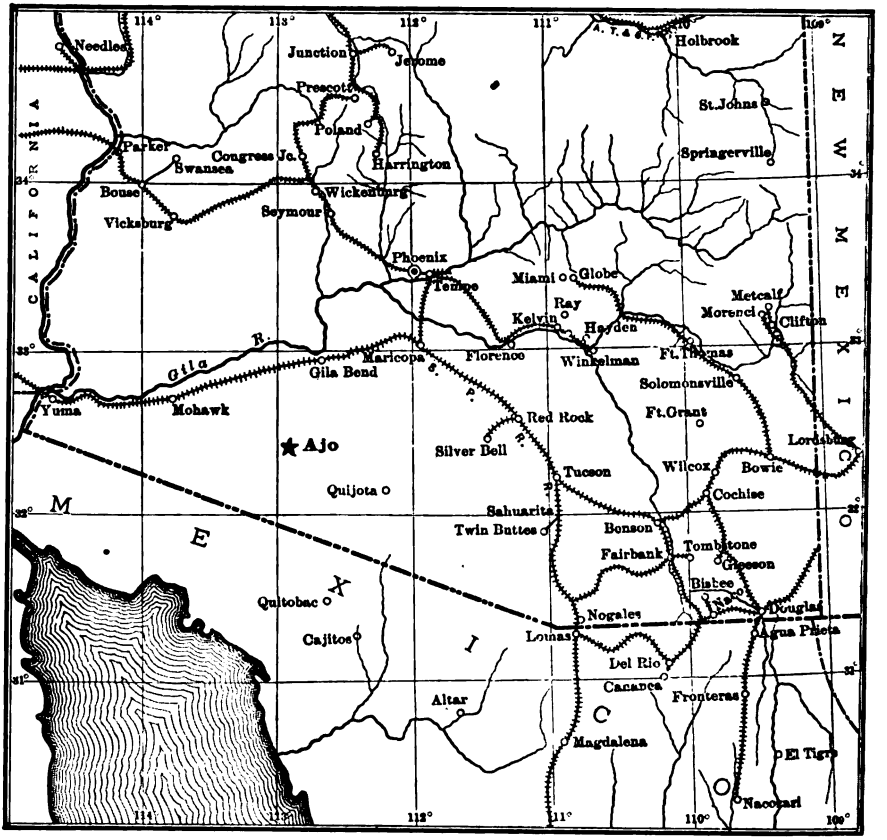


FIG. 1.—MAP OF SOUTHERN ARIZONA.

centrates hardly profitable. In an effort to avoid the cost of freighting ore, one of the companies fell a prey to the promoters of patent processes. The McGann vacuum smelter and a complicated hydrofluoric acid leaching plant still stand as monuments to the hopes of disappointed stockholders. The end of this period in the life of the district came with the panic of 1907, when the final products of the reorganizations—the New Cornelia Copper Co. and the Rendall Ore Reduction Co.—were forced to close down.

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The Ajo camp enjoyed a second short-lived boom in the winter of 1909–1910. The Lewisohn interests secured an option on the New Cornelia Copper Co., and Seeley W. Mudd and associates optioned the Rendall Ore Reduction Co. Diamond drilling and underground work were started on the New Cornelia property, and churn drilling and underground work on the Rendall. Many engineers came to examine or buy smaller properties, and claims were located for miles in all directions. Since the drilling did not give satisfactory results, both options were soon given up. The camp returned to a moribund condition, and until the fall of 1911 no work was carried on save for a little leasing on small high-grade surface veins.

Nearly all of this early work was in the lower ground surrounding the hard silicified outcrops of Copper mountain. A few shallow holes on Copper mountain had developed low-grade, very siliceous malachite ore, and three or four deeper holes had penetrated below the carbonate zone to a disseminated chalcopyrite and bornite ore assaying from 2 to 4 per cent. copper. Both the carbonate and the sulphide ores were so unusual in character that it was doubtful if they could be treated with profit by processes in common use even if large orebodies existed.

In the fall of 1911 the Calumet & Arizona Mining Co., John C. Greenway, General Manager, took an option on all the available stock of the New Cornelia Copper Co., and started diamond drilling to prove more thoroughly what lay beneath the highly stained outcrops of Copper mountain. As work progressed favorably, test-pitting was started to increase the speed of development and to check the results of drilling. Later, drifting was done in the sulphide zone to prove the continuity of ore between drill holes. This work has shown that the silicified iron and copper stained hills are the outcrop of a great low-grade copper orebody, covering an area of about 55 acres and reaching a maximum depth of over 600 ft. below the surface.

After this development was begun, options were taken on the property of the Rendall Ore Reduction Co. by James Phillips, Utley Wedge and others under the name of the Ajo Copper Co., and on the Childs group of claims, between the New Cornelia and the Rendall properties, by representatives of the United States Smelting, Refining & Mining Co. The latter option was given up after a few churn-drill holes had been sunk. The Phillips interests still hold the old Rendall property, but are not at present doing any development.

## Geology

Except for a local conglomerate, all the rocks in the Ajo district are igneous, and there is nothing to indicate the geologic ages. The earliest formation exposed is a series of rhyolite lava, breccia, and tuff beds.

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These vary from white hard rhyolite to reddish, soft, coarse volcanic tuff, with the hard rhyolite beds usually near the bottom of the series. Next in age is an intrusion of monzonite porphyry, which cuts and uplifts the rhyolite. In the center of the district, the porphyry is a coarse quartz-monzonite, with large orthoclase, plagioclase, and biotite crystals, and small variable amounts of quartz. The ground mass between crystals is generally white feldspar. This variety of the porphyry is distinguished by the pinkish orthoclase crystals. Further north, the porphyry grades to a granitic type, with less of the soda-lime feldspar, and much white orthoclase, biotite, and quartz. In the eastern part of the intrusion, the porphyry is finer grained and darker, without the pink feldspars. All through the porphyry mass there are local variations ranging from a fine-grained dark diorite to nearly white granite porphyry. Probably several minor intrusions closely followed the principal one, though the later dikes can seldom be distinguished either on the surface or in diamond-drill cores.

After the monzonite came a few dikes of diorite or diabase porphyry, usually dark gray in color, with square crystals of white feldspar in a very fine-grained ground mass. These are probably allied with the great mass of Tertiary andesite and basalt lava flows which cover hundreds of square miles of the surrounding desert region. This lava surrounds the Little Ajo mountains on all sides, beyond desert valleys, and small outcrops of it are left above the desert wash immediately northeast of the district. Probably at one time it covered the whole country. In a drilled well in the valley 6 miles northeast of Ajo, the thickness of andesite and basalt lava flows has been proved to be over 1,200 ft.

The most recent rock found at Ajo is a coarse conglomerate which lies between the hills and the valley to the east and south. The fragments in this conglomerate are rhyolite and monzonite, and the cementing material is limonite and silica, with traces of malachite. The age relation between the conglomerate and the Tertiary lava flows is not certain. Evidently the conglomerate is a local result of rapid erosion of the mineralized rhyolite and monzonite of Copper mountain, formed by the cementing by iron and silica bearing surface waters of rocks washed down the steep mountain side.

The only great alteration of the rocks, other than that accompanying the mineralization, is along rhyolite-monzonite contacts. Here the monzonite becomes much finer grained, and certain rhyolite beds have been recrystallized into speckled, fine-grained, gray crystalline rocks. The recrystallized rhyolite is sometimes hard to distinguish from fine-grained varieties of monzonite, especially where inclusions of rhyolite in monzonite are completely recrystallized. In the eastern part of the basin, the recrystallization of occasional beds of rhyolite strongly suggests dikes of fine-grained crystalline rock following rhyolite bedding.

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## Structure

The most important feature of the geologic structure is the large laccolith or batholith of monzonite porphyry, which uplifted the older rhyolite beds to form a dome. The crest of the dome was eroded away at a fairly recent time. This porphyry mass is 8 or 10 miles long by from 1 to 4 miles wide, with the long axis extending N. 20° W. The northern part of the intrusion forms high rocky hills north of the Ajo basin. The southern end forms Copper mountain, in the center of the basin. The copper deposit occurs in this southern part of the monzonite, where the axis of the laccolith is plunging to the southeast beneath the rhyolite. On top of Copper mountain, the small remnants of rhyolite lie fairly flat. On the west flank of the intrusion the dip of the rhyolite is steeply to the southwest, while the beds on the east flank dip to the southeast at an angle of about 20°. The pitch to the south of the domed rhyolite beds is fairly steep. All of the contacts are irregular.

Copper mountain and the Ajo basin surrounding it are thus formed by the monzonite and by the contact beds of rhyolite. East and west of the basin are hills of rhyolite lava, breccia, and tuff, with flat-lying beds of andesite lava and basalt in the distance, usually beyond gravel-filled valleys. South of the basin a large body of the red-brown conglomerate stretches away to the dark andesite and basalt mass of Black mountain.

There are many inclusions of rhyolite within the monzonite, and dikes or irregular intrusions of monzonite break up through the rhyolite, especially east of Copper mountain. Diorite or diabase dikes cut both monzonite and rhyolite, but apparently have had no important effect on structure or mineralization.

Both the monzonite and the rhyolite near it are thoroughly shattered by fractures which run in all directions. While some of these fractures are accompanied by considerable gouge, and may be faults of some importance, most of them can be traced for only a short distance, and are probably contraction fissures. In any portion of the porphyry there is generally a well-marked direction of strongest fracturing, but in the porphyry mass as a whole no such generalization can be made.

The somewhat idealized east-west sections (Fig. 2) show the most important points in the geologic structure of the Ajo basin.

## Mineralization

### *The Disseminated Orebody*

The mineralization in the Ajo district has formed a low-grade disseminated copper deposit with higher-grade veins in the monzonite, and narrow rich veins in the adjoining rhyolite. In the monzonite, the dis-



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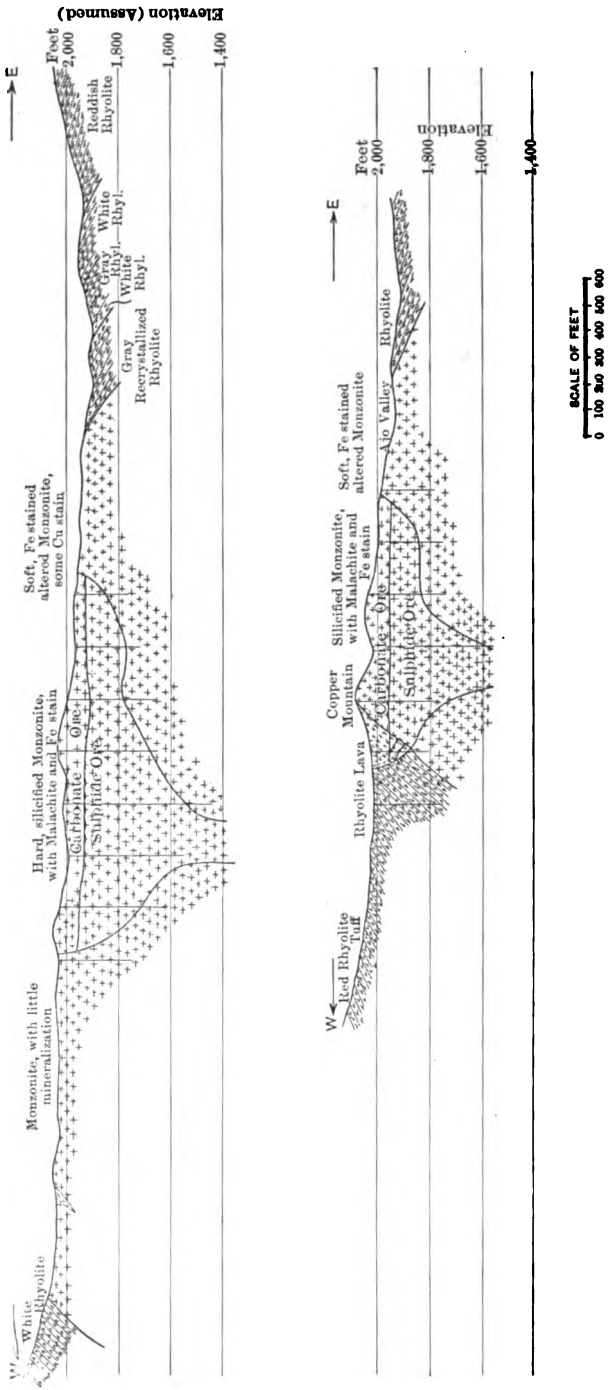


FIG. 2.—VERTICAL SECTIONS THROUGH NEW CORNELIA OREBODY.

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seminated deposit has a roughly pear-shaped outline, with the neck of the pear to the south. The outline of the orebody agrees almost exactly with that of Copper mountain and of the silicified, iron-stained hills north of the mountain. The area covered by the orebody is about 55 acres. The depth of the ore varies greatly. Around the outskirts of the orebody, it often extends less than 50 ft. below the surface, while in the center drill holes from 400 to 600 ft. deep have not found the bottom of the ore. In vertical section the deposit has the form of a gigantic elongated mushroom, the stem of which reaches below any point yet reached in drilling. The accompanying vertical sections give a fair idea of the form of the orebody.

Unlike Bingham, Ely, Santa Rita, and other low-grade camps, in which the copper is in the form of chalcocite, the disseminated ore in the Ajo district is a mineralization of the porphyry with chalcopyrite and bornite. Within the orebody, the monzonite is thoroughly shattered by a network of fractures. While the larger breaks often run about N. 20° W., parallel with the axis of the intrusion, other fractures have all possible directions. Along many of the fractures there are quartz veins from $\frac{1}{4}$ in. to 1 ft. or more wide. Between these veins the monzonite is more or less silicified, being often completely replaced by quartz. With the quartz, chalcopyrite and bornite were introduced into the mass, both in narrow seams or films along fractures and as very small flakes, disseminated through the porphyry. In most cases both the disseminated sulphides and those in seams or veinlets are required to bring the ore up to commercial grade. Within the main portion of the orebody pyrite is entirely absent or is present only in very small quantities. Chalcocite is rarely found except in very thin films just below the water level. There is a small and variable amount of magnetite disseminated through the ore. This is apparently earlier than the copper minerals, and may be an accessory original constituent of the rock.

Along some of the larger fractures the seams of bornite and chalcopyrite widen to veins an inch or more across. Where several of these veins lie close together and parallel, there are bands of ore from 10 to over 100 ft. wide, assaying from 3 to 5 per cent. copper. The grade of the ore in general is exceedingly variable, changing abruptly from slightly mineralized porphyry containing less than 0.5 per cent. copper to ore assaying over 3 per cent. As would be expected, the richer ore is in the more thoroughly fractured portions of the porphyry. The divisions of rich and lean material are often in more or less parallel bands, following in any part of the orebody the general direction of fracturing at that point. Since the bands are so narrow and irregular that the richer ore could not be sorted out without excessive expense, the whole mass has been considered as a solid orebody, the grade of which is lowered by the lean material included in it.

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### Sulphide Ore Primary

While in different parts of the orebody, and often within a few feet, the relative proportions of chalcopyrite and bornite vary greatly, on the whole the two minerals continue in approximately the same proportions in the deepest ore developed as in that a few feet below the oxidized zone. In the deepest ore there is no indication of any action by circulating surface waters. The general occurrence of the chalcopyrite and bornite seems to indicate that both minerals were formed during the original mineralization, and were not at all due to enrichment by solutions descending from the surface. L. C. Graton has examined in polished section some of the richer bornite ore from the New Cornelia orebody, and concludes that the bornite and chalcopyrite are both primary.[1] The variations in grade are due to differences in the intensity of the original mineralization, not to any effect of surface enrichment. In this fact lies the chief difference between the New Cornelia orebody and the disseminated deposits now being mined in other districts.

### Boundaries of Orebody Commercial

The boundaries of the orebody at Ajo are generally commercial. On the southwest side, the ore in the monzonite extends to the contact with the overlying rhyolite. In places the rhyolite beds for some distance from the contact are mineralized strongly enough to form an ore of commercial grade. But more often the mineralization in the rhyolite is confined to narrow films of chalcopyrite and bornite on joint planes, and the grade of the material is under 1 per cent. East of the orebody, the change from ore to lean material is caused not so much by a decrease in the amount of mineralization as by a change from chalcopyrite and bornite to pyrite. There is less silicification in the pyritic porphyry than in the ore, and the dioritic type of porphyry is more common. West and northwest of the ore, the change from commercial ore to material too lean to be considered ore is caused generally by a decrease in the amount of chalcopyrite, bornite, and quartz, with no great change in the character of the mineralization. West of the ore, the porphyry changes from the typical monzonite to a dioritic phase, while to the north there is a gradation to a more granitic porphyry.

Below the outer portions of the orebody, where the bottom of the ore has been developed, drill cores show that the ore gives place to less fractured, less highly silicified monzonite, containing very little chalcopyrite and bornite. Occasionally a sudden change to a more acid or more basic variation of the porphyry is accompanied by a drop from ore to lean

[1] L. C. Graton: The Sulphide Ores of Copper, *Trans.*, xlv (1913), Figs. 18 and 26, pp. 75 and 79; also private communication.

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material. The association of the ore with the normal, coarse-grained monzonite is very noticeable. Since the variations in the rock seem too slight to have had any selective influence on the precipitation of metallic minerals, it is probable that the formation of the coarse monzonite type of porphyry and the deposition of the ore were both dependent on the conditions existing at the top of the intrusive mass, below the uplifted rhyolite.

## Surface Alteration

The result of alteration by surface waters at Ajo is entirely unlike that in other low-grade copper districts. Instead of having an oxidized capping from which nearly all the copper content has been leached, the sulphide ore is capped by crushed, silicified monzonite, shot through with seams and stains of malachite, limonite, hematite, and a little chrysocolla. The rock is still hard, though the feldspars are kaolinized to a greater or less extent. Often kaolinized feldspar crystals are stained bright green by malachite. The actual surface of the hills is tinted by iron oxides a deep red brown, with brilliant copper staining in protected places under the cliffs and wherever the surface rock has been shot away. Over an area of 30 or 40 acres it is hard to find even a small piece of rock which on breaking does not show copper stains and seams of malachite. While there is a little chrysocolla in veinlets, and occasionally cuprite, chalcocite, and bornite in richer veins, it is safe to say that over 85 per cent. of the copper in the oxidized zone is in the form of malachite. Now and then disseminated chalcopyrite and bornite are left unaltered in hard ore between fractures. But such remnants of sulphides are unimportant compared with the great mass of carbonate ore.

As in the case of sulphide ore, the assay value of the carbonate ore varies greatly. Rich and lean bands alternate. Sometimes, along large fractures, the rock is softened and leached, with less copper than in the adjoining harder rock. But more commonly the variation in values is not accompanied by any change in the degree of alteration of the rock. On the average, the copper content of the oxidized zone is constant from the surface to the bottom of the zone, and is almost exactly the same as that of the underlying sulphide ore.

The oxidized ore continues down to an almost horizontal plane, about 20 ft. below the deepest arroyos and 150 ft. below the highest hills. The remarkable regularity of the dividing line between sulphide and carbonate ore is shown by the vertical section. This line, or rather plane of demarcation, agrees almost exactly with the present ground-water level. The transition from carbonate to sulphide ore is very abrupt. Generally drill cores show less than 5 ft. of partly oxidized ore, the classification of which is doubtful. Occasionally, where leaching is unusually great along large fractures, bands of carbonate and sulphide ore alternate for

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15 or 20 ft. vertically. But on the whole, the amount of sulphide ore above the plane, or of carbonate ore below it, is very small. There is also no appreciable enrichment at the top of the sulphide zone, with a few local exceptions. This sudden change on a horizontal plane from carbonate to sulphide ore will greatly simplify both the mining and the treatment of the New Cornelia ore.

East of the orebody, where the disseminated copper sulphides give place to pyrite, the oxidation extends to practically the same horizontal plane at water level. The character of the oxidized zone, however, is entirely different. Instead of being hard and siliceous, the rock is soft, with much kaolin and soft yellow limonite. Copper staining is comparatively slight. Instead of standing up in bold brown and green cliffs, the outcrops are reddish and form valleys or low hills. The oxidized zone contains much more limonite and alumina, but less silica and very little copper. At the top of the lean pyritic material there is a 5 or 10 ft. zone of enriched chalcocite ore assaying from 1.5 to 3 per cent. copper. In short, the conditions in this part of the Ajo basin are almost exactly similar to those in the great disseminated chalcocite districts, except that the primary mineralization was too lean in copper, or the leached zone was too shallow, to form an enriched orebody of commercial size.

## Veins in Rhyolite

It was not the low-grade orebody, but the rich veins in the surrounding rhyolite, which first led to the exploitation of the Ajo district. One strong vein outcrops on the Quien Sabe claim of the New Cornelia Copper Co., on the lower slopes of Arkansas mountain, and four or five veins have been partly developed on the Ajo Copper Co. property. Rich malachite and cuprite ore from 6 in. to 3 ft. wide outcrops in these veins, which follow steep fractures in hard, slightly iron and copper stained rhyolite. High-grade cuprite and copper-glance ore, with a little native copper, was encountered a few feet below the surface. At a depth of about 50 ft., the glance begins to give place to bornite. Usually the center of the vein is very rich bornite and chalcocite ore from 1 in. to 4 or 5 ft. wide, and on both sides of this high-grade streak the shattered rock contains stringers of bornite and chalcocite, which make it a good concentrating ore. In the early operations in the district, considerable stoping was done in several of the veins to a maximum depth of over 100 ft. The stopes are seldom more than 6 ft. wide, and the large dumps show that much of the material taken from stopes was too low grade to treat with profit. In the Ajo Copper Co. property, one rich bornite vein from 1 to 3 ft. wide was developed for nearly 300 ft. down the dip. The high-grade stringer continues to the bottom, but the mineralization of the walls appears to decrease in depth.

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Most of the veins are associated for at least part of their length with dikes of monzonite, which tend to widen with depth. The monzonite adjacent to the veins is often mineralized enough to form a 2 or 3 per cent. ore, with values in malachite near the surface, and in chalcocite and bornite below a depth of 50 to 75 ft. In one or two places, lenses of this disseminated ore in monzonite dikes reach a width of over 50 ft., and promise to yield a considerable tonnage. They have as yet not been developed enough to prove how deep the ore will continue, and whether or not they are of great importance.

## Genesis of Ore and Geologic History

The genesis and geologic history of the Ajo ore seem unusually easy to trace. After the monzonite intrusion had uplifted the rhyolite, the slow cooling of the porphyry was accompanied by considerable contraction. This resulted in a thorough jointing and fissuring of the monzonite, especially near the rhyolite contact, and in a less complete fracturing of the rhyolite itself. Near the center of the intrusion, some of the fissures continued to great depth. Probably soon after the solidification of the outer layer of porphyry, hot mineral-bearing solutions rose along these deep fractures. The solutions were heavily charged with iron, sulphur, silica, and later copper. The iron and sulphur were carried through the joints and fissures in the rock far from the source of the mineralization, and caused a general, though scanty, pyritization of the monzonite mass. This pyritization was heaviest in the eastern part of the intrusion. Later, as the proportion of silica and copper in the solutions increased, these solutions rose through the larger fractures in the monzonite until they encountered the impervious, less thoroughly fractured overlying beds of rhyolite. These dome-shaped beds acted as a dam, stopping the upward flow of the mineralizing solutions and causing them to spread out through the jointed monzonite on both sides of the large, deep-seated fractures. Here they remained imprisoned until they gave up their mineral content, depositing veins of quartz, chalcopyrite, and bornite along the fissures and joint planes, and partly replacing the rock itself with the same minerals. The mineralization was greatest near the large central fractures through which the solutions had risen, and in the more thoroughly jointed portions of the monzonite near rhyolite contacts. In this manner was formed the mushroom-shaped disseminated orebody, grading on the sides and bottom to rock less thoroughly mineralized, or mineralized with iron instead of copper sulphides.

Some of the larger fractures, usually accompanied by monzonite dikes, extended for a considerable distance up into the rhyolite. Along these fractures the rich bornite veins in rhyolite were deposited, sometimes extending a long distance from the large disseminated body. The por-

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phyry dikes accompanying the veins were more or less mineralized, and small quantities of chalcopyrite, bornite, and pyrite were deposited in the rhyolite walls of the veins.

Probably soon after the end of the period of mineralization, before the rhyolite capping of the ore had been eroded away, the whole country was covered by a great thickness of andesite lava and basalt. This covering protected the ore from surface alteration until the present geologic period.

Erosion in early Quaternary times must have been very rapid. Deep canyons were cut in the recent lavas. The covering was stripped off from the Ajo monzonite, and much of the orebody itself was rapidly washed away. The erosion was so rapid that oxidation did not keep pace with it. The copper in the ore that was eroded away was either left in the rock fragments washed down the hillsides or was carried off by the surface drainage. The detritus from the eroded portion of the orebody filled the canyons and valleys that had been cut in the lava east and south of Ajo. Part of the detritus was cemented into a hard ferruginous conglomerate by surface water which had flowed over the pyritic porphyry east of the ore and had taken into solution some of the iron of the oxidized pyrite. Boulders of oxidized ore remain scattered through the conglomerate and through the valley wash or gravel for several miles from Ajo.

Following this period of rapid erosion, very stable conditions have lasted up to the present time. The water level has remained nearly stationary, at a rather shallow depth, long enough to allow thorough oxidation of the rock down to this level. Where the mineralization was chiefly with pyrite, much sulphuric acid was formed by the oxidation of the pyrite to limonite. This acid thoroughly softened and kaolinized the feldspars of the monzonite, and helped take into solution the copper from the traces of chalcopyrite which accompanied the pyrite. This copper was carried down by the descending surface waters and was redeposited at water level to form the thin layer of chalcocite ore.

In the siliceous chalcopyrite and bornite orebody, it seems that little sulphuric acid was formed during the oxidation. The descending surface water, probably always small in amount in that arid climate, contained an excess of lime and of carbonic acid. The result of this condition was that as soon as the copper sulphides were oxidized, the copper was precipitated almost in place in the form of malachite. The iron of the chalcopyrite and bornite remained in place in the form of limonite and hematite. The silicified monzonite was very slightly altered, because of the small quantities of sulphuric acid, and remained hard and resistant to weathering, forming steep hills. The little sulphuric acid which was formed probably reacted at once with the calcium carbonate in the oxidizing waters to form calcium sulphate. Thin seams of the resulting gypsum occur here and there in the carbonate ore, and there is much gypsum in the gravel of the valleys.

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The result of this cycle of changes was a large body of low-grade primary bornite and chalcopyrite ore, oxidized above a horizontal plane at the present water level to a malachite ore of exactly the same grade as the sulphide. The ore was deposited from rising deep-seated solutions. There is no overburden of lean or barren rock, since the oxidation was accompanied by practically no leaching or transportation of the copper from the primary ore. The ore rises above the surrounding barren ground, and continues without any considerable increase or decrease in values from the actual surface to the bottom of the orebody.

### Development of the Low-Grade Orebody

The Calumet & Arizona Mining Co. developed the disseminated orebody on the New Cornelia property by diamond-drill holes and by test pits. The probable ore-bearing ground was co-ordinated with east-west and north-south lines at 200-ft. intervals, and drill holes were sunk at the intersections of co-ordinate lines. The drilling was done under contract by the E. J. Longyear Co., of Minneapolis, Minn. Sampling was under the direction of representatives of the Calumet & Arizona Mining Co. Drill holes were sampled in 5-ft. sections. All of the flow of sludge was caught in barrels and settled. When the end of the 5-ft. section was reached the water in the barrels was decanted off and the sludge sample was dried and quartered down to 3 or 4 lb. The rods were pulled at least every 5 ft., and core samples were taken at even 5-ft. intervals where possible. Both core and sludge samples were sent to the assay office of the Calumet & Arizona Mining Co., in Bisbee, for analysis. Small portions of both core and sludge for every 5 ft. of drilling were kept for future reference in labeled tin boxes.

Owing to the thoroughly fractured, uneven nature of the rock, the recovery of core was low, and neither core nor sludge samples alone were satisfactory. To obtain an accurate assay value for the ore developed, the length of core for every 5-ft. advance was measured, and on the basis of this length of core the sludge and core assays were combined to give a final value which represented all the material removed from the hole during the 5-ft. advance. The E. J. Longyear Co. furnished a chart which greatly simplified the work of combining core and sludge samples.

After drilling had been in progress for about six months, test-pitting was started to check the results of drilling. The pits were about 4 by 6 ft. in size, and were sunk with windlasses. Drilling was done by hand. Mexican and Papago Indian miners were employed for this work, usually under contract, and soon became so efficient that to a depth of 50 ft. the test pits were sunk more cheaply than diamond-drill holes. During the last year of development, test pits 50 ft. deep were sunk on co-ordinate points in advance of drilling, in order both to expedite and to lessen the cost of development.

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In sinking test pits, every tenth bucket windlassed from the hole was taken for a sample. The large samples thus obtained for every round shot were crushed, quartered, and sent to Bisbee for analysis. After test-pitting was finished, the pits were resampled in 5-ft. sections by channeling. All four sides of the pits were channeled with even rectangular vertical grooves 6 in. wide by 3 in. deep, giving about a 500-lb. sample for every 5-ft. section. These samples were crushed, quartered, and sayed at Ajo, and pulp samples were reassayed in Bisbee. The channel samples averaged about 0.15 per cent. lower than the bucket samples and were taken as the final samples of the test pits.

When test pits were sunk below water level into sulphide ore, a small flow of water was encountered. This made sinking slow and expensive. Therefore only a few hundred feet of sinking was done in sulphide ore. To more thoroughly check the results of drilling in sulphide ore, and to prove that the ore was constant in grade between drill holes, drifts were run on co-ordinate lines from the bottoms of two of the deepest test pits. Raises were put up from these drifts to check drill holes. To give a fair sample of material cut by the drifts, the ore broken by every round of holes was hoisted separately and dumped on an iron plate. From here it was shoveled into cars or over the dump, and every tenth shovel was thrown into a bin for the sample. These large samples, each representing a carefully measured round, were taken to the mill, crushed, quartered, and sent to Bisbee for analysis.

Up to September, 1913, when development work was stopped, 84 diamond-drill holes had been sunk, varying in depth from 200 to 1,000 ft. The total footage of diamond drilling was 23,097 ft. Nineteen drill holes were stopped in ore.

In all, 77 test pits were sunk, with a total footage of 3,955 ft. Of this test-pitting, 3,606 ft. were in carbonate ore, and 349 ft. in sulphide ore; 1,059 ft. of test-pitting checked drill holes in carbonate ore, 175 ft. checked drill holes in sulphide ore, and 2,721 ft. were in advance of drilling. The total amount of drifting in sulphide ore was 1,513 ft. The combined sinking and raising on drill holes in sulphide ore amounted to 317 ft.

## Results of Development

The sinking, drifting, and raising proved that the value of ore indicated by drilling was very accurate. The channel samples of test pits in carbonate ore averaged 0.005 per cent. lower than corresponding diamond-drill samples; the test pits and raises in sulphide ore averaged 0.05 per cent. lower than corresponding diamond-drill samples; and the drifts in sulphide ore averaged 0.26 per cent. higher than the assay value of blocks of ore through which the drifts were run, as indicated by drill holes at the corners of the blocks. In making the ore estimate, diamond-

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drill samples were accepted wherever drilling was done, and channel samples of test pits were accepted where sinking was done in advance of drilling.

## Ore Estimate

The estimate of ore developed on the property of the New Cornelia Copper Co. at Ajo by the Calumet & Arizona Mining Co. is as follows:

| Ore estimate: | Tons | Copper, Per Cent. |
|---|---|---|
| Carbonate | 11,954,400 | 1.54 |
| Sulphide | 28,303,600 | 1.50 |
| Total | 40,258,000 | 1.51 |
| Available by steam shovel: | | |
| Carbonate | 11,954,400 | 1.54 |
| Sulphide | 20,526,800 | 1.54 |
| Total | 32,481,200 | 1.54 |
| Not available by steam shovel: | | |
| Sulphide | 7,776,800 | 1.40 |
| Rock which must be removed to make steam-shovel tonnage available: | | |
| Rock in carbonate zone | 708,400 | 0.65 |
| Rock in sulphide zone | 2,600,000 | 0.63 |
| Total rock | 3,308,400 | 0.63 |

The gold and silver content of this ore is very low, generally amounting to less than 15c. per ton.

In computing the ore tonnage, no ore was included which assayed less than 1 per cent. copper. The estimate of ore available for steam-shovel mining depends, of course, on the assumed maximum grade of tracks in approaches and pit, and on the amount of lean rock in the sides and bottom of the pit which it will pay to move in order to give access to deep ore. The proportion of the Ajo ore mined by steam shovel will probably be greater rather than less than that given in the above estimate.

The fact that the Ajo ore outcrops at the surface, with no barren overburden, not only will do away with the high initial stripping cost which generally attends steam-shovel mining, but also will prevent a loss of ore or a vitiation of grade through a mixture of ore with capping. This feature should make the mining grade and tonnage agree very closely with the estimate.

## Treatment of Ores

While the question of the treatment of the New Cornelia ores has not as yet been fully settled, enough work has been done to prove that the ore can be treated at a good profit by processes in use in other districts. Experiments on ore treatment have been carried on in Douglas and elsewhere for about a year and a half. This work has been under the direc-

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tion of John C. Greenway, General Manager, with the advice of Dr. L. D. Ricketts. The first important tests were made by Stuart Croasdale on leaching carbonate ores. These tests showed that over 80 per cent. of the copper in carbonate ore crushed to ½ in. can be leached out by sulphuric acid in from 36 to 72 hr., and that this copper can be precipitated at a reasonable cost by scrap or granulated pig iron. Good percolation was secured in leaching tanks with an ore column over 10 ft. high. Owing to the very high insoluble content of the ore, the acid consumption was low enough so that regeneration of sulphuric acid or ferric sulphate from the solutions from which the copper has been precipitated is not necessary to the success of the process. When it had been shown that leaching with fresh sulphuric acid and precipitation on metallic iron would yield a fair profit on the carbonate ores, experiments were directed toward lessening the cost of leaching and precipitation by using processes in less common use. The making of sponge iron for a precipitant by metallizing the calcines obtained by roasting pyrite ores was successful only when tried on a very small scale. A process developed by F. L. Antisell, of the Raritan refinery, involving electrolytic precipitation with an insoluble anode made largely of coke, and regeneration of ferric sulphate, has been very successful on a small scale. A. D. Jamieson, who was associated with Mr. Antisell in these experiments, is now testing this process on a larger scale in Douglas. Thus far the work has been successful. Utley Wedge has met with encouraging results in a process involving a sulphatizing roast of mixed carbonate and sulphide ores, with subsequent leaching. It seems reasonably certain that one of these newer processes will make possible a great saving in cost of copper over simple leaching with sulphuric acid and precipitation on iron.

The Ajo sulphide ores can be concentrated by ordinary wet methods, but the thin flakes of bornite and chalcopyrite slime badly, and the recovery is low. The chalcopyrite and bornite ore is beautifully adapted to flotation, and tests made by the Minerals Separation Co. show a high recovery. Very fine crushing is necessary to allow the separation of the sulphides, and the hardness of the ore will make the crushing fairly expensive. It seems certain, however, that sulphide as well as carbonate ore can be mined and treated with a good recovery at great profit.

### Future Plans of the New Cornelia Copper Co.

With a great tonnage of ore developed, and the general outline of the method of treatment in sight, the most important problem at Ajo is the development of a sufficient supply of water for the mill. To solve this question, two deep wells were drilled in the large valley northeast of Ajo. Water was found in both wells at depths of from 550 to 750 ft. A boiler plant and compressors were installed at one well, and the water was

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pumped from the drill hole by an air lift. The limit of capacity of the lift was about 175 gal. per minute. A pumping test at this rate lasting over two weeks did not exhaust the water from the hole. With this flow from an 8-in. drill hole, it is reasonably certain that a shaft, with a little drifting, will furnish an ample supply of water. A shaft has been started for this purpose.

To determine the details of treatment, a large experimental plant is being started at Ajo. The units in this plant will be of practically the same size as those in the final plant. Large leaching tanks will be erected, and an electrolytic precipitation plant. Probably both the Antisell process and another process involving electrolytic precipitation, devised by Messrs. Pope and Hahn, of New York, will be tested in this experi-

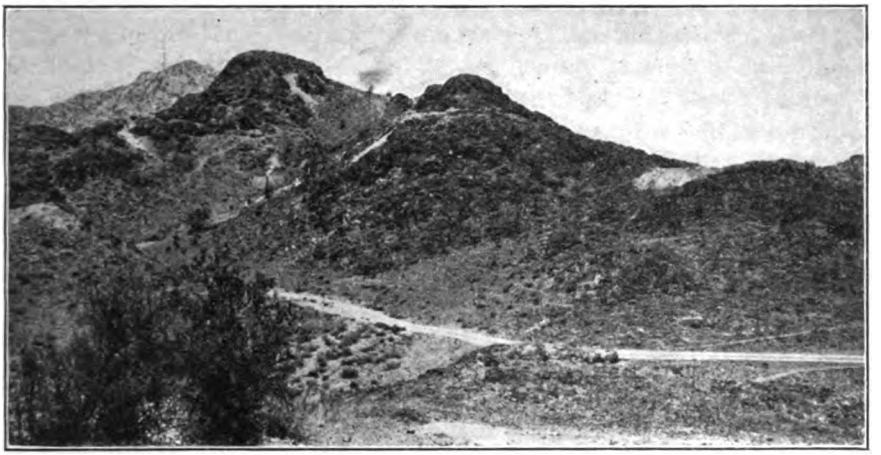

FIG. 3.—NEW CORNELIA OREBODY, AJO. LOOKING NORTH.

mental mill. Since it will be several years before mining of the carbonate ores proceeds far enough to expose sulphide ore, the development of a process for treating sulphide ore can be carried on more leisurely.

A railroad will be the next step in bringing the New Cornelia property to the productive stage. Preliminary surveys have been made from Ajo both to Gila, on the Southern Pacific Railroad, and to Tucson, the junction point of the Southern Pacific and the El Paso and Southwestern systems. The distance to Gila is about 44 miles, and to Tucson about 130 miles. Both lines are through gently sloping desert country, and railroad building should cost less than $20,000 per mile. It has not yet been determined which route the railroad will follow.

The orebody already developed at Ajo will supply a 4,000-ton mill for over 26 years, or a 6,000-ton mill for 18 years. The indications are that the life of the mine will be greatly lengthened by the development of

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a large tonnage of deep ore along the fracture zone in the center of the ore-body. The vital points in the question of ore treatment have apparently been successfully solved. For the next quarter of a century the Ajo will be one of the great copper districts of the Southwest.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Electrical Fume Precipitation at Garfield

BY W. H. HOWARD, GARFIELD, UTAH

(Salt Lake Meeting, August, 1914)

As the result of a series of analyses and volume determinations of gases discharged from the converters at the Garfield Smelting Co.'s smelter at Garfield, Utah, it was found that a considerable amount of lead fume was escaping from the stacks of the converter plant, and that if this lead could be recovered at a reasonable expense an attempt should be made to collect it.

Two methods suggested themselves: (1) filtration through bags; (2) electrical precipitation of the condensed fume by a Cottrell plant.

## *Filtration through Bags*

Tests were made in filtering converter gases through the ordinary woolen bags of lead-smelter bag houses to determine how long such material could withstand the corrosive action of the gases. While blowing on white metal only, the fabric was usually destroyed in 10 to 14 hr. In tests conducted in filtering gases when blowing on matte or slagging stage exclusively and when relatively considerable lead was being volatilized, bags in one instance lasted 60.5 hr. It therefore was evident that the Garfield converter gases did not contain in themselves sufficient neutralizing elements for the purpose of direct bag filtration, even during the slagging stage.

Such results were more or less anticipated, but it was thus proved that under all conditions of blowing Garfield mattes a very considerable amount of neutralizing agent, zinc oxide or lime, or both combined, as in the Sprague process, would be required to render bag filtration possible. Again, owing to the constantly changing quantities of $SO_3$ from a given battery of converters, depending on the number blowing at a given time and on the stages of the blow, it would be practically impossible to feed automatically the required amount of neutralizer. In order to safeguard the bags sufficient neutralizer would have to be continuously added to meet the maximum $SO_3$ output, thus wasting neutralizers and giving an undesirably diluted final product. Such a condition is entirely different

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from what would prevail in a battery of roasters, where a fairly constant volume of gas of fairly constant tenor can usually be maintained.

Having calculated the amount of neutralizer required, it was estimated that the cost of a bag house and its operation and the treatment of the resultant diluted product would not prove a commercially satisfactory proposition.

### *The Cottrell Process*

As the complete elimination of $SO_3$ under existing conditions at Garfield was not imperative, a method which would recover the valuable constituents of the gases was the one to be considered.

The volume of gas, including entrained air at the converter hood produced from each Peirce and Smith converter, calculated at standard temperature and pressure, was found to be approximately 25,000 cu. ft. per minute, or an average of over double that volume at flue temperature.

An average of four to five converters is operated under normal plant conditions and practically never are over four blowing simultaneously, equivalent to a production of 200,000 cu. ft. of gas per minute at flue temperature for the whole department.

The ratio of $SO_3$ to $SO_2$ has been very variable, depending on the character of the matte treated and on the period of the blow. The average of several determinations showed 1 to 18, but at times it has been as high as 1 to 8.

The Cottrell process described in its simplest terms consists in the conversion of a high-tension alternating current of 30,000 to 60,000 volts into a direct intermittent current of the same potential, and the application of this current to certain electrodes suspended in a flue conveying the gases to be treated. These electrodes in their simplest form consist of narrow suspended metal plates regularly spaced 5 to 12 in. apart, depending on voltage, and between adjacent plates, parallel and equidistant from them, is stretched a fine wire to every space, the wires forming the discharge electrodes and the plates the collecting electrodes. There is thus maintained through a silent or glow discharge an electrically charged field through which the gases travel and all dust or condensed fume or moisture becoming electrified is deposited on the plates.

### *Cottrell Plant and Work at Coram, Cal.*

From an inspection of the Cottrell plant then installed at the works of the Balaklala Cons. Copper Co., at Coram, Cal., it was thought that the Cottrell process might be advantageously used for the recovery of the lead fume from the converters at Garfield; if not completely, then to an extent that might prove profitable. The treaters consisted essentially of chambers in which were suspended sheet-iron plates 6 in. wide by 10 ft. long and

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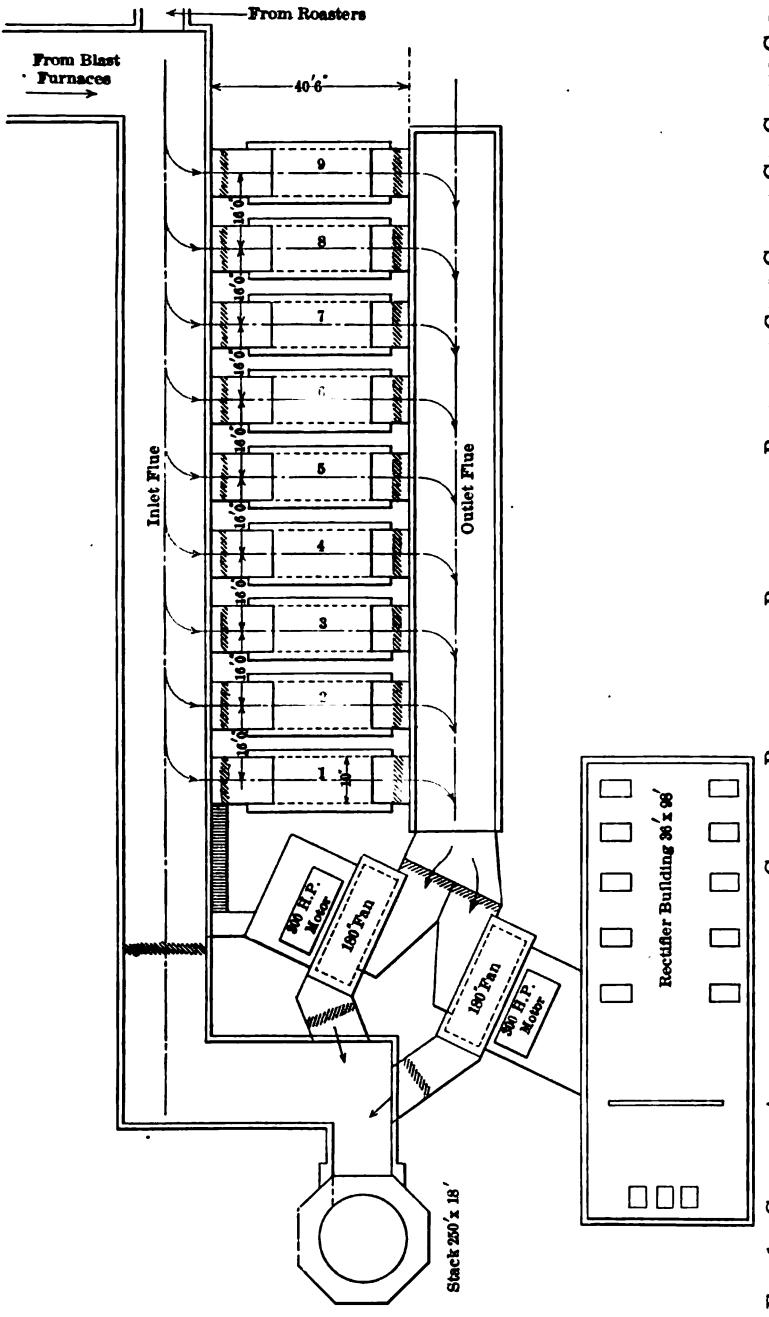


FIG. 1.—GENERAL ARRANGEMENT OF THE COTTRELL PRECIPITATING PLANT OF THE BALAKLALA CONS. COPPER CO., CORAM, CAL.



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5 in. apart, these forming the collecting electrodes. The discharge electrodes consisted of two iron wire strands in which were twisted pieces of asbestos fiber, forming numerous points for the discharge of a high-ten-

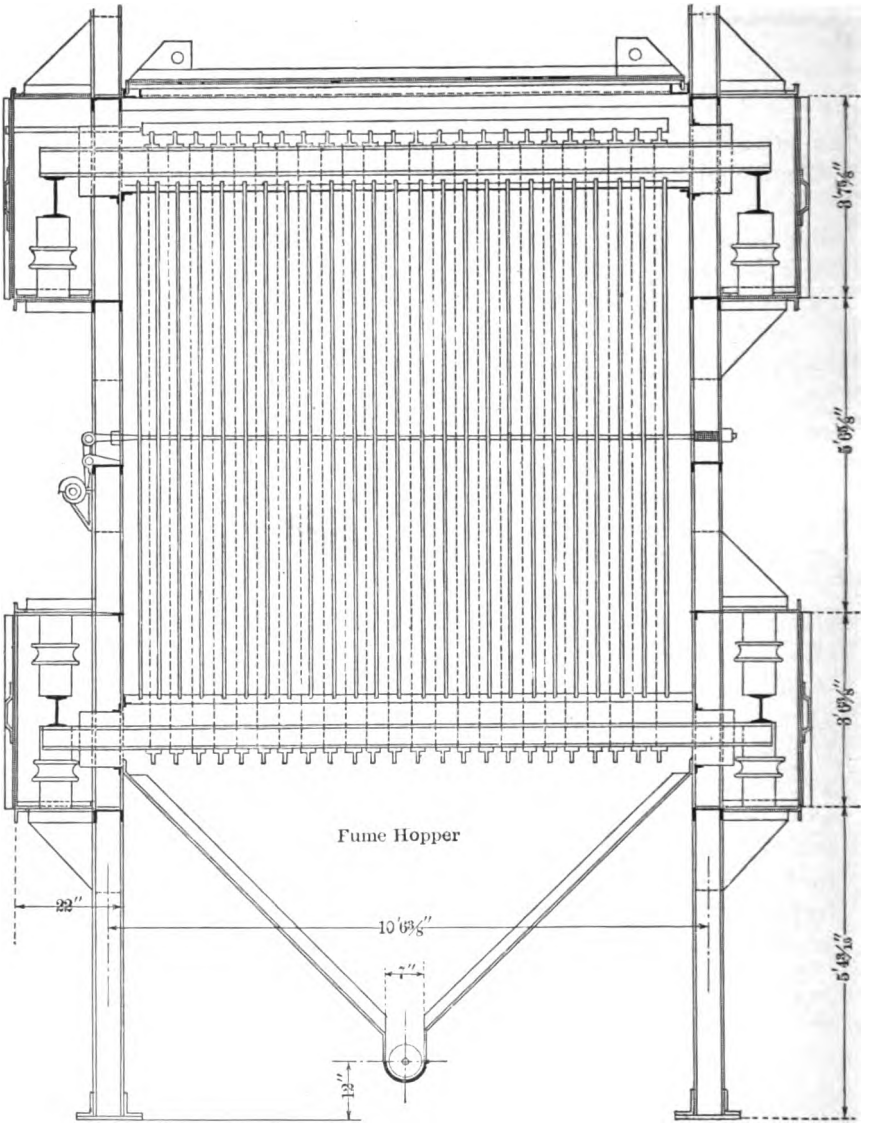


FIG. 2.—SECTION THROUGH COTTRELL PRECIPITATING UNIT, BALAKLALA CONS. COPPER CO.

sion direct intermittent current of 28,000 to 30,000 volts. By referring to Figs. 1 and 2 it will be seen that the installation consisted of nine units,

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eight operating and one being cleaned, and it might be supposed that a proper distribution could not be secured with such an arrangement as shown, but with louver dampers at the ends of each unit a fairly even flow through all units was obtained. The chief mechanical defects in a unit of this description are the spaces between the ends of the plates and the roof, and also along the bottom of the unit, where gases would escape treatment.

I was informed that the apparatus worked best with gases between 120° and 140° C. A temperature approaching but not less than 120° seemed the most desirable, though evidently, in the light of our present knowledge, not sufficiently low to condense all smoke-forming elements, which on being discharged into the atmosphere would become visible as smoke.

On my visit there in the spring of 1911 the recovery of solids from the smoke was usually 80 to 90 per cent., an attainment not sufficient to meet the demands of the courts. It was my belief, however, that if an extension of time had been allowed the elimination required would have finally been accomplished.

### *Preliminary Experimental Work at Garfield*

Actuated by this faith in the possibilities of the process to successfully treat converter gases, it was decided to carry on experiments at Garfield with a view to determining the best working conditions as to temperature, acidity, etc., and to discover the most efficient form of electrodes for the purpose. This work was intrusted to Mr. Rathbun, formerly chief electrician at the Balaklala smelter, who was fully familiar with the process as conducted there, and R. F. Barker, field chemist at Garfield.

This experimental work was started in August, 1911. Equipped with the necessary apparatus to produce a 20,000 to 30,000 volt direct intermittent current, the regulation plates and pubescent electrodes were first tried. Then iron pipes of different diameters and of varying lengths were experimented with. It was assumed that a pipe with an axially placed wire would give more uniform electrical treatment than the plates and wires, and the experiments proved this. With the current available of 30,000 volts, it was finally determined that 5-in. pipes 10 ft. long placed vertically gave the best clearances if the velocity of the gases treated did not exceed 15 ft. per second through the pipes; and finally the pubescent electrode, which was a source of considerable trouble, was replaced by a No. 22 gauge steel wire.

Pipes of larger diameter with the application of higher voltage were considered, but it was thought that the smaller pipe would secure better distribution; and having proved the efficiency of the 5-in. pipe it was

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decided to adopt that size, leaving the use of large-pipe and high-voltage installations to future development.

### 50,000 *Cu. Ft. Converter-Gas Treater at Garfield*

Having proved the greater efficiency of the pipe as compared with the plate as a collecting electrode and substituting a single wire to replace the regular pubescent discharge electrode, a plant was designed for the treatment of 50,000 cu. ft. of converter gas per minute.

The treater, as illustrated in Figs. 3 and 4, consisted of 608 5-in. iron, pipes (well casing), 10 ft. long, placed vertically over corresponding openings in a floor, similarly to the flues of a vertical boiler. Centrally, through each pipe, was stretched a No. 22 gauge steel wire. The treater was divided electrically into four sections so that different voltages could be applied to different parts of the plant.

Current was taken from a 2,300-volt alternating-current line stepped up to 23,000 to 30,000 volts. This high-voltage alternating current was by means of revolving rectifiers converted into an intermittent direct current of the same potential for the precipitating electrodes. The rectifiers were run by three-phase synchronous 220-volt motors, power to operate which was taken from the same source as that converted at the rectifiers, but stepped down to required voltage.

On the assumption of the treatment of 50,000 cu. ft. of gas per minute through 600 pipes, the average velocity through the pipes would be about 10 ft. per second, or the gases would be under treatment for 1 sec. in the 10-ft. pipe.

On Dec. 6, 1912, the gases from one converter were turned into the treater, being drawn through it by the draft of an adjacent stack 150 ft. high. It was discovered in the preliminary experiments that during blowing on white metal to copper, when a sufficient amount of sulphuric acid was present in the gas, the coating on the pipes from the condensed fume was conductive and the process offered no difficulties; but that during the slagging period, with a small amount of $SO_3$ present, a non-conductive coating was formed in the pipes and after a short time the current was interrupted and further precipitation ceased. It was found, however, that by the injection of a small amount of moisture in the gases before reaching the treater the coating in the pipes was rendered conductive and the process worked satisfactorily at about 90° C. With mixed gases from the slagging and white metal stages, as would occur under normal operations with three or more converters running, no addition of moisture was found necessary. But being limited to the treatment of the gases from one converter, fine sprays of water were injected into passing gases during the slagging stage, amounting to about 3 to 5 per cent. of moisture in the dust collected. Owing to occasional arcing

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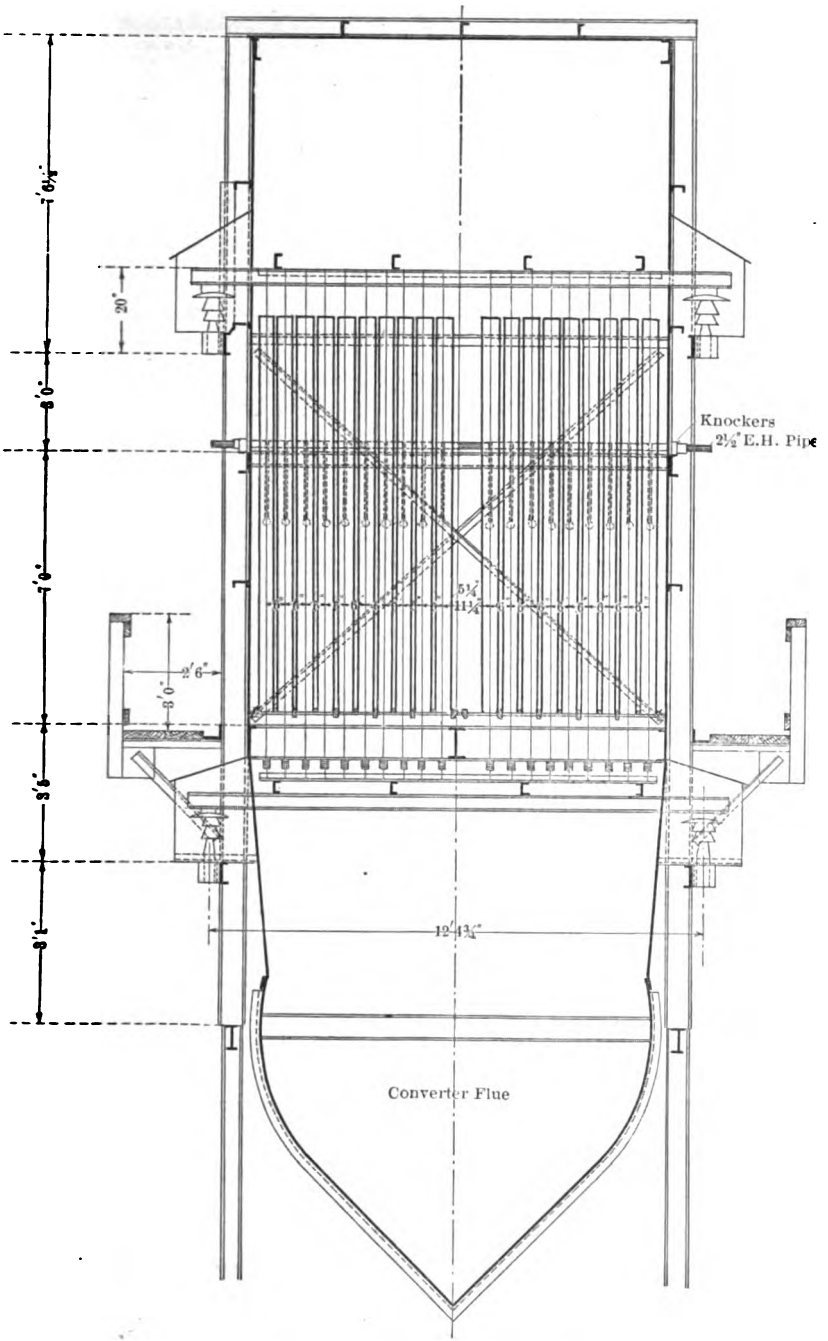


FIG. 3.—TYPICAL CROSS-SECTION OF EXPERIMENTAL COTTRELL PRECIPITATING PLANT OVER CONVERTER FLUE.

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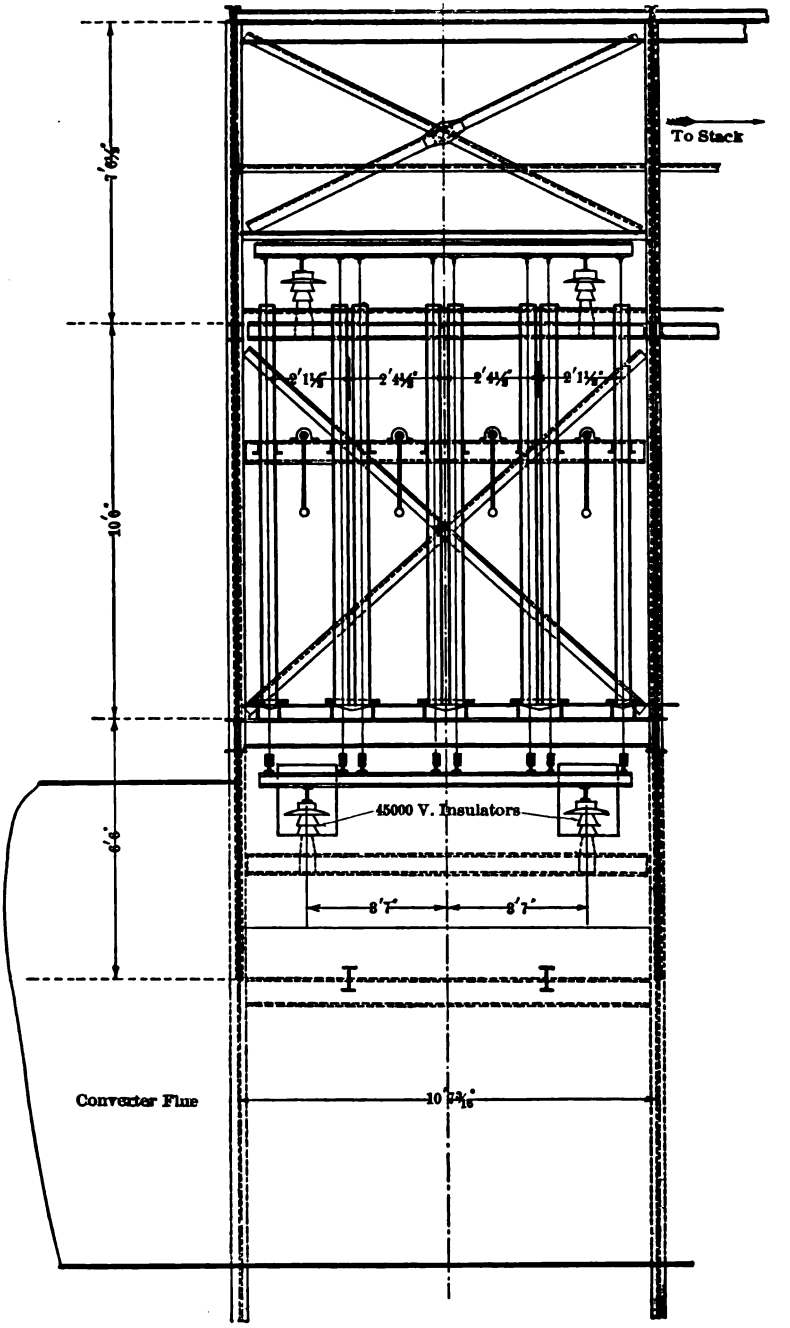


FIG. 4.—LONGITUDINAL SECTION OF ONE UNIT, EXPERIMENTAL COTTRELL PRECIPITATING PLANT.

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the No. 22 wires were at times burned off, but with No. 14 gauge wire this trouble was practically eliminated.

The plant was run continuously for several weeks and from analyses of the escaping gases it was determined that the average recovery of lead was 97.25 per cent. The dust collected ran 41 per cent. lead, mostly in the form of basic sulphate.

During the progress of the work it was observed that during the blow on white metal, where a conductive deposit was obtained in the pipes, practically all of the lead was precipitated from the smoke at a temperature of 340° C., but that the $SO_3$ and $As_2O_3$ escaped. To precipitate the latter and obtain complete smoke clearances a temperature of about 90° C. was necessary. This suggested the possibility of fractional precipitation by using treaters in series, whereby passing gases could be treated at different temperatures, thus classifying the elements of the smoke. This gave an aspect to the possibilities of the process not hitherto considered.

As the result of the recoveries made in the 50,000 cu. ft. installation it was decided to apply to the Executive Committee of the company for authority to erect a plant to treat all the smoke from the converters, this plant to be an adjunct to a new flue system then under consideration, which will later be referred to.

Pending the granting of the required authority it was decided to erect a small treater for experimental work on blast-furnace, MacDougall-roaster, and reverberatory gases.

### *Experimental Plant for Treating Blast-Furnace, MacDougall, and Reverberatory Gases*

The treater, which consisted of 152 5-in. pipes and the necessary electrical equipment, was erected about 300 ft. from the main smelter stack, or approximately 2,700 ft. from the blast furnaces, 1,800 ft. from the MacDougall-roaster installation, and 1,700 ft. from the reverberatory furnaces, where the three flues of these departments run close together and parallel on to the chimney. A 30-in. sheet-iron pipe with the necessary branches and dampers was connected with a fan, whereby the required gases for test could be withdrawn from any flue and delivered to the treater.

The tests conducted on these gases were with the view of not only recovering values, but also eliminating all smoke-forming substances, such as sulphuric acid and moisture. Under such requirements of perfect or high percentages of clearances, lower temperatures and lower velocities than shown for converter-gas treatment for lead values became necessary.

In the tests (Tables I and II) gases were continuously sampled during process of treatment. The apparatus was run in the day time only in

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*Table I.—Tests on Blast-Furnace Gases. Three Furnaces in Operation. Mar. 28 to April 3, 1913*

| Test No. | 1 | 2 | 3 | 4 | 5 | 6 | 7 |
|---|---|---|---|---|---|---|---|
| Length of test, minutes | 500 | 530 | 430 | 540 | 535 | 450 | 500 |
| Temperature of gas entering treater, degrees C. | 95 | 84 | 85 | 84 | 115 | 84 | 90 |
| Average temperature of gas in electrodes, degrees C. | 85 | 70 | 70 | 70 | 90 | 70 | 75 |
| Velocity of gas through electrodes, feet per second | 5 | 3 | 3.3 | 3.4 | 7.2 | 7.1 | 7.3 |
| Total volume treated, cubic feet | 3,100,000 | 1,966,300 | 1,780,200 | 2,257,200 | 4,761,500 | 3,960,000 | 4,550,000 |
| *a* Clearance, per cent. | 98 | Perfect | Perfect | Nearly perfect | 92 | 95 | 93 |
| Total volume of B. F. gas passing to stack, cubic feet per minute | 280,000 | 290,000 | 290,000 | 292,000 | 282,000 | 291,000 | 284,000 |
| Per cent. of this volume treated | 2.48 | 1.54 | 1.74 | 1.73 | 3.68 | 3.5 | 3.65 |

*a* Test No. 1. Clearance, 98 per cent. The solids escaping were mostly moisture and some sulphuric acid.

Tests Nos. 2, 3. Gas was perfectly clear. All dust, sulphuric acid, and moisture collected.

Test No. 4. All dust collected, but very slight amount of sulphuric acid and moisture escaped.

Test No. 5. Clearance, 92 per cent. At the higher temperature and velocity in this test practically all dust was collected, but some acid and moisture escaped.

Tests Nos. 6, 7. Practically all dust collected, but not all acid and moisture on account of higher velocities. Clearance 95 and 93 per cent.

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Table II.—*Test for Nine Days on MacDougall-Roaster Gases*

| Test No. | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 |
|---|---|---|---|---|---|---|---|---|---|
| Date, April, 1913 | 11–12 | 16–17–18 | 21 | 22 | 23 | 24 | 25 | 26 | 27 |
| Length of run, minutes.... | 920 | 1,397 | 500 | 520 | 530 | 520 | 545 | 530 | 300 |
| Temperature of gas entering treater, degrees C.... | 110 | 85 | 85 | 50 | 70 | 70 | 94 | 100 | 84 |
| Average temperature of gas in electrodes, degrees C. | 90 | 70 | 70 | 35 | 55 | 55 | 80 | 85 | 70 |
| Velocity of gas through electrodes, feet per sec. | 7.6 | 7.5 | 5.3 | 3.0 | 3.2 | 3.4 | 8.3 | 8.5 | 8.2 |
| Total volume treated, cubic feet..... | 8,638,800 | 12,992,100 | 3,275,000 | 1,934,400 | 2,098,800 | 2,173,600 | 5,613,500 | 5,565,000 | 3,039,000 |
| *a* Clearance, per cent..... | 96 | Perfect | Nearly perfect | Nearly perfect | Perfect | Perfect | 97 | Nearly perfect | Nearly perfect |
| Total volume roaster gas passing to stack, cubic feet per minute..... | 327,000 | 342,000 | 349,000 | 371,000 | 343,000 | 342,000 | 346,000 | 323,000 | 315,000 |
| Per cent. of this volume treated..... | 3.19 | 3.08 | 2.21 | 1.32 | 1.37 | 1.45 | 3.34 | 3.56 | 3.66 |

*a* Tests Nos. 1 and 7. All dust precipitated, but not all $H_2SO_4$ and moisture.

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order to have Mr. Barker's full attention and thus enable him personally to see that both operating and analytical work would be carried on in a manner leaving no doubt as to the accuracy of the results.

From an inspection of the tables it will be seen that to obtain perfect clearance a velocity of about 3.5 ft. per second in the pipes was necessary. In special experiments with gases diluted with air perfect clearances were obtained with 7 ft. per second in the electrodes. This is equivalent to reducing the dust particles per unit of volume, and such a condition would be obtained by passing gases through a proper settling chamber preparatory to Cottrell treatment.

Several days were devoted to the treatment of mixed gases from the reverberatories, roasters, and blast furnaces, with results equally as good as shown for blast furnaces and roasters separately.

The apparatus worked generally well electrically under most conditions. Different kinds of deposits were obtained, wet and dry, and in some cases wet mud dripped from the pipes, but very little trouble was experienced in making the apparatus work.

### *Summary of Tests at Garfield*

The results of the experiments of December, 1912, to April, 1913, were:

1. The establishment of the pipe electrode ,now being generally adopted in recent Cottrell installations.

2. The replacement of the pubescent electrode by a single wire.

3. Proof that 95 per cent. to practically complete clearance of Garfield smelter gases can be accomplished if temperatures are maintained under 100° C.

4. That by treatment of gases at different temperatures, by treaters in series with the necessary drop in temperatures between treaters, fractional precipitation may be possible. For example, collecting the bulk of the lead in one treater and then by the further cooling of gases collecting most of the arsenic, sulphuric acid, etc., in a second treater.

### *Experiments at Murray Plant, American Smelting & Refining Co.*

The method of fractional precipitation has since been tried experimentally on the treatment of roaster gases at the Murray plant of the American Smelting & Refining Co. Dust, acid, and moisture carried by the gases produced a sticky, muddy deposit on the electrodes when tried in one treater, and the difficulty of removing this mud became a problem. Two small treaters were then placed in series. The temperature of the gases entering the first treater averaged 125° C. and entering the second treater 70° C. In the first as high as 95 per cent. of the total

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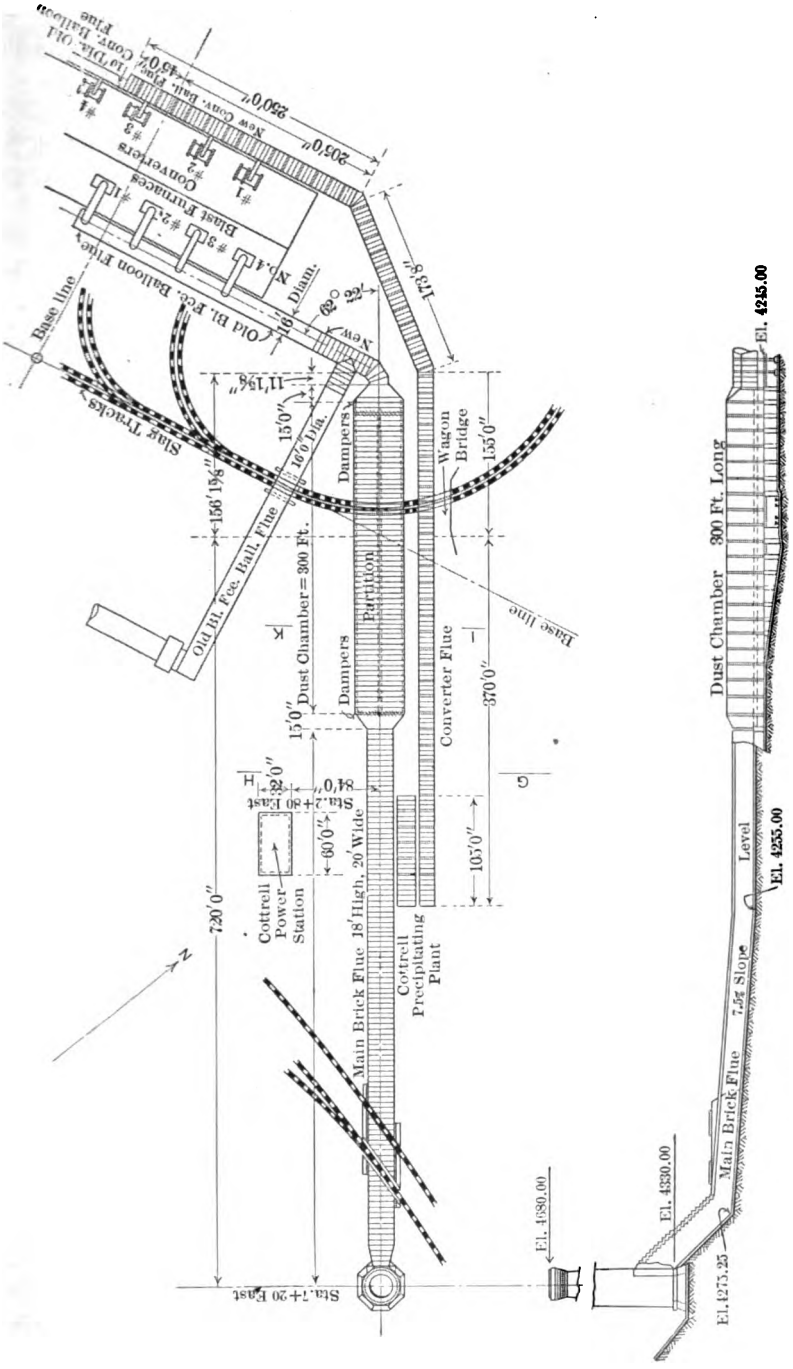


FIG. 5.—CONVERTER AND BLAST-FURNACE FLUE SYSTEM, GARFIELD SMELTER.

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precipitate was collected, in a comparatively dry condition. In the second treater most of the remaining dust, acid, and water, approximating 4 per cent., was collected, sometimes sufficiently moist to drip from the pipes. These results have been encouraging and further experiments will be tried. It is too early to state that this problem has yet been completely solved, as the maintenance of proper temperatures is of critical importance to the work.

I wish to emphasize that the results obtained at Garfield and at Murray were on the treatment of gases particularly favorable to the process and it does not follow that all smelter smoke can be treated as successfully. Again, while experimental results have warranted our trying the process on a large scale, we have no definite assurance that the element of time may not develop difficulties not now apparent.

*Cottrell Plant for Treatment of Gases from Converter Plant*

Fig. 5 is a general plan of a new flue system for the blast-furnace and converter gases, built primarily on account of the inadequacy of

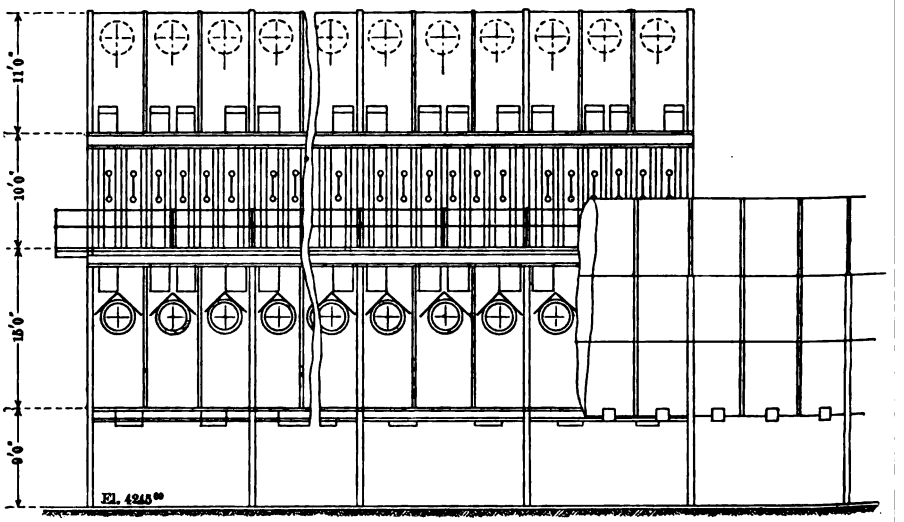


FIG. 6.—LONGITUDINAL ELEVATION OF COTTRELL PRECIPITATING PLANT OF GARFIELD SMELTING CO.

the old flue system to handle properly all the gases of the plant under the present output.

For the collection of dust from the blast furnaces the gases will pass through a dust chamber 300 ft. long by about 920 sq. ft. cross-sectional area, about 100 ft. of which will be filled with suspended wires as followed in the very large chamber of the Boston & Montana Reduction Works at Great Falls with such excellent results.

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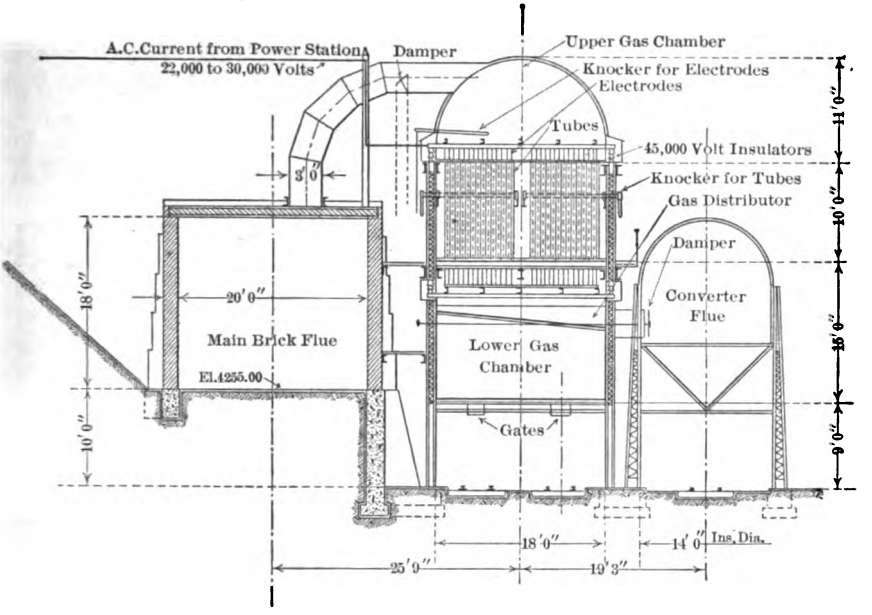


FIG. 7.—CROSS SECTION OF COTTRELL PLANT.

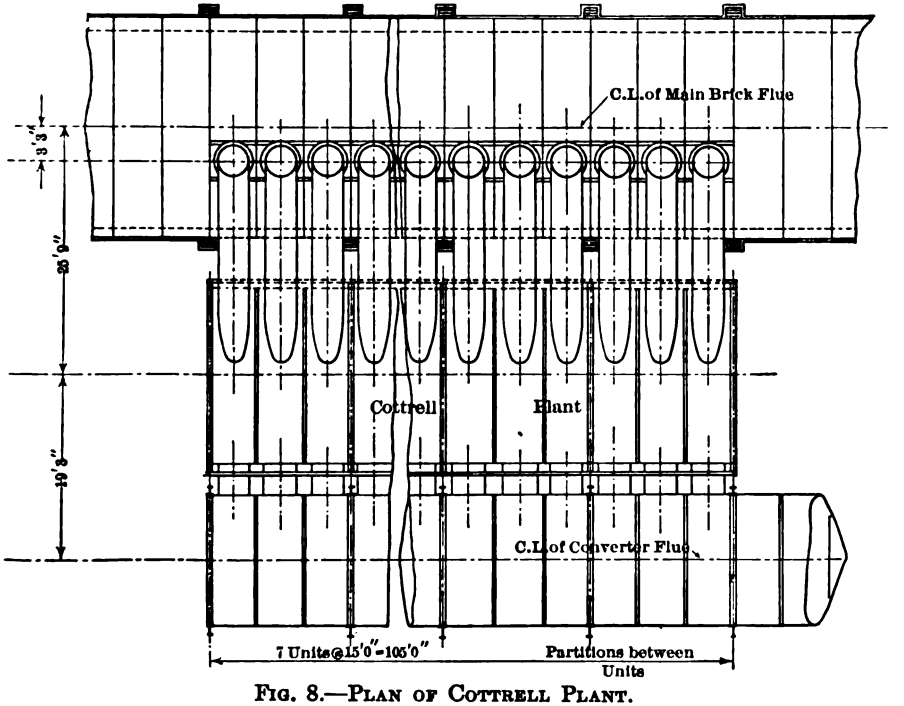


FIG. 8.—PLAN OF COTTRELL PLANT.

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From the chamber the gases pass up a flue connecting with a chimney 350 ft. high by 22 ft. internal diameter at the top, built on a solid rock foundation 85 ft. above the general smelter level, or equivalent to a total height of 435 ft.

The converter gases which are destined for Cottrell treatment for recovery of lead fumes, etc., will pass through a long steel flue of 210 sq.

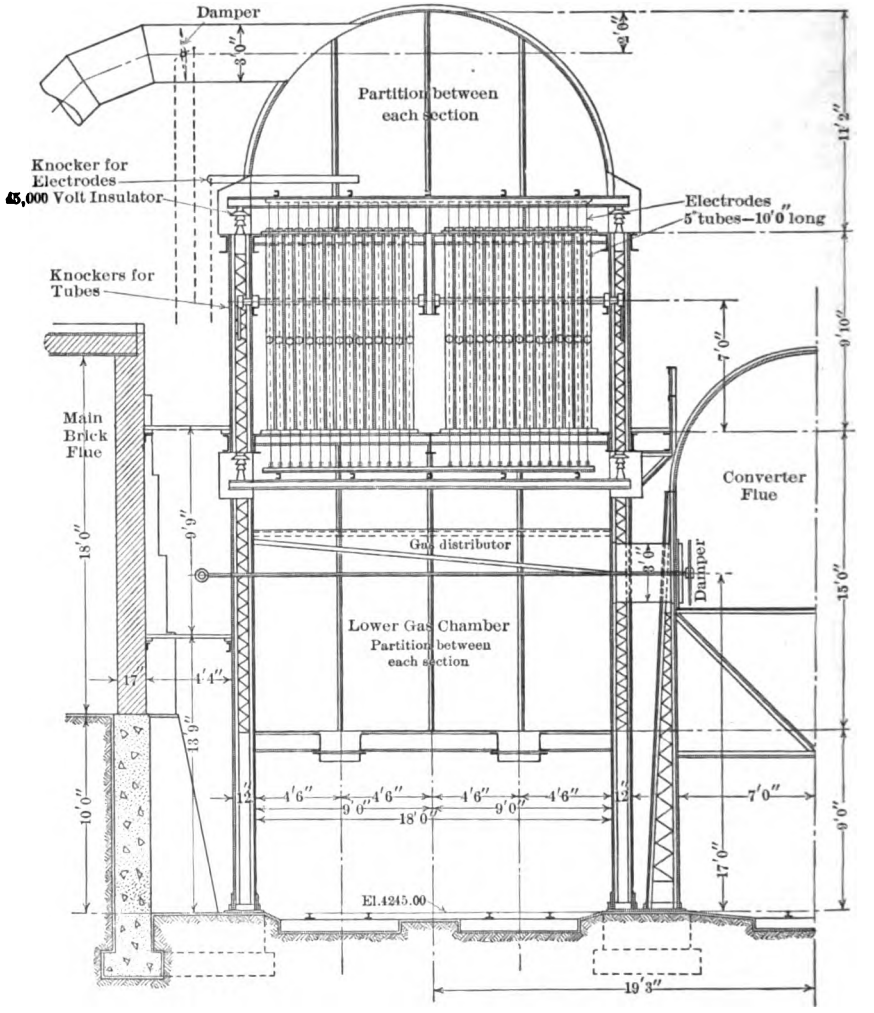


FIG. 9.—CROSS-SECTION OF 360-TUBE UNIT.

ft. cross-sectional area and about 1,000 ft. long, which will collect practically all cuprous material from the smoke, and the gases in their passage will become sufficiently cooled for delivery to the electrical precipitators, as illustrated in Figs. 6 to 10.

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*Electrical Precipitation.*—The electrical precipitator consists of seven units, containing 360 5-in. pipes, 10 ft. long, per unit, or a total of 2,520 for the whole installation. Six units will be usually operated while one is being cleaned.

On the basis of seven operating, with an output of 250,000 cu. ft. of gas a velocity of 12 ft. per second will be obtained; on the basis of

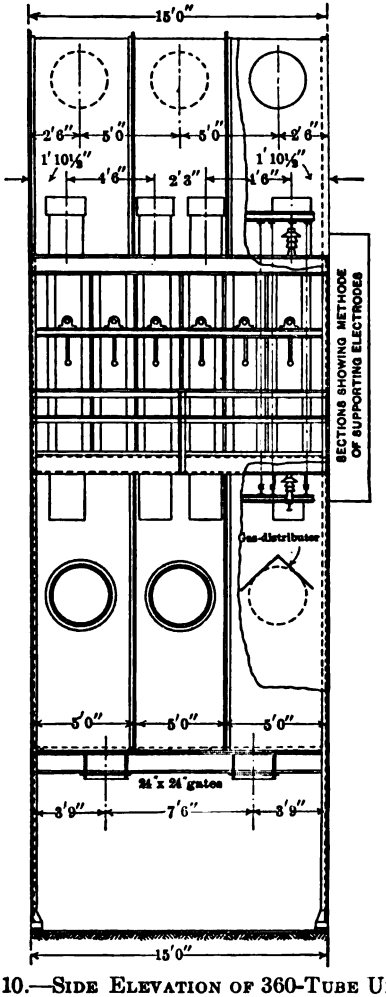


FIG. 10.—SIDE ELEVATION OF 360-TUBE UNIT.

200,000 cu. ft. a velocity of 9.6 ft. per second, and with six units on the same volume 11.2 ft. per second, all within the required velocity for the special treatment of converter smoke. Certain changes contemplated in converter hoods will further reduce the volume of gas delivered to the treater.

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Gases from the converter flue will be delivered to the lower chambers of treater through 36-in. pipes, three pipes per unit, each pipe having a deflector extending across to insure the best possible distribution. After passing through the pipes the gases reaching the upper chamber then pass through 36-in. pipes to the main flue leading to the chimney, there being three pipes per unit, and again tending to the maintenance of an even flow through the electrodes.

*Transformer and Rectifier House.*—With the apparatus used in the single converter gas plant of 608 pipes referred to it was estimated that the maximum power used was 25 h.p. In determining the size of transformers necessary for the new installation this figure was used as a basis. As the units consist of 360 pipes each and as it may be necessary at times to connect two on the same transformer, it was decided that 20-kw. transformers would have liberal capacity for any condition that might arise. The power necessary for the treatment of 250,000 cu. ft. of gas per minute will probably amount to 100 h.p.

*Electrical Apparatus.*—This includes seven complete motor-generator-rectifier sets and transformers, and switchboard panel to control these.

A set consists of a 30-h.p., 250-volt, direct-current motor, direct connected to a 20-kw., 4-pole, 60-cycle, 220-volt single-phase alternating-current generator. The 220-volt current is stepped up to 20,000 to 30,000 volts by means of a 20-kw. transformer and then led back to the rectifier, which is on the same shaft with the motor-generator. Here it is rectified before going to the treater. The voltage before going to the transformer may be regulated to any desired value by varying the field excitation with a rheostat located at the switchboard.

On the switchboard are located the necessary switches and instruments for the motor-generator sets. Each generator circuit is supplied with an automatic circuit-breaker switch which can be set to carry a short circuit for a definite length of time before opening.

It is expected that the plant will be completed some time in July, and I believe it should accomplish what it has been designed for; that is, the collection of those elements in the smoke which may be of commercial value.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Dorr Hydrometallurgical Apparatus

BY JOHN VAN N. DORR, DENVER, COLO.

(Salt Lake Meeting, August, 1914)

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## Introduction

It is 10 years this summer since the first of the contributions which it has been my privilege to make to the working tools of the hydrometallurgist was set at work, but a full description of what has come to be known as "Dorr Machinery" has never been published in the *Transactions* of the Institute, and it has been suggested to me that an account of the same would prove of interest to members, although most of those engaged in cyaniding and concentration are probably familiar with it. The account which I shall give of the development of the machines must be recognized as being written from the viewpoint mainly of my own local experience and not from that of one who was familiar with the general metallurgical practice throughout the world. Most advances in any art are conceived before the time is ripe for their development and any invention which comes into general use will usually be found to have been tried out in a crude way and abandoned more than once. If the early experimenter went as far as the Patent Office, his work may serve no other purpose than to prevent the issuance of the broadest patent to those who later may develop a commercial process.

The cyanide process in the United States early in 1904 was mainly confined to the leaching of dry-crushed ores, both coarse and fine, and sand treatment of tailings from concentration and amalgamation. In the Black hills some plants were crushing in cyanide solution, leaching sands and handling the slime by the well-known decantation process.

Filter pressing, in use for some years in Australia, had been tried in several places in this country and the Moore process had been abandoned at Mercur after a short run, but was doing good work at the mill of Lundberg, Dorr & Wilson, at Terry, and being installed at Bodie and the Liberty Bell.

A general feeling, however, still existed that slimes were to be avoided when possible, on account of the losses in treatment and the difficulty of making a good leaching product when they were present in large amounts, especially when the crushing was done in cyanide solution.

## The Dorr Classifier

### *History*

The object of the first classification done in cyaniding work was to produce a leachable sand and overflow as small an amount of sand as possible with the slime, which was usually run to waste.

In the early days various arrangements of adjustable tank gates were used, and sometimes direct overflow of tanks filled with a rotary distributor sufficed. Later, cone classification, of which the Merrill system, as

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introduced at the Homestake, was probably the most perfect, was adopted. Crushing in cyanide solution, however, first used successfully in the United States by J. M. Henton about 1901, added materially to the difficulty of cone work, especially in small plants on soft and variable ores, and in the Black hills at least caused a search for other methods.

An early attempt to solve the problem that was considered very successful at the time consisted of a box about 3 ft. cube, situated over each leaching tank and equipped with a hinged bottom. The stream of pulp was fed into this box and the solution and slime overflowed until the box was full, when the mill man released the bottom and the sand fell with a splash into the vat below, while another charge was collected.

I am told that this method was used in Rhodesia years ago and brought good profit to the inventor. It might be regarded as a poor apology for the double-treatment tanks in use in South Africa and in the United States.

A description of a small plant in the South Atlantic States, which appeared in one of the mining weeklies in 1905, shows that the problem was not considered satisfactorily solved at that date. The writer somewhat proudly told how he had overcome the difficulty of securing clean sand by flowing his pulp into mortar boxes alternately and shoveling the settled sand therefrom into his leaching vats.

These boxes were later abandoned for double cone classification of the sand, which worked well on quartzite ore but gave great trouble on soft material, especially in small plants with irregular feed.

The cone system was installed in the 100-ton Lundberg, Dorr & Wilson mill above mentioned, but on our ore, which varied from a soft, clayey shale to hard quartzite, it was impossible even with a man constantly watching the cones to get a good leachable sand and avoid overflowing sand with the slime. This resulted in heavy losses through bad leaching of the sands and great difficulties in operating the Moore process on the slime.

I had felt for some time that a mechanical method of separating sand, that would not require constant attention, was feasible, and realizing that the success of the property, which was operating on a close margin, depended on good classification, proceeded to attack the problem. I knew of the work of George Moore in washing slime from the roasted fine ores at Mercur with a spiral lined cylinder and later with a screw conveyor in an inclined trough, but both methods were somewhat discredited by the abandonment of the Moore process itself at the Mercur plant.

A description by J. E. Johnson, Jr., in the *Transactions* of the Institute, of a sand shovel he was using to remove sand from a tailings launder at an iron-concentrating mill gave me confidence that I was on the right track. My aim was a settling box with uniform cross-section for settling, from which the settled material could be removed with a minimum amount of

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slime-bearing solution without disturbance of the surface near the overflow point.

I also felt that it was necessary to have no submerged parts moving on each other to cause undue wear.

A mechanical rake that would carry the sand up an inclined trough and discharge it above the water level occurred to me as able to fill these requirements and yet give ample space for quick settling at the surface while allowing the maintenance of an agitated zone at the bottom to prevent the slime settling with the sand.

After considerable investigation the motion desired was secured by means of suspending the rakes from cross bars carrying rollers traveling on suitable tracks, with switches, as shown in Fig. 1. The rollers passed under the switches on the up stroke, but were lifted on the down stroke and thus raised the rakes over the sand which had been moved forward.

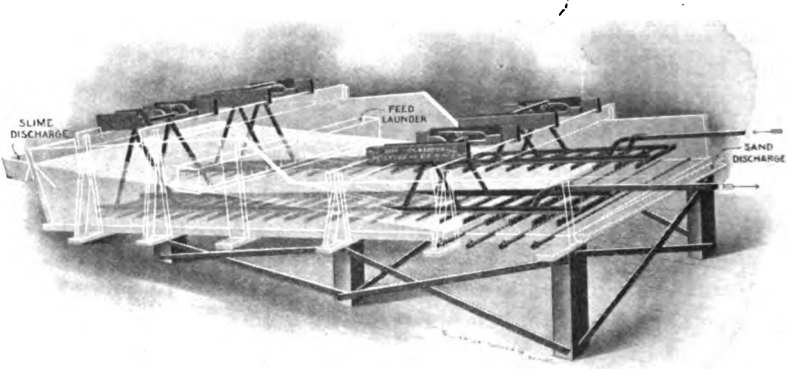


FIG. 1.—THE DORR CLASSIFIER (PATENTED). ORIGINAL DESIGN.

The installation of the machine thus designed made such an improvement in both sand and slime products that our extraction was greatly increased and lost time on account of classification was eliminated. I may say, as a matter of personal interest, that it made the difference between the failure and success of a mill that had a subsequent useful life of nine years and treated profitably ore from which not over $2.50 per ton was recovered.

Mechanical sand separation is a commonplace thing to-day in cyaniding and may have been used then in concentration and other work. It is of interest, however, to note that one of the best-known engineers in this country told me in 1906 when he saw the machine working, on his return from South Africa, that if he had seen the blueprints of it he would not have believed it possible that it could give the products it was then making.

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*Description*

The Dorr classifier has been improved from time to time to meet new conditions and as now manufactured is shown in Fig. 2, which illustrates the standard duplex machine.

It consists of a settling box, in the form of an inclined trough with the upper end open, in which are placed mechanically operated rakes or scrapers for the purpose of removing the quick-settling material from the open end. Each rake is carried by two hangers, one at the sand-discharge end suspended from an arm attachment to a rocker arm or lever, which terminates in a roller. The other depends from a bell crank connected by a rod to the same rocker. The roller is pressed against a cam on the crank shaft.

The rakes are lifted and lowered at opposite ends of the stroke by the

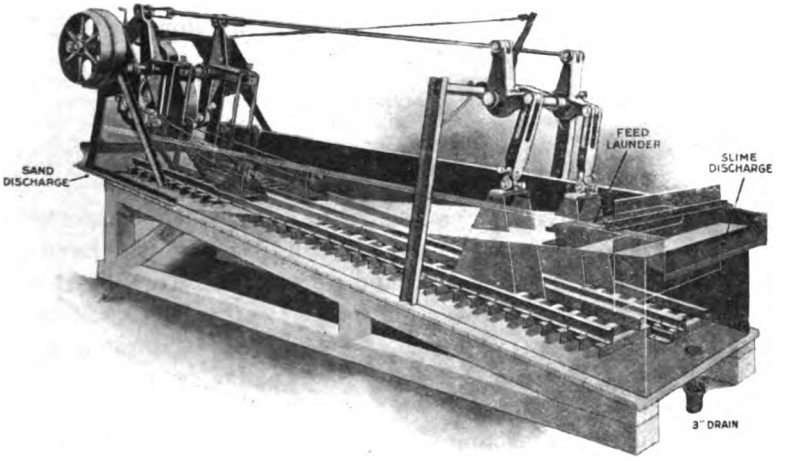


FIG. 2.—THE DORR CLASSIFIER (PATENTED). MODEL "C."

action of the cams transmitted through the rocker arms and bell cranks, and the horizontal motion is obtained directly from the crank.

The bell cranks at the slime end are carried by a second larger bell crank held in position by a chain attached to a spool on a worm gear at the head end of the classifier. By this means the rakes can be raised 10 in. at the lower end and operated in that position or any intermediate one. This allows the classifier to be started readily when nearly filled with sand after an unexpected shutdown, and the regulation of the depth of the settling box when in operation to vary the products being made.

The pulp is fed across the settling box as shown and a uniform flow to the lip at the end is maintained while the sand settles to the bottom and is advanced by the rakes until it emerges from the liquid and is discharged with from 20 to 30 per cent. moisture.

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The agitation near the bottom of the tank, caused by the reciprocating motion of the rakes, assists in keeping the slime in suspension, but is not normally sufficient to cause fine sand to overflow.

### *Regulation of Products*

The machine is intended of course to make only two products, and the point of separation can be varied by the following means:

1. The use of the baffle shown, which allows a reduction of the cross-section of flow so as to overflow more fine sand.
2. Raising the rakes and so operating them in a shallower tank.
3. Increasing the speed of the rakes until the agitation keeps fine sand in suspension.
4. The attachment of perforated spray pipes to the rakes at a point where they remove the sand from the liquid, allowing a rewashing of the sand and removal of any slime that may be carried down with it.

### *Cost of Operation*

As with any other machine the costs of operation will depend on the work done and care taken by the mill men.

The only place where wear caused by the sand would be expected is on the bottom edge of the scrapers, and careful measurements have shown that in more than one case four years' constant service did not remove more than $\frac{1}{8}$ in. at this point. This rather surprising result must be due to the scrapers being suspended and the lower edge pushing grains of sand ahead so as to prevent the bottom coming into actual contact with the sand bed.

It is not unusual for machines to run for a year or more without repairs of any sort being needed, and I know of one classifier which ran for nearly four years with repair costs under $5 and a careful examination at the end of the period showed it to be in good condition. I believe an allowance of 0.05c. per ton treated for repairs will usually be ample.

### *Power*

The power required will depend on the load, but tests have shown that $\frac{1}{4}$ h.p. is generally sufficient.

### *Capacity*

The capacity of the machine depends, as would be expected, on the nature and dilution of the pulp fed to it and the point of separation desired. The table below showing results at different plants will give the best idea of it.

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*Operating Data on Dorr Classifiers*

| Mill | Tons per 24 Hr. | Feed | | | | Sand | | | Slime | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Dilution | +100 | +200 | −200 | +100 | +200 | −200 | +100 | +200 | −200 |
| Portland, Colorado | 145[a] | 5 to 1 | 38.3 | 13.4 | 47.1 | 75.3 | 16.2 | 7.4 | 0.9 | 5.8 | 94.7 |
| Golden Cycle, Colorado | 151[a] | 3.5 to 1 | 45.0 | 19.0 | 36.0 | 68.0 | 23.0 | 9.0 | ..... | 11.0 | 89.0 |
| Cia Benef. de Pachuca, Mexico. | 144[b] | 4 to 1 | 42.7 | 25.1 | 32.5 | 43.6 | 40.1 | 16.8 | 0.1 | 3.8 | 96.8 |
| Tonopah Extension, Nevada.. | 73[c] | 6 to 1 | 65.0 | 11.1 | 23.9 | 70.0 | 18.3 | 11.7 | 0.6 | 8.5 | 90.9 |
| Amparo Mining Co., Mexico | 125[a] | 7.5 to 1 | 52.4 | 18.5 | 29.1 | 71.4 | 27.4 | 1.2 | ..... | 6.5 | 93.5 |
| Tonopah-Liberty, Nevada.... | 85[c] | 7 to 1 | 30.5 | 41.9 | 26.3 | 75.7 | 24.0 | 0.1 | 2.5 | 14.3 | 81.4 |
| Goldfield Consolidated, Nevada. | 264[d][b] | 3 to 1 | 40.0 | 12.0 | 48.0 | 74.0 | 19.0 | 7.0 | 2.6 | 19.5 | 79.1 |
| Tonopah-Belmont, Nevada... | 70[e] | ..... | 65.6 | 2.7 | 28.9 | ..... | ..... | ..... | 0.4 | 6.6 | 93.0 |
| Alaska-Treadwell, Alaska[e].... | 89[c] | ..... | 48.7 | 41.5 | 9.8 | 48.7 | 41.5 | 9.8 | ..... | 2.0[e] | 98.0 |
| Nipissing, Canada | ..... | ..... | 78.7 | 3.6 | 16.8 | 85.8 | 7.2 | 6.9 | ..... | 0.5 | 99.5[f] |

*a* New ore.

*b* Includes tube-mill return. Under normal conditions, with a tube mill in closed circuit with a Dorr classifier, 35 to 50 per cent. of original feed returned.

*c* New ore; tube-mill product returned, but amount not stated.

*d* 168 tons come from the stamps, the rest being returned from the tube mill.

*e* Classifiers used in connection with the regrinding of concentrates; rakes operated 24 strokes per minute; 100 per cent. of concentrates pass 200 mesh; the 2 per cent. is silica from tube-mill pebbles.

*f* Of this 200 mesh product 15 to 20 per cent. is sand; crushing is done in two sets of tube mills; the sand from the first classifiers being reground in closed circuit with tube mills and other classifiers, the combined slime overflows giving the screen test shown.

## *Uses*

The Dorr classifier was originally designed for the separation of clear slime and a leachable sand, but with the adoption of so-called all-sliming

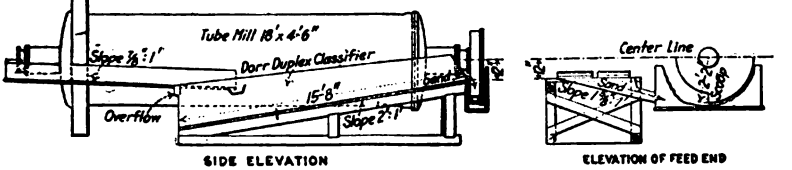


FIG. 3.—ARRANGEMENT OF DORR CLASSIFIERS AND TUBE MILLS OPERATING IN CLOSED CIRCUIT IN THE TONOPAH-BELMONT MILL.

treatment in cyaniding its value in connection with regrinding in tube mills was apparent to all and it came rapidly into use for that purpose also. The advantage of operating a classifier and tube mill in closed circuit with no other means of elevation of the return is appreciated by all practical men. Fig. 3 shows the arrangement of tube mills and classifiers as installed at the Tonopah-Belmont mill.

The use of a Dorr classifier with two or more compartments for washing the sands was early suggested and a double-washing classifier for

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giving sands an acid bath was put in use at the Morning mill with tube flotation several years ago.

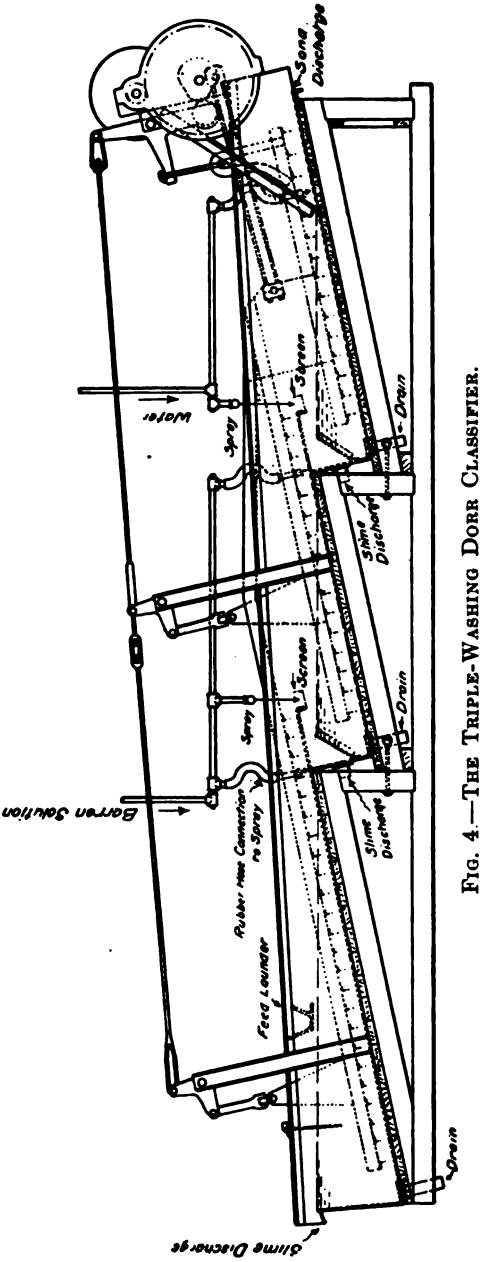


FIG. 4.—THE TRIPLE-WASHING DORR CLASSIFIER.

We have carried the idea further and made several triple-washing classifiers for use when sands are to be discharged from gold-bearing cyan-

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ide solutions without leaching and must be freed from cyanide and dissolved gold.

Fig. 4 shows the triple-washing Dorr classifier with pipe connections arranged so as to wash the sands as they come from the first two compartments with barren solution, and from the last with water.

One of these machines following a duplex Dorr classifier has been installed recently at the Hollinger mill to wash the concentrates after they have been reground and cyanided in a tube mill.

The high-grade solution overflowing from the first machine assays $65 in gold per ton and the concentrates are finally discharged with 20 per cent. moisture which assays 16c. per ton; thus giving a dissolved loss per ton of concentrate of only 4c. About 30 tons per day are thus treated.

In 1910 we furnished a series of classifiers to a Pennsylvania company for the purpose of leaching and washing, by the counter-current method, cupreous pyritic residues that had been given a chloridizing roast, but I understand that other matters interfered and their use for that purpose was never thoroughly tested.

In 1913, Captain Wolvin, of the Butte-Duluth Co., conceived the idea of acid leaching and washing oxidized copper ores in Dorr classifiers from watching them at the Butte Superior mill, and at his request we furnished him with five classifiers for trial and later two more for the same series with all parts subject to contact with the acid made of hard wood. These machines proved most successful, increasing the extraction over the former leaching in vats at coarser mesh at least 20 per cent., and as a result the Butte-Duluth Co has added to its equipment five heavy Dorr classifiers 8 ft. wide by 30 ft. long which I understand will be operated in series and have an estimated capacity of 400 tons.

I believe the results of the excellent work done by Captain Wolvin and Mr. Sherwood will be embodied in a paper to be presented to the Institute.

Dorr classifiers have been successful as dewaterers in handling magnetic iron and sulphide concentrates, and are discharging the former with only 12 per cent. moisture.

## The Dorr Thickener

### *History*

In the summer of 1906 I was engaged by the Mogul Mining Co. to change the Kildonan mill at Pluma, S. D., from a 100-ton dry cyanide plant into a 300-ton wet-crushing mill following the practice at the Lundberg, Dorr & Wilson mill previously mentioned.

The settling of slime in cyanide work in the United States at that time

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was an intermittent process usually carried out in flat-bottom tanks, but a few engineers had followed the practice introduced, I believe, by John Randall, of building very large cones and operating with a continuous overflow and intermittent or continuous underflow. We had used a 22-ft. and a 20-ft. cone at the Lundberg, Dorr & Wilson mill and appreciated greatly the advantage of continuous thickening, so desired to use the same method at the other plant. The Kildonan had been started as a chlorination mill in 1894, but in 1902 the property had changed hands and as cyanide was proving successful on Black Hills ores the chlorination barrels were scrapped and replaced by some leaching tanks to treat a 20-mesh product. A little later the company was induced to install an aerating leaching process in which the bottoms of the leaching tanks were covered with perforated pipes and sand was to be leached and agitated at the same time. As might be expected, it was not a success, but added several 35 by 5 ft. tanks to the equipment.

When I undertook to remodel the mill, as before mentioned, there was no room in it to place the number of large cones that would have been required, and, as I had experienced the difficulty of preventing the building up of solids on a 60° slope when making a thick underflow, I sought some other means of thickening continuously. I conceived the idea that it might be possible to operate a mechanism or scraper in a tank at such a speed that it would not disturb the upper part of the tank and interfere with settling and yet would prevent the slime from becoming solid and allow it to be discharged at one point. It was recognized that if the slime did become solid no mechanism could be built strong enough to move it. The design of the machine presented a problem, in the solution of which I was greatly assisted by L. B. Eames, now superintendent of the Goldfield Consolidated mill, who was an assistant on construction. We decided to utilize one of the 35 by 5 ft. tanks mentioned, after deepening it to 12 ft. Realizing the immense leverage of scraper arms on any shaft, we considered a track on the periphery with a rotating frame traveling on it from which would depend the rake arms, but finally decided to try the central drive. It was recognized here as in the classifier that stuffing boxes and wearing parts in the pulp should be avoided, and the present arrangement, which seemed the simplest, was adopted.

The use of a 35-ft. tank for the first test was certainly experimenting on a large scale, especially as the first trial thickener represented the only equipment provided in the plant for that purpose. I was prepared, if it had not worked, to use the tank as a settler in another way, which I felt would give as good results as were obtained in practice elsewhere.

We had done considerable good work with diaphragm pumps at Terry, and had found them capable of pumping anything up to small monkey wrenches. I had planned, therefore, to place one or more above the tank and provide each with a number of suction pipes, reaching to the bottom

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and so spaced that the tank would become virtually a collection of small cones to be pumped from in rotation.

The thickener started off, however, with no trouble and we found that the first speed selected, 1 rev. in 20 min., was satisfactory. It was operated until the plant burned in 1912.

Continuous thickening with Dorr thickeners is accepted nearly everywhere now, but its possibilities were not recognized at first. When the first thickener had been operating at the Mogul for several months, a Colorado engineer, manager of one of the best operated companies in that State, happened to see it. After watching it for some time he was so confident it could not be used for continuous work on his extremely light colloidal slime that he waited nearly a year before trying it. When finally installed it showed an apparent saving of over $70 per day in increased recovery and diminished costs.

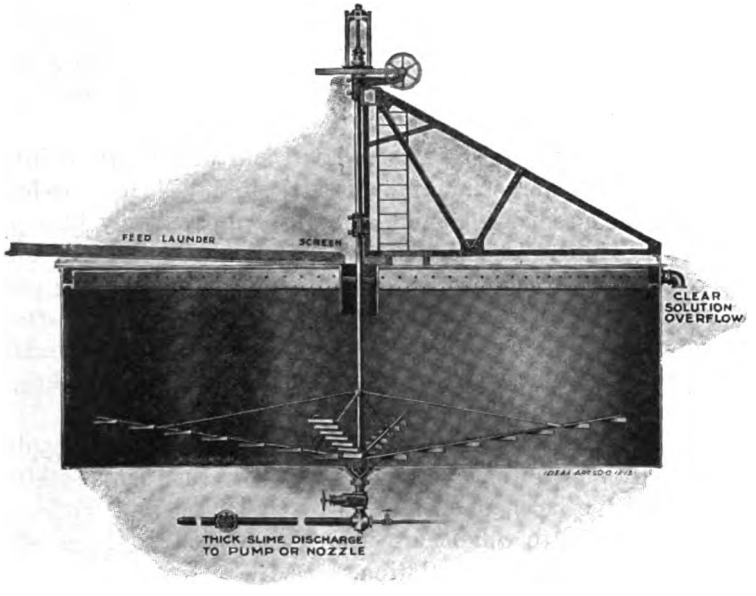


FIG. 5.—THE DORR CONTINUOUS THICKENER (PATENTED).

When we came to manufacture the machines commercially many improvements were made, the most important being the addition of an overload alarm to give warning when some part of the mechanism was in danger of being strained by the resistance offered by the accumulation of thickened material in the tank.

*Description*

The Dorr continuous thickener consists of a slow-moving mechanism placed in a suitable tank, by means of which the operation of settling may

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be made continuous through the removal of the settled material to a point of discharge and the prevention of its accumulation as a solid in the tank.

As usually furnished, it consists of a central vertical shaft with radial arms equipped to bring the thickened material to a discharge opening at the center by the slow rotation of the mechanism. The thick material may be discharged by gravity at this point into a launder, or piped to the side of the tank and raised by air lift or pump to the level of the overflow or higher.

The machine is arranged for raising the shaft so that the arms will not be imbedded in the thick material if the power should be shut off for any length of time. The shaft can be lowered again gradually while running. Shaft and gear bracket are supported by a bridge over the tank or suspended from the roof trusses (Fig. 5).

In some cases, notably at the Liberty Bell mill, thickeners have been installed driven from below the tank through a mercury bearing, and proved efficient, although they cannot be raised.

The thin pulp is delivered near the center of the tank in a suitable well with a float to cause minimum disturbance, and the overflow is taken off by a peripheral launder.

The thickened pulp can be accumulated and withdrawn at intervals or a continuous discharge maintained as desired. Nozzle discharge is in use in some concentrating mills where a comparatively thin pulp is desired, and also in one case where a product of only 30 per cent. moisture is being obtained. Many are using diaphragm pumps for this purpose. They have the advantage of ready regulation and require little attention. Having a positive displacement they tend to regulate automatically the amount of solids withdrawn, for, when the pulp becomes thicker, more solids are pumped with each stroke.

The diaphragms have a life of from two to eight weeks, depending on operating conditions, so that renewal expenses are very small. Although arrangements have been made to change the speed and stroke of the pumps, a pet cock to admit air into the suction has proved most satisfactory as a means of regulation.

### *Speed*

The thickener has been operated at speeds ranging from 1 rev. in 2 min. to 1 in 40 min. A quick-settling sandy material will offer great resistance unless a comparatively high speed is maintained, while colloidal slime will give a slightly thicker underflow at a very slow speed. It will be found when handling sandy material that if the sand accumulates so that it is being moved around the tank by the channel arms as well as being advanced toward the center by the plows the resistance increases rapidly and the speed should be increased.

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*Power*

This varies with the size of the tank and the nature of the feed. Of course, the motor input is much larger than the actual power consumed, owing to the low load factor commonly used, as it is essential to have power enough to meet an unusual strain. Spring measurements have shown approximately $\frac{1}{20}$ h.p. being transmitted to the worm shaft of a 44-ft. thickener handling a classified slime. It would not be advisable, however, to install less than a 1-h.p. motor on a single machine, but $\frac{1}{2}$ h.p. each can be allowed when several are driven from one lineshaft.

*Repairs*

Normal operation of the Dorr thickener causes no wear except on the worm, and many machines are running to-day that have not cost a cent for several years. On the other hand, if started after a shutdown without

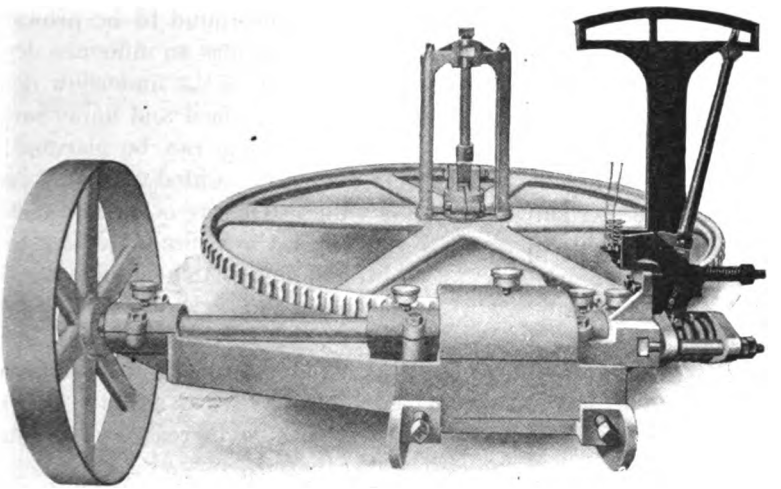

FIG. 6.—OVERLOAD ALARM AND RESISTANCE INDICATOR ON DORR THICKENER, WITH ARRANGEMENT FOR RAISING SHAFT WHILE OPERATING.

raising, a strain of any amount may be given so that the "weakest point," which I believe exists in everything but the "One Hoss Shay" of Oliver Wendell Holmes, will have to yield. We have attempted to avoid breakage by furnishing a pulley that cannot transmit enough belt pull to do any damage, but the average mill man is likely to use a crowbar to start something without realizing that he can thus apply more force than the belt.

The overload alarm, mentioned before and shown in Fig. 6, is arranged to indicate the resistance offered by the mechanism as shown by the thrust

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on the worm shaft and to ring an alarm when the load becomes excessive. A solenoid or other means can be used to correct automatically the condition causing the excessive load, by reducing the feed or increasing the underflow. The alarm has proved very valuable, especially on quick-settling pulp, when it is desired to obtain the thickest discharge.

### *Labor*

The attendance required varies with the regularity of conditions maintained and is usually confined to lubrication once a shift, so that the care of the thickeners is included in the duties of some man employed principally on other work. At the Anaconda four men per shift are more than enough to care for 160 tanks thickening 26,000,000 gal. per day containing 2,500 tons of solids.

### *Capacity*

The capacity of Dorr thickeners has been found to be primarily a function of area, although the depth of the tank has an influence depending on the dilution of the feed and the dilution of the underflow desired. With a given area and depth and a very dilute feed and underflow, the capacity depends on the amount of liquid that can be clarified; *i.e.*, additional solids, but no additional liquid, can be added to a tank already fed to capacity without overflowing slime. On the other hand, with a feed perhaps 8 of liquid to 1 of solids and a thick discharge of 2 to 1 or less, it will be found usually that additional liquid can be added to a thickener operating at capacity without overloading it, but any addition of solids will cause slime to overflow.

If a plant requires more settling capacity, raising the temperature of the solutions may prove an economical way to add 10 to 20 per cent. The capacity of any filter will also be increased and a higher extraction may pay for the cost of heating.

The table following represents data given me from time to time. It shows the settling area in use per ton in different mills which in many cases were not feeding their thickeners at capacity.

### *Density of Underflow*

This depends on the nature of the pulp to be settled and the size of the particles. An argillaceous pulp, such as that at the Liberty Bell in Colorado, although containing a large percentage of reground siliceous material will not settle thicker than 60 per cent. moisture, while a finely ground quartz will give as low as 27 per cent. moisture. At the Porcupine-Crown plant, handling a quartz product of 75 per cent. −200 mesh, the

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average final pulp discharged contained 30 per cent. moisture when the feed was at the rate of 1 ton of solids per day for each 4.7 sq. ft. of tank area.

*Operating Data on Dorr Thickeners*

| Mill | Sq. Ft. Settling Area per Ton of Solids Thickened per 24 Hr. | Sq. Ft. Settling Area per Gallon Overflowed per Minute | Remarks |
|---|---|---|---|
| San Rafael, Mexico | 4.5 | ..... | Tube-mill product, 75 per cent. − 200 mesh, discharge 45.5 per cent. solids. |
| Liberty Bell, Colorado | 15.0 | 12.6 | Tube-mill product, much light argillaceous slime. Discharge 33 per cent. solids: + 100, 17 per cent.; + 200, 13 per cent.; − 200, 70 per cent. Feed 9 : 1. Solution fed at capacity; solids not. Large area per gallon overflowed per minute due to density of underflow and nature of the slime. |
| Mogul, South Dakota | 3.92 | ..... | Tube-mill product; ore siliceous: + 60, 0.6 per cent.; + 100, 7.8 per cent.; + 200, 26 per cent.; − 200, 65.6 per cent. Discharge 56 to 59 per cent. solids. Continuous decantation. |
| Batopilas, Mexico | 0.6 to 0.9 | ..... | 40-mesh product; 90 per cent. passing 100 mesh. |
| Zambona, Mexico | 3.1 | ..... | Tube-mill product. Discharge 40 per cent. solids. |
| Dominion, Ontario | 5.4 | ..... | Tube-mill product, 88 per cent − 200 mesh, ore diabase. Discharge 40 per cent. solids. Feed 6 : 1. |
| Porcupine-Crown, Ontario. | 4.25 | ..... | Tube-mill product, 75 per cent. − 200 mesh. Discharge 65 per cent. solids. Quartz ore. Continuous decantation. With 5.1 sq. ft. settling area per ton settles to 71 to 73 per cent. solids. |
| El Palmarito, Mexico | 4.5 | ..... | Tube-mill product: pure quartzite, 97 per cent. − 200 mesh. Feed 7 : 1. Discharge 65 to 70 per cent. solids. Continuous decantation. |
| Amparo, Jalisco, Mex. | 4.9 | 1.4 | Tube-mill product, siliceous: 93.5 per cent. − 200 mesh. Feed 24.5 : 1. Discharge 23.5 per cent. solids: used to feed vanners. |
| Veta Colorado, Parral, Mex. | 5.0 | 3¼[a] | Tube-mill product, rather argillaceous: 71 per cent. − 200 mesh. Feed 11 : 1. Discharge 33 per cent. solids for agitator. Have settled to 65 per cent. solids. |
| Smuggler-Union, Telluride, Colo. | ..... | ..... | Very clayey slime with classified sand. Screen test: + 40, 1.48 per cent.; + 60, 7.27 per cent.; + 100, 14.81 per cent.; + 200, 11.63 per cent.; − 200, 65.81 per cent. |
| | 30.0 | 26.0 | Settling from cold water, slightly alkaline. Feed 8 : 1. Discharge 50 per cent. solids, 1.429 sp. gr. |
| | 10.0 | ..... | Settling from cyanide solution. Feed, 2.5 : 1. Discharge 40 per cent. solids, 1.316 sp. gr. |
| A large copper company, Arizona. | 11.6 | 8.11 | Considerable argillaceous slime. Feed 10.4 per cent. solids. Discharge 25.3 per cent. solids. |
| Pennsylvania Steel, Lebanon, Pa. | 14.2 | 2.48 | Thickening ahead of vanner concentration. Feed 2.8 per cent. solids. Discharge 10.6 per cent. solids. Overflow 0.4 per cent. solids, extremely fine, which does not interfere with using water again. |

[a] Not up to capacity of overflow.

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*Operating Data on Dorr Thickeners (Continued)*

| Mill | Sq. Ft. Settling Area per Ton of Solids Thickened per 24 Hr. | Sq. Ft. Settling Area per Gallon Overflowed per Minute | Remarks |
|---|---|---|---|
| Nevada Consolidated, Ely, Nev. | ..... | 1.25 | "Each 17-ft. thickener supplies wash water for 20 Wilfley tables and occasionally for wash on vanners. One thickener has a greater capacity than twelve 8-ft cones." Area of 17-ft. tank is 226 sq. ft.; of the twelve 8-ft. cones, 525 sq. ft. |
| Broken Hill, Proprietory, Australia. | ..... | 1.80 | Dewatering slime from lead-zinc concentration mill. Feed 100 : 1. Discharge 55 per cent. solids. |
| Anaconda Copper, Mont. | ..... | 5.95 | Dewatering slime from concentrator. Forty 4-deck thickeners, each 28 ft. in diameter by 3 ft. 3 in. deep, handle about 26,000,000 gal. of pulp per day which contains approximately 2 per cent. solids. A clear overflow obtained, the underflow containing about 15 per cent. solids, which is fed to buddles. |

The data given here show that when pulp is carried in cyanide solution a provision of 5 to 6 sq. ft. per ton for a siliceous tube mill product is ample and from 7 to 15 sq. ft. for a clayey material or classified slime product. When very dilute products are handled the area required is determined usually by the gallons per minute to be overflowed.

Reground pulp, when the original material is not homogeneous, such as an ore containing quartz and some argillaceous material, may be found to have what D. L. H. Forbes has well called a "critical density." If we attempt to thicken it below this density a segregation may occur and the more siliceous portion of the pulp be discharged while an accumulation of fine slime takes place until it overflows, even though the feed is greatly reduced.

An increase of several per cent. in the moisture of the underflow will cause a return to normal conditions and the thickener will have again its usual capacity.

These conditions have been observed at the Hollinger mill at Porcupine, where a mixture of quartz and schist is milled, but they do not appear to exist at the Porcupine-Crown. I have seen some indications that the slow motion of the arms in the thickener may be of actual service in securing a thicker discharge than could be obtained by undisturbed settling.

### *Uses*

The Dorr thickener was orginally introduced in cyaniding for thickening slime or reground pulp previous to agitation and filtration and came into general use for that purpose. The advantages of continuous thickening were enhanced when continuous agitation was shown to be profit-

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able and the use of a continuous filter made a complete process where actual operating labor was reduced to a very small amount.

*Continuous Counter-Current Decantation.*—This method for the recovery of pregnant solutions was first operated in this country, I believe, by John Randall in the Black Hills in 1901 with the large cones heretofore mentioned. Being familiar with his work and the difficulties met in maintaining continuous operation in cones, I recognized in 1907 that the thickener would make the process feasible, and designed the first plant to use it in 1910. This plant has been in successful operation up to the time of writing.

Since then over 20 plants have been erected or are now being installed to operate by this method. Several of them are replacing filters of different types with results apparently satisfactory to their owners, although no detailed figures have yet been published.

The method is especially adapted to the treatment of quick-settling pulp requiring a weak solution for the dissolution of gold and silver. Where pulp cannot be thickened to 50 per cent. moisture or less, or where a strong solution of cyanide is required, it will usually pay to install a filter after the decantation system to reduce the mechanical loss in cyanide and at the same time the dissolved losses which will then be higher.

The subject of continuous decantation is too large to be discussed in detail in this paper, but I hope that a paper will shortly be presented to the Institute covering actual results obtained and discussing it in theory as well as in practice. The advantages of its use in the leaching of copper ores with acid solutions is apparent, as was demonstrated by Dr. Rudolph Gahl several years ago in a test plant at the Detroit Copper Co. I believe that considerable test work is now being done on the same line.

*Concentration.*—The use of the thickener in concentration followed shortly after its wide adoption in cyaniding. The Nevada Consolidated Co. was the first of the large copper companies to use it in place of cones and obtained good results, as is shown by the following quotation from a letter from the mill superintendent there:

" . . . One thickener, 17 by 9 ft. deep, has a greater capacity than 12 by 8 ft. cone tanks, for the reason that there is only 1 unit to watch instead of 12, without the constant result of feed to 12 cones becoming unbalanced. Also the one 8-in. spigot used on the Dorr requires no attention compared with the dozen $\frac{3}{8}$-in. spigots on the cones. . . ."

It will be noted that the large difference in capacity per square foot is attributed to advantages in regular operation. Under *test* conditions no advantage has been found in a comparison of the thickener with 8-ft. cones on dilute pulps.

The recent installation of the Anaconda Copper Co. will be described, I understand, at this meeting.

The development of the flotation process has created a further demand



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for the machine and it has come into extensive use at Broken Hill, and in most of the plants in this country using that process.

The idea of many able concentrating men that the capacity of a settler is in proportion to its peripheral overflow rather than its area, appears, in fact, to have no good foundation where large-sized units are used for completely settling out all solids. One reason for this, I think, is that in such work it is common to have a varying amount of clear liquid always on the surface of the tank. When thickening pulp in an alkaline solution, as in cyanide work, where there is a sharp line between a coagulated slime of definitely heavier specific gravity and clear solution, it is often found that a thickener will have nearly the same capacity if the overflow is taken off by a pipe at one point, instead of by a peripheral launder carefully leveled. With the above conditions there will be a flow of liquid directly from the feed box in the center to the overflow pipe at one side. This will naturally tend to raise the slime level in that path above that in the remainder of the tank, and as a result, the slime-bearing liquid will sink as if it were a non-miscible liquid of greater specific gravity, and the clear liquid from all over the surface of the tank will flow toward the outlet pipe in its place. Samples taken at varying depths have shown almost equal density at equal depths at points in the path of such a flow and at the opposite side of the tank.

I do not wish to be understood as recommending the omission of the overflow launder; for when a tank is crowded to its limit it always allows more leeway before the slime overflows, and many operators have found considerable increase in capacity by its use, but cite this as bearing on the relations between capacity and area.

Caetani has discussed the matter very thoroughly in the *Mining and Scientific Press* of Mar. 22, 1913.

Recent work has shown the advantages of thickening both ahead of, and after flotation to the greatest density possible, using the warm overflow of the final thickeners to dilute the underflow of those ahead of flotation to the proper density for that process. This means conservation of a large percentage of the heat required and also a large saving in oil needed. One plant now following this procedure has found that, whereas laboratory tests indicated that over ½ lb. of oil was required per ton, the actual consumption does not exceed ⅛ lb.

*Industrial Uses.*—The thickener has been used successfully in rubber reclamation to settle some finely divided residues and it would appear that there should be many opportunities for its use where operations are on a large enough scale to make the advantages of continuous over intermittent work apparent. At one plant its introduction saved the labor of 12 men.

*Acid-proof Thickeners.*—The absence of submerged wearing parts has made the construction of acid-proof thickeners a comparatively simple

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matter and we have manufactured them for the Chiquicamata plant of the Chile Exploration Co., and for the Butte-Duluth Copper Co.

## The Dorr Tray Thickener

The tray thickener was designed by me in 1913 for the purpose of securing large settling area in a comparatively small space. I have not wished to make the invention widely known until we have had ample experience with actual operation under various conditions. Its development has been slow, owing to my desire to determine by actual installations, its limitations, advantages, and the simplest methods of construction.

### *Description*

The basic principle lies in the use of a plurality of superimposed shallow settling areas operating with a common mechanism either in

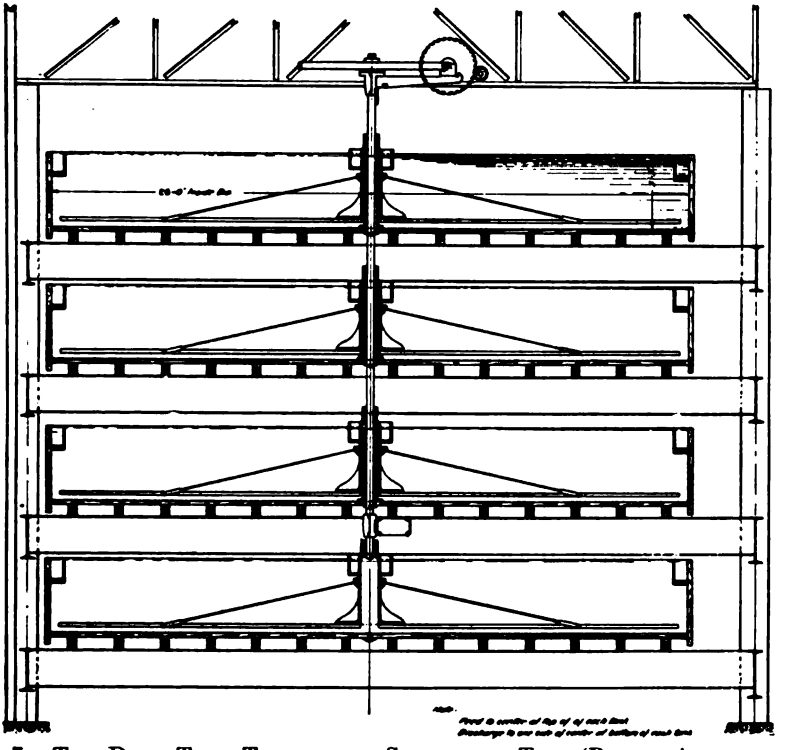

Fig. 7.—The Dorr Tray Thickener. Superposed Type (Patent Applied for).

separate tanks, as in the installation at Anaconda, Fig. 7, or with submerged trays in a deep tank, as at the Homestake, Tomboy, and Liberty

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Bell. The former method presents the advantages of accessibility, but the latter requires less head for a given capacity and gives no loss of

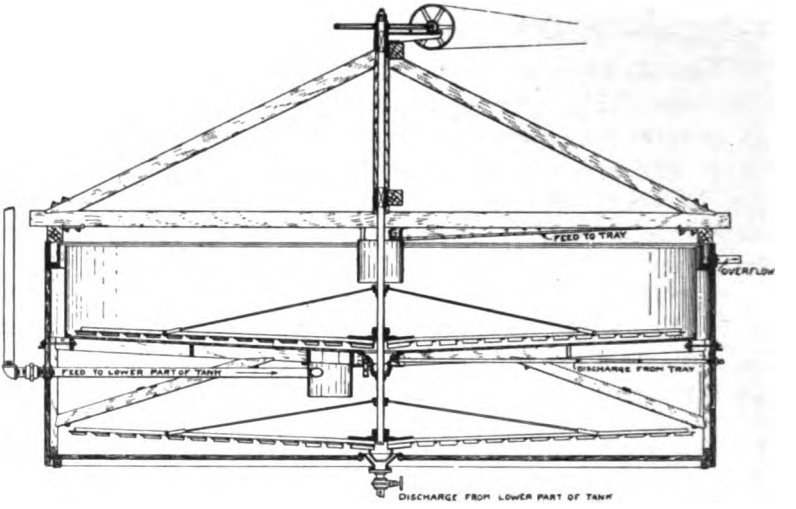


FIG. 8.—THE DORR TRAY THICKENER. SUBMERGED TYPE (PATENT APPLIED FOR).

mill height for the overflow. It has the merit of offering a simple means of increasing the capacity of any thickener in operation without the need

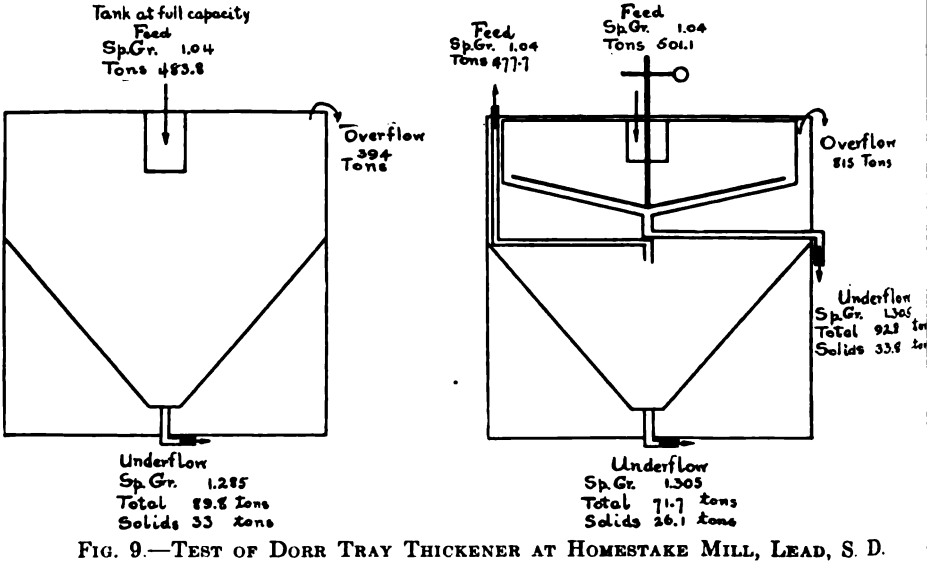


FIG. 9.—TEST OF DORR TRAY THICKENER AT HOMESTAKE MILL, LEAD, S. D.

of adding building, foundations, superstructures or materially changing launders, etc. We have only gone as far as adding one tray at present,

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in the case of the submerged trays, but have found no reason why two or three should not be used.

Fig. 8 shows our latest type of tray supported, as will be noted, both from the side and bottom of the tank. A cup attached to the rotating shaft carries a rubber gasket which makes contact with the small cone in the center of the tray and prevents passage of the slime to the space below. The thick-slime discharge is carried out at the side of the tank at present. Each settling space has an independent feed but the combined overflow of both is usually taken off at the rim of the tank. It has been found unnecessary to make the tray strong enough to support itself when full with the tank below empty, as an automatic float is provided to connect the tanks in an emergency.

The first test made on this type was at one of the settling plants of the Homestake Mining Co. through the courtesy of A. J. Clark, metallurgist in charge, and gave the results shown in Fig. 9. As will be noted, the tray was added to a cone-bottomed tank and the depth is greater than usual. It showed an increase in capacity of approximately 90 per cent.

The first tray to be installed in a Dorr thickener, already in operation, was at the Tomboy last fall and since then they have been added at the Liberty Bell and Tonopah-Belmont. We have also furnished more to the Tomboy.

### *Capacity*

Careful tests at the Liberty Bell, where a very good form of automatic distributor of pulp is in use, indicate that under conditions there, the tray placed in a 33 by 10 ft. thickener has added 100 per cent. of its settling capacity. The tests mentioned and those made by the Anaconda Co. indicate that the additional capacity to be obtained by the addition of the tray will depend upon the relative depth and discharge of the tank in which it is placed, the dilution of the feed and the underflow, and the nature of the material settled. It appears conservative to estimate that an addition of from 70 to 95 per cent. can thus be made. It must be recognized that a deep tank has a definite reserve capacity which is valuable when the feed and settling qualities of the pulp vary. If such variations occur during each day a thickener 12 ft. deep might need only the area required by the average feed, while one 3 or 4 ft. deep would have to be able to handle the maximum feed during a period and so might require considerably larger diameter. Mr. Waddell, manager of the Ohio Copper Co., has told me that the variations in the slime at the Nevada Consolidated were so great that tanks 20 ft. deep were warranted as equalizers. This plant now has eight tanks 50 ft. in diameter by 20 ft. deep besides 13 smaller ones.

The greatest field for the tray thickener is where tanks have to be

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housed and space is valuable; or heated pulps have to be settled and a minimum of radiation is required.

## The Dorr Agitator

### *History*

The Dorr agitator was designed in 1910 for a mill that I was then planning, but as it was decided to omit agitation at first, it reached the Patent Office before receiving an actual test. The first work done with it on a small scale was by Noel Cunningham, now superintendent of the Hollinger mill, who at that time was doing some test work for Mr. Jones of the Bel-

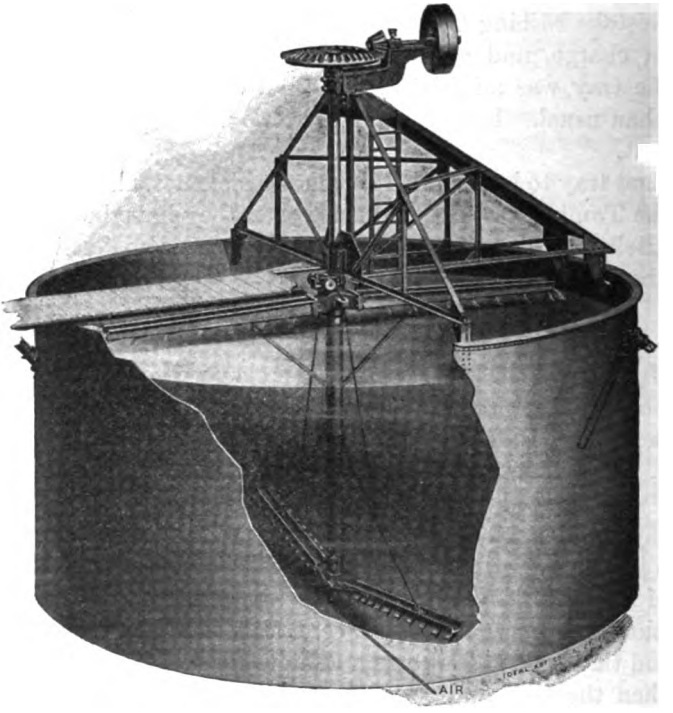

Fig. 10.—The Dorr Agitator (Patented).

mont, and the results obtained showed an average of nearly 4 per cent. better extraction than was made by the Pachuca tank against which it was tested.

The first machine to operate on a commercial scale was installed at the mill of the Hollinger Mines, Ltd., to handle an extremely quick-settling siliceous product which would stall the agitators originally used in 36 hr. It proved entirely successful and was operated for four months when a

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change to partial continuous decantation made direct agitation unnecessary and it was replaced by a thickener. The second installation of machines proved able to handle a 30-mesh product successfully.

Early in 1913 agitators were installed to handle concentrates at the high-grade mill of the Buffalo mines, at Cobalt, and they have given continuously good results on this most difficult work.

*Description*

The Dorr agitator as now made consists of a vertical pipe suspended in the center of a tank carrying at its lower end hinged sweeps with

FIG. 11.—THREE DORR AGITATORS, 32 FT. DIAMETER BY 15 FT. DEEP, AT BUCKHORN MILL, NEVADA.

scrapers adapted by their rotation to bring any settled material to the pipe. The upper end of the pipe terminates in a head casting carrying two distributing launders and supported by a short vertical shaft with driving mechanism carried by the roof trusses of the building or a special truss over the tank.

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A worm and sheaves provide means for raising or lowering the hinged arms at will. Air is introduced into the lifting pipe either at the bottom or by means of a pipe through the shaft, which is then made hollow.

Fig. 10 shows the machine and the method of operation. The Dorr agitator may be briefly described as a Dorr thickener, with only two hinged arms, and a central air-lift pipe with distributing launders substituted for the central shaft. Fig. 11 shows three of these machines operating in the Buckhorn mill, Nevada.

### *Speed*

The speed can be regulated to suit conditions. It is usually operated at from 1 to 3 rev. per minute, depending on the size of the tank and nature and density of the pulp being agitated.

### *Power*

The mechanical power taken varies from ¼ to 2 or 3 h.p. according to the size of tank, speed, nature and dilution of the pulp handled. The air required is from 8 to 40 cu. ft. per minute at 10 to 20 lb. pressure.

Tables below show data as given us by some of the engineers using the agitators. It will be recognized that it is difficult to measure small quantities of low-pressure air and mechanical power and to secure satisfactory data on the operation of individual machines.

The following covers what we have been able to secure on the operation of our agitators:

### *Data on Operation of Dorr Agitators*

*Buffalo Mines, Cobalt, Ont.*

Agitators on Concentrates in High-grade Mill

| Tank, Feet | Speed, Rev. per Min. | Dilution | Cu. Ft. Free Air per Minute |
|---|---|---|---|
| 20 by 12 | 10 | 1 : 2.8 | 28 |
| 10 by 10 | 7 | 1 : 2.8 | .. |
| 10 by 10 | 9 | 1 : 2.8 | .. |
| 10 by 10 [a] | 3 | 1 : 2.8 | .. |

[a] Using worm-gear drive.

Mechanical power not measured; all machines giving good results.

Agitator on Slime from Low-grade Mill

| Tank | Dilution | Cu. Ft. Free Air per Minute |
|---|---|---|
| 16 ft. 6 in. by 19 ft. | 1 : 1 | 40 |

The air measurements on these agitators were carefully made and are stated to be accurate within 1½ per cent.

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*Nevada Hills, Nevada*

Size of agitator, 30 by 14 ft.; speed, 6 rev. per minute; capacity, 80 tons solids at 1 :2½; semi-siliceous silver ore, 70 per cent., 200 mesh; mechanical power, 1.4 h.p.; air, 50 cu. ft.

Remarks: "Agitation good; no dead areas; no cost for repairs in 1½ years; practically no attendance."

*Liberty Bell, Colorado*

Ore largely silver, 17 per cent. +100, 70 per cent. −200 mesh; dilution 1 :2 . 15.

Size of agitator, 33 by 11 ft. Air that passes a ⅛-in. hole in $\frac{1}{16}$-in. lead gasket at about 19 lb. pressure.

Size of agitator, 17 by 17 ft. Cone bottom (formerly used for Hendryx agitator). Air passing through $\frac{1}{16}$-in. hole in gasket; mechanical power about 0.4 to 0.5 h.p. at 8 rev. per minute.

Remarks: Results of five tests showed practically no difference in extraction as compared with the Hendryx.

*Tonopah Mining Co., Nevada*

Ore silver, 75 per cent. −200 mesh; dilution 1 :1.

Size of agitator, 36 by 20 ft. Total power for air and mechanical operation estimated at 5 h.p.; speed, 3 rev. per minute; power estimated for paddle agitator with four air lifts previously used, 17 h.p.

Remarks: "Results satisfactory in every way."

*Mogul Mining Co., South Dakota*

Siliceous gold ore, 60-mesh product, 7 per cent. +100 mesh; 64 per cent. −200 mesh. Dilution 1 :1½.

Size of agitators, 16 by 16 ft. Air that passes $\frac{3}{16}$-in. diaphragm at 15 lb. pressure; power at 2 rev. per minute, measured by spring transmission to bevel-gear shaft, $\frac{1}{20}$ h.p.

*Buckhorn Mines Co., Nevada*

Ore, gold, very clayey.

Motor input on three 32 by 15 ft. agitators and lineshaft, 2.82 h.p.; actual output (allowing for power factor) 2.13 h.p. Agitation satisfactory; extraction reported to be better than tests indicated.

*Ophir Mines, Colorado*

(Not in operation)

Ore, gold; 75 per cent. −200 mesh, somehat clayey; dilution, 1 :1.

Size of agitators, 16 by 16 ft. Air, 8 to 10 cu. ft.; about 20 lb. pressure; speed, 1 rev. per minute; mechanical power under ⅛ h.p.

These agitators ran satisfactorily on 30-mesh stamp product.

## *Renewals*

Any machine of this type can be broken when started with the pulp packed if the arms have not been raised. It has, however, stood up

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under very hard usage and demonstrated that with very ordinary care there should be no breakage even with quick-settling material. All parts subject to a normal wear will last several years.

### *Efficiency*

The value of an agitator depends on the cost of making the maximum economic extraction by its use. This will be determined by its cost of agitation per ton per hour and the time required to obtain dissolution.

All the data I have been able to obtain indicate that the cost of agitation per hour is extremely low. Extraction tests that have been made indicate that the rate of dissolution on both gold and silver ores in Dorr agitators, with a uniform although less violent agitation, is at least as rapid as in any other agitators with which it has been compared.

The most favorable conditions for the dissolution of gold and silver in cyanide solution vary with each ore treated. The maintenance of an excess of dissolved oxygen throughout the whole mass of pulp and the free movement of all particles of solids in the liquid immediately adjacent seem to be the only conditions that can be generally specified.

Definite work both in milling and the laboratory indicate that many ores will give a more rapid extraction and allow the use of a weaker solution when agitated as a dilute pulp of 3 to 1 or 4 to 1, and also cause less chemical consumption of cyanide.

The Dorr agitator with its combination of air and mechanical agitation gives a flexibility that is apparent and it insures keeping *all* the solids in suspension all the time whether the pulp is subjected to a gentle or a violent movement. This is essential especially in continuous agitation.

It will thus be seen that the intensity of agitation can be regulated to suit the chemical needs of the ore being treated, rather than the mechanical necessity of keeping it in suspension in all parts of the tank.

### *Selective Agitation*

The question of selective agitation was discussed by Mark R. Lamb[1] in the *Transactions* of the Institute. As used here the expression is taken to mean continuous agitation under conditions which cause the coarser particles of the ore to remain in the agitator longer than the averge time of treatment and thus give them the longer exposure they may require to obtain the best extraction.

This can be accomplished readily with the Dorr agitator by agitating at a dilution which allows the coarser material to classify out and discharging the raised pulp near the center while the outflow is taken from near the periphery. It will be seen that if a segregation takes place and

[1] *Trans.*, xl, 775 (1909).

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the agitator is fed a pulp carrying 10 per cent. plus 100 mesh, the discharge may be only 5 per cent. plus 100 mesh at first. With twice as much inflow as outflow of 100-mesh material it will concentrate in the agitator while gradually increasing in the outflow until an equilibrium may be reached in which the feed and discharge will both be 10 per cent. and the average pulp in the tank 20 per cent., so that the exposure of coarse sand would be approximately double the average. While this is theoretical, enough work has been done to indicate that the idea can be put to practical use.

## Conclusion

In closing this description of Dorr equipment I wish to express my indebtedness to many friends in charge of the operation of plants using it, who by their constructive criticism and willingness to suggest and try out new ideas have greatly assisted me in making the many improvements that have been added from time to time and overcoming weak points as they have developed.

The classifier, for example, was designed originally to handle 100 tons of ore per day; but more work has been thrust upon it until at the Goldfield Consolidated it has taken care of 260 tons daily, including tube-mill returns. The demand of A. H. Jones, of the Tonopah-Belmont mill, that it be made capable of acting as a sump to take the drainage of the tube mill led to the designing of the present type, Model C, with rakes that can be readily raised 10 in. at the lower end.

I am particularly indebted to D. J. Nevill, of Stearns-Roger Manufacturing Co., who is our consulting mechanical engineer and who is largely responsible for the mechanical development of the present machines.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Annealing of Cold-Rolled Copper

BY EARL S. BARDWELL, GREAT FALLS, MONT.

(Salt Lake Meeting, August, 1914)

THE determination of suitable and safe annealing temperatures is one of the most important problems arising in the operation of a copper rolling mill. Certain of the larger mills have worked this problem out for themselves and in these mills the operation has become more or less standardized. In some of the mills pyrometers are used to measure the temperature of the annealing furnaces and careful watch is kept to see that overheating does not take place. In a considerable number of the mills, however, dependence is placed entirely on the judgment of the operator. The almost total lack of literature bearing on this subject seems sufficient warrant for the publication of this paper.

The discussion of this subject naturally divides itself into two parts: (1) The effect of annealing temperature on physical properties; and (2) changes in the microstructure of refined copper due to cold rolling and annealing.

## I. THE EFFECT OF ANNEALING TEMPERATURE ON PHYSICAL PROPERTIES

For the purpose of investigating this phase of the subject six test bars were selected, varying in oxygen content between the limits of 0.036 and 0.070 per cent. The chemical composition of each of these test bars was as follows:

| | Bar Number | | | | | |
|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 |
| Copper + silver ..... | 99.95 | 99.92 | 99.945 | 99.94 | 99.95 | 99.93 |
| Oxygen..... | 0.041 | 0.070 | 0.046 | 0.050 | 0.036 | 0.058 |
| As + Sb ..... | 0.0038 | 0.0038 | 0.0038 | 0.0038 | 0.0031 | 0.0031 |

The bars were heated in an electric muffle to about 1,650° F. and rolled into rods. The rods were annealed, pickled, and drawn to No. 12 B. & S. gauge wire. The wire which was to be used for the annealing test was cut into 38-in. lengths and each length made into a coil 6 in. in diameter. The separate coils were marked so that they might readily be identified

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as they were taken out of the annealing furnace. For annealing the wires an electric muffle was used and the annealing temperatures were measured by means of a thermo-couple. The temperatures as measured are accurate to about 10° and we were able, in practically every case, to so regulate the furnace that the temperature did not rise more than 20° above or fall more than 20° below the desired temperature while the wires were in process of being annealed. The muffle was brought to the desired temperature and six wires, one from each of the test bars, were introduced. The wires were allowed to remain in the furnace for 20 min., at the end of which time they were removed and plunged into a dilute sulphuric acid solution to remove scale. In this manner wires from each of the test bars were annealed at a series of temperatures varying between the limits of 300° and 1,800° F.

*Electro-Conductivity of Annealed Wires.*—The conductivities of the wires were measured by means of a Hoopes bridge manufactured by the Leeds-Northrup Co., of Philadelphia. The results of the conductivity tests are given in Table I.

TABLE I.—*Conductivities of Copper Wires Annealed at Various Temperatures for 20 Min.*

| Temperature, | Per cent. Conductivities of Wires Drawn from Bar Numbers | | | | | |
|---|---|---|---|---|---|---|
| Degrees F. | 1 | 2 | 3 | 4 | 5 | 6 |
| 60 | 97.3 | 96.2 | 97.6 | 96.9 | 97.2 | 97.0 |
| 300 | 97.45 | 96.35 | 97.6 | 97.1 | 97.4 | 96.9 |
| 400 | 97.5 | 96.4 | 97.8 | 97.1 | 97.5 | 97.1 |
| 525 | 97.8 | 96.6 | 97.9 | 97.3 | 97.7 | 97.4 |
| 625 | 97.9 | 97.2 | 98.0 | 97.5 | 97.9 | 97.4 |
| 700 | 99.5 | 99.0 | 100.0 | 100.0 | 99.9 | 99.8 |
| 800 | 100.3 | 99.6 | 100.4 | 100.0 | 100.3 | 100.1 |
| 900 | 100.2 | 99.6 | 100.4 | 100.0 | 100.3 | 100.1 |
| 1,000 | 100.3 | 99.5 | 100.3 | 100.0 | 100.2 | 100.1 |
| 1,100 | 100.3 | 99.3 | 100.3 | 99.9 | 100.1 | 99.9 |
| 1,200 | 100.2 | 99.3 | 100.3 | 100.0 | 100.1 | 99.9 |
| 1,300 | 100.1 | 99.2 | 100.3 | 99.9 | 100.2 | 99.8 |
| 1,400 | 100.1 | 99.1 | 100.3 | 100.0 | 99.9 | 99.5 |
| 1,500 | 99.85 | 98.7 | 100.1 | 99.6 | 99.65 | 99.2 |
| 1,600 | 99.4 | 98.5 | 99.65 | 99.1 | 99.25 | 98.7 |
| 1,700 | 98.9 | 97.6 | 98.7 | 98.3 | 98.3 | 98.1 |
| 1,800 | 97.8 | 96.7 | 98.0 | 97.3 | 97.0 | 97.0 |

The per cent. conductivity is referred to the annealed copper standard, which at 20° C. is represented by a resistivity of 0.15328 ohm (meter, gram). The conductivities reported at 60° F. are the conductivities of the hard-drawn wire. The low conductivities in the case of wires annealed at temperatures between 300° and 700° F. are due to imperfect or incomplete annealing, whereas the low conductivities obtained in the case of wires annealed at the higher temperatures are due to changes in

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the physical structure of the copper brought about by annealing at these temperatures.

The curves, Fig. 1, are plotted with conductivities as ordinates and temperatures at which the wires were annealed as abscissæ. It will be noted that until an annealing temperature of about 600° F. is reached the conductivity increases very gradually; between 600° and 800° F. a very rapid increase in conductivity takes place. Wires annealed at 800° F. and at higher temperatures up to 1,200° F. show very little or no decrease in conductivity. Wires annealed at temperature higher than 1,200° F. in general show a decrease in conductivity, slight at first, but increasing rapidly until in the case of wires annealed at 1,800° F. the conductivity has become very nearly equal to that of the hard-drawn wire.

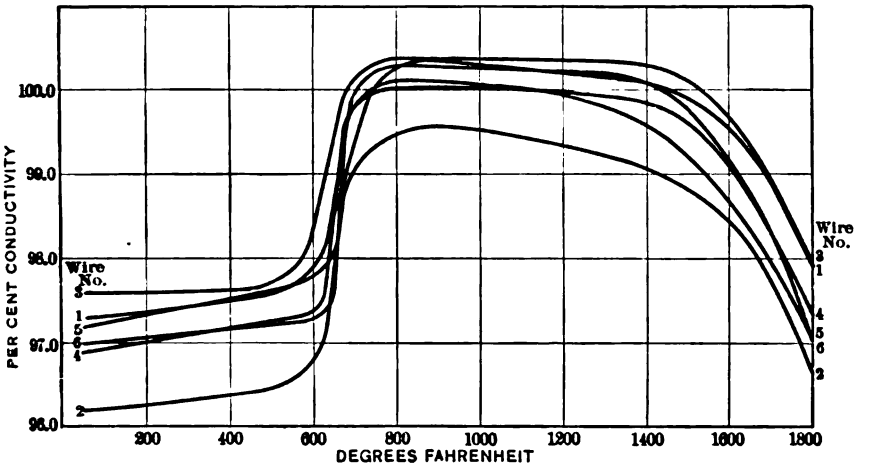


FIG. 1.—VARIATION IN CONDUCTIVITY OF COPPER WIRES WITH VARIATIONS IN ANNEALING TEMPERATURE,

It is interesting to note that wires drawn from bars 1, 3, 4, and 5 show no considerable decrease in conductivity until an annealing temperature in excess of 1,400° is attained, whereas the wires drawn from bars 2 and 6 show a marked decrease in conductivity when annealed at a temperature higher than 1,200° F. This is more especially marked in the case of the wire drawn from bar 2, which bar is the higher in oxygen.

*Tensile Strength and Elongation of Annealed Wires.*—After testing the annealed wires for conductivity they were tested for tensile strength and elongation. In making the tensile-strength determinations 1-ft. lengths of each of the various wires were used and the elongation measured to thousandths of an inch. Slight imperfections in the wire are responsible for the erratic results obtained in the case of several of the wires. Results which were clearly due to flaws in the wire have been disregarded.

Table II gives the results of the tensile-strength tests. In Fig. 2 these results are shown graphically.

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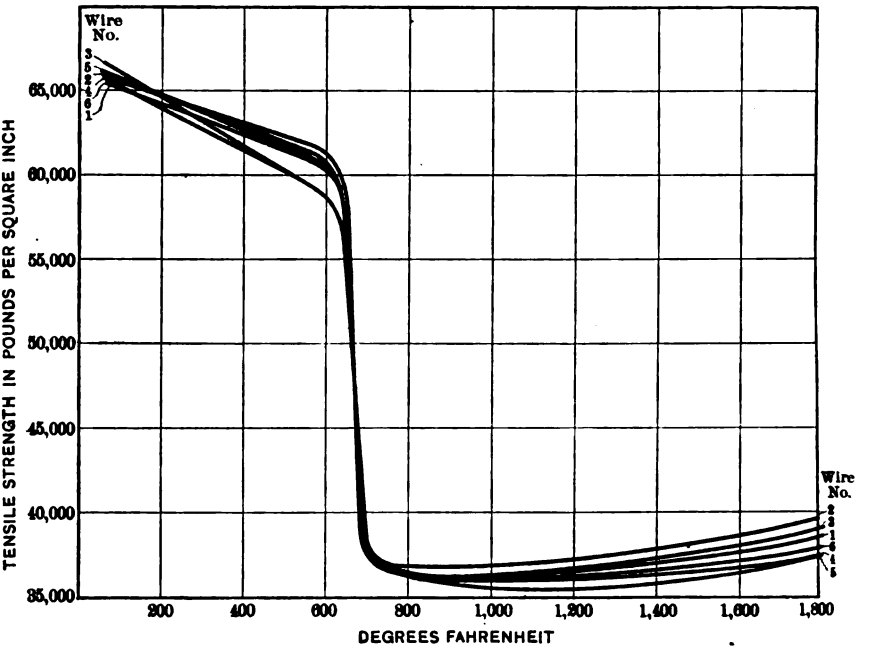


FIG. 2.—VARIATION IN TENSILE STRENGTH OF COPPER WIRE WITH VARIATIONS IN ANNEALING TEMPERATURE.

TABLE II.—*Tensile Strength of Copper Wires Annealed at Various Temperatures for 20 Min.*

| Temperature, Degrees F. | Tensile Strength, Pounds per Square Inch, of Wires Drawn from Bar Number | | | | | |
|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 |
| 60 | 63,100 | 65,700 | 66,700 | 66,100 | 63,700 | 65,600 |
| 300 | 62,800 | 62,500 | 63,500 | 60,000 | 63,800 | 63,800 |
| 400 | 61,200 | 61,750 | 61,900 | 59,900 | 61,700 | 62,500 |
| 525 | 61,200 | 61,750 | 59,200 | 59,800 | 61,800 | 61,800 |
| 625 | ..... | 60,750 | 58,100 | 57,400 | 62,500 | ..... |
| 700 | 39,400 | 37,800 | 38,200 | 37,800 | 35,800 | 37,400 |
| 800 | 36,200 | 36,400 | 36,200 | 37,000 | 37,000 | 36,600 |
| 900 | 35,800 | 37,000 | 36,200 | 36,200 | 33,500 | 36,200 |
| 1,000 | 36,900 | 37,000 | 36,800 | 36,200 | 36,000 | 36,200 |
| 1,100 | 35,800 | 35,600 | 36,400 | 36,200 | 35,400 | 36,200 |
| 1,200 | 36,400 | 37,000 | 36,400 | 35,800 | 35,100 | 36,600 |
| 1,300 | 34,700 | 36,800 | 37,200 | 35,200 | 35,500 | 35,600 |
| 1,400 | 36,900 | 35,600 | 37,400 | 36,400 | 36,200 | ..... |
| 1,500 | 36,400 | ..... | 38,400 | 37,600 | 35,800 | 36,400 |
| 1,600 | 38,200 | 38,700 | 37,400 | 35,200 | 37,700 | 37,800 |
| 1,700 | 38,100 | 38,600 | 38,000 | 36,900 | 37,400 | 38,900 |
| 1,800 | 38,400 | 39,800 | 39,200 | 37,200 | 37,300 | 37,200 |

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The tensile strength was figured by multiplying the breaking weight in pounds by the reciprocal of the cross-sectional area in square inches. The scale formed by annealing and subsequently removed by pickling caused a considerable reduction in the cross-sectional areas of wires annealed at the higher temperatures.

The curves show essentially the same characteristics.

In the range between 300° and 600° there is a slight decrease in tensile strength, followed by a sudden drop in the narrow range between 600° and 700°. Between 700° and 1,400° there is very little change in the tensile strength, which reaches a minimum at about 1,100°. Beyond 1,400° the tensile strength seems to increase slightly.

Table III gives in detail the per cent. elongation obtained in the case of each wire.

TABLE III.—*Per Cent. Elongation of Copper Wires Annealed at Various Temperatures for 20 Min.*

| Temperature, | Per cent. Elongation of Wires Drawn from Bar Number | | | | | |
|---|---|---|---|---|---|---|
| Degrees F. | 1 | 2 | 3 | 4 | 5 | 6 |
| 60 | 0.7 | 0.9 | 1.0 | 0.9 | 1.2 | 1.0 |
| 300 | 1.7 | 1.9 | 1.7 | 1.5 | 1.9 | 1.6 |
| 400 | 1.7 | 1.9 | 2.3 | 2.1 | 1.7 | 1.9 |
| 525 | 1.6 | 2.3 | 1.8 | 2.1 | 2.0 | 2.0 |
| 625 | .... | 2.5 | 2.7 | 1.8 | 2.0 | 1.8 |
| 700 | 21.2 | 39.4 | 35.2 | 29.6 | 26.1 | 28.5 |
| 800 | 46.0 | 42.6 | 48.0 | 42.4 | 42.4 | 40.5 |
| 900 | 46.8 | .... | 43.2 | 43.6 | 41.2 | 41.8 |
| 1,000 | 42.7 | 43.2 | 44.5 | 42.7 | 41.1 | 41.2 |
| 1,100 | 46.2 | .... | 45.3 | 43.8 | 41.8 | 40.0 |
| 1,200 | 42.5 | .... | 49.7 | 41.2 | 42.5 | 42.1 |
| 1,300 | 43.7 | 42.0 | 43.6 | 43.5 | 41.9 | 41.8 |
| 1,400 | 41.7 | 42.8 | 40.8 | 40.1 | 41.4 | 40.0 |
| 1,500 | 37.7 | .... | 39.0 | 38.4 | 34.8 | 37.9 |
| 1,600 | 32.9 | 37.2 | .... | .... | 36.8 | 35.2 |
| 1,700 | 34.6 | 34.0 | 32.9 | 32.3 | 33.1 | 34.4 |
| 1,800 | 26.3 | 25.6 | 28.7 | 30.0 | 28.3 | 27.4 |

These results are shown graphically in Fig. 3. As will be noted upon examining the curves, there is a slight increase in the per cent. elongation as we pass from 300° to 600°, followed by a rapid increase in the range between 600° and 800°. From 800° to 1,300° the per cent. elongation remains fairly constant, reaching a maximum at about 1,100°, after which it decreases slowly at first and then more rapidly until at 1,800° the elongation is from 15 to 20 per cent. lower than in the case of wires annealed at 1,100°.

In Table IV the results of physical tests on the six wires annealed at different temperatures will be found averaged.



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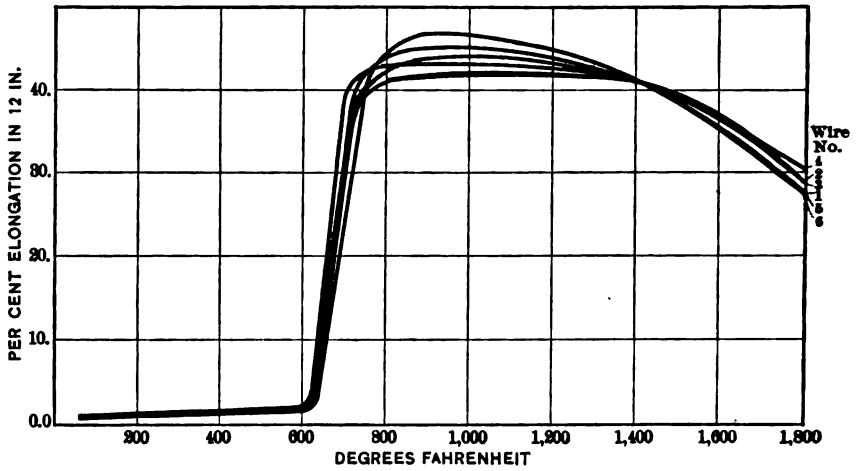


FIG. 3.—VARIATION IN PER CENT. ELONGATION OF COPPER WIRES WITH VARIATIONS IN ANNEALING TEMPERATURE.

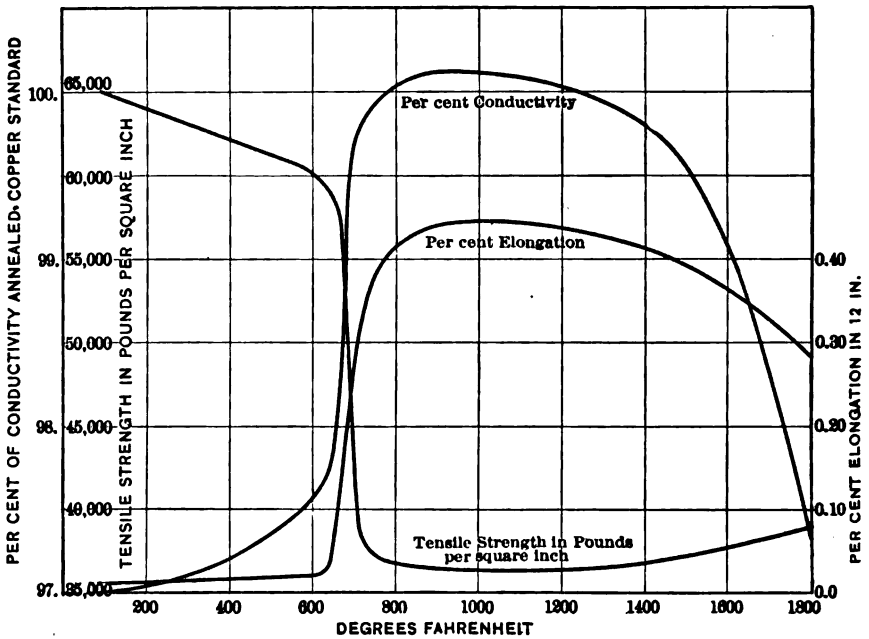


FIG. 4.—VARIATION OF PHYSICAL PROPERTIES OF COPPER WIRE WITH VARIATIONS IN ANNEALING TEMPERATURE.

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TABLE IV.—*Composite Tabulation of Results of Physical Tests on Wires Annealed at Different Temperatures*

| Temperature, Degrees F. | Conductivity, Per cent. | Tensile Strength, Pounds per Square Inch | Elongation, Per cent. |
|---|---|---|---|
| 60 | 97.C | 65,150 | 0.9 |
| 300 | 97.1 | 62,750 | 1.7 |
| 400 | 97.2 | 61,490 | 1.9 |
| 525 | 97.4 | 60,925 | 2.0 |
| 625 | 97.6 | 59,690 | 2.1 |
| 700 | 97.7 | 37,730 | 30.0 |
| 800 | 100.1 | 36,570 | 43.6 |
| 900 | 100.1 | 35,820 | 43.3 |
| 1,000 | 100.1 | 36,520 | 42.5 |
| 1,100 | 100.0 | 35,930 | 43.4 |
| 1,200 | 100.0 | 36,220 | 43.6 |
| 1,300 | 99.9 | 35,820 | 42.7 |
| 1,400 | 99.8 | 36,500 | 41.1 |
| 1,500 | 99.5 | 36,920 | 37.5 |
| 1,600 | 99.1 | 37,500 | 35.5 |
| 1,700 | 98.3 | 37,980 | 33.5 |
| 1,800 | 97.3 | 38,180 | 27.7 |

These data are plotted in Fig. 4. The curves show that the range between 800° and 1,300° F. is best suited for annealing. Wires annealed at around 1,100° F. have the maximum conductivity, maximum per cent. elongation, and minimum tensile strength.

Captain C. Grard, of Paris, France[1] has carried out a series of tests with reference to the effect of various annealing temperatures on cold-worked brasses of various compositions and copper. This investigator has not studied the effect of annealing temperature upon the electro-conductivity of the copper; he has, however, studied the effect of the annealing temperature upon the tensile strength and elongation of cold-worked copper. Fig. 5 is a reproduction of the curves given in Grard's paper. In Fig. 5 the temperatures are expressed in degrees centigrade, whereas in Figs. 1, 2, 3, and 4 the temperatures are given in degrees Fahrenheit. While the curve for elongation is of the same general shape as that obtained by us, the transition which we have found to take place between 600° and 800° F. he has found to take place between 150° and 200° C., or in the range between say 302° and 392° F. This may be partly due to the fact that the copper which Grard used for his experiments was much higher in oxygen than that used by us. As nearly as can be ascertained from Grard's paper, the copper used for his experi-

[1] Captain C. Grard, Paris: Industrial Application of Microscopic Metallography for Controlling the Work put on Copper and Brasses, International Association for Testing Materials, vol. ii, No. 11, $II_{18}$.

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ments contained 0.15 per cent. oxygen. Grard does not state the length of time that each specimen was subjected to the annealing temperature. The difference noted between his curves and ours may be due to specimens being annealed for longer periods at the lower temperatures than was the case at higher temperatures.

Grard divides his curve into five zones. From 0 to 150° C. he calls the "zone of cold working," from 150° to approximately 200° C. is the "relaxed zone," from 200° to 400° the "zone of recuperation," from 400° to 650° is the "zone of complete annealing," and from 650 ° C. up is the "bending zone." Converting degrees centigrade into degrees Fahrenheit we have:

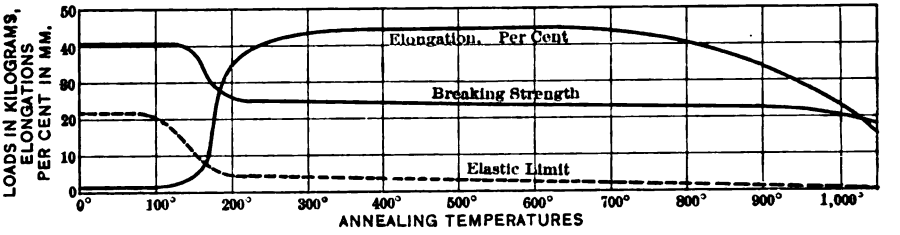


FIG. 5.—GRARD'S CURVES SHOWING EFFECT OF ANNEALING TEMPERATURE ON PHYSICAL PROPERTIES OF COPPER.

| | |
|---|---|
| Zone of cold working..... | 32° to 302° F. |
| Relaxed zone..... | 302° to 392° F. |
| Zone of recuperation..... | 392° to 752° F. |
| Zone of complete annealing..... | 752° to 1,202° F. |
| Bending zone..... | 1,202° to melting point of copper |

Using the same terminology, the curves for per cent. elongation shown in Fig. 3 might be divided into zones as follows:

| | |
|---|---|
| Zone of cold working..... | 32° to 600° F. |
| Relaxed zone..... | 600° to 750° F. |
| Zone of recuperation..... | 750° to 900° F. |
| Zone of complete annealing..... | 900° to 1,200° F. |
| Bending zone..... | 1,200° to melting point of copper |

This is, of course, only approximate. It is extremely difficult if not altogether out of the question to determine with any considerable degree of accuracy the per cent. elongation of a 1-ft. length of No. 12 copper wire that has not been annealed. The results which have been obtained are consistent and serve our purpose. As will be noted, the point which we have fixed upon as marking the beginning of the bending zone is the same point that Grard has fixed upon. The electro-conductivity can be measured with a much greater degree of accuracy than can the per cent. elongation, and the conductivity of a copper wire is perhaps an even more delicate index of the changes that take place in

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the copper upon annealing. If we base the limits of the several zones on the conductivity curves (see Fig. 1) we get results somewhat as follows:

| | |
|---|---|
| Zone of cold working..... | 32° to 300° F. |
| Relaxed zone..... | 300° to 675° F. |
| Zone of recuperation..... | 675° to 800° F. |
| Zone of complete annealing..... | 800° to 1,100° F. |
| Bending zone..... | 1,100° to melting point of copper |

Grard states that the first two zones should never correspond to a finished state of the worked metal, as the metal in either of these zones is in a state of unstable equilibrium. He states further that the so-called bending zone should be avoided. This means that copper which attains in the annealing furnace a temperature higher than 1,100° to 1,200° F. has been overheated. Where copper is to be cold rolled after annealing a temperature of 1,100° F. will be found to give most satisfactory results. I do not wish to convey the idea that heating to higher temperature than 1,100° F. invariably injures the copper. Such is not the case. The nature of the troubles that may be brought about by overheating will be discussed on another page.

## 2. Changes in the Microstructure of Refined Copper due to Cold Rolling and Annealing

Fig. 6 is a photomicrograph of a piece of copper that has been cold rolled from $\frac{3}{16}$ to 0.033 in. In the photomicrograph note the slip bands, very fine parallel lines at right angles to the direction of rolling. These slip bands are characteristic of a strained condition of the metal resulting from severe mechanical treatment. In order to illustrate the affect of annealing upon the microstructure of cold-rolled copper, pieces of the above sheet were annealed in a small electric furnace at a number of different temperatures, each specimen being allowed to remain in the furnace for 20 min. The temperatures were measured by means of a thermo-couple standardized by the United States Bureau of Standards and are correct to 5° C. The temperatures which were measured in centigrade degrees have been converted to Fahrenheit degrees in order to avoid confusion and to facilitate comparison of microstructure with physical properties as shown by the curves in Figs. 1, 2, 3, and 4.

Fig. 7, a photomicrograph of a piece of the above-mentioned cold-rolled copper annealed for 20 min. at 392° F., shows the metal to be still in a strained condition. Under the microscope we see no evidence of crystals. The slip bands noted in Fig. 6 are visible also in Fig. 7.

In Fig. 8 we note a change. The slip bands have disappeared and in their place we have small crystals very closely grouped. We have now passed out of the zone of cold working and have entered the relaxed zone, which may be said to be characterized by very small crystals.

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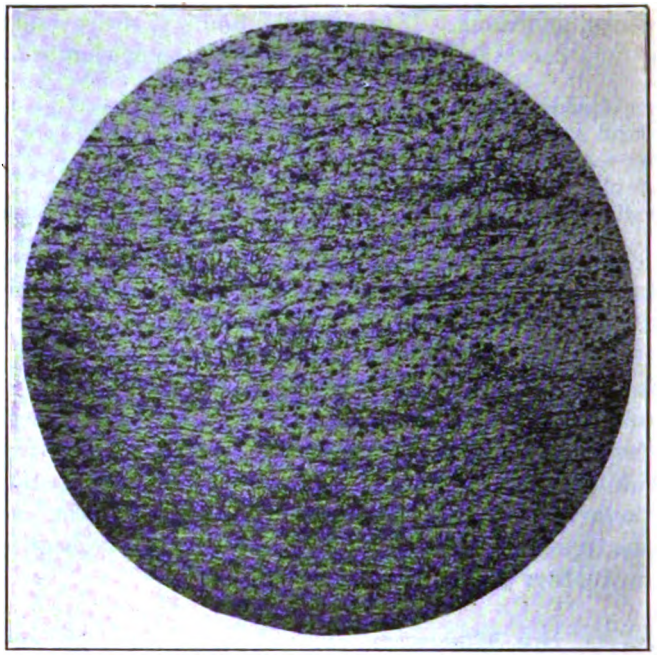

FIG. 6.—SHEET COPPER AFTER COLD ROLLING FROM 3/16 IN. TO 0.033 IN. × 100.

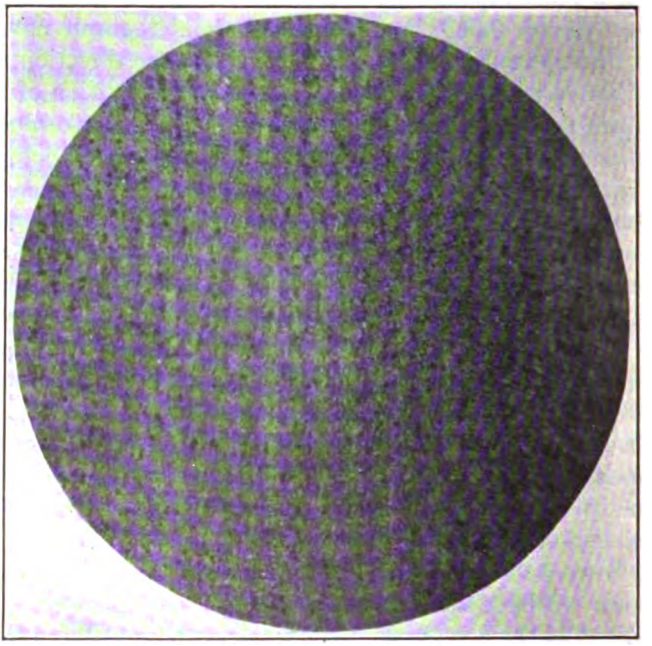

FIG. 7.—AFTER ANNEALING AT 392° F. × 100.

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Fig. 9 is a photomicrograph of a specimen of the cold-rolled sheet which has been annealed for 20 min. at 752° F. This photomicrograph is characteristic of the zone of recuperation. The crystals are still very small, but are slightly larger than those shown in Fig. 8 and more regular in shape.

With Fig. 10 we enter the zone of complete annealing. The crystals in this specimen, which was annealed at 932° F., are much larger and more regular in outline. With Fig. 11, 1,112° F., they have reached their maximum regular growth. From this point on the crystals increase very

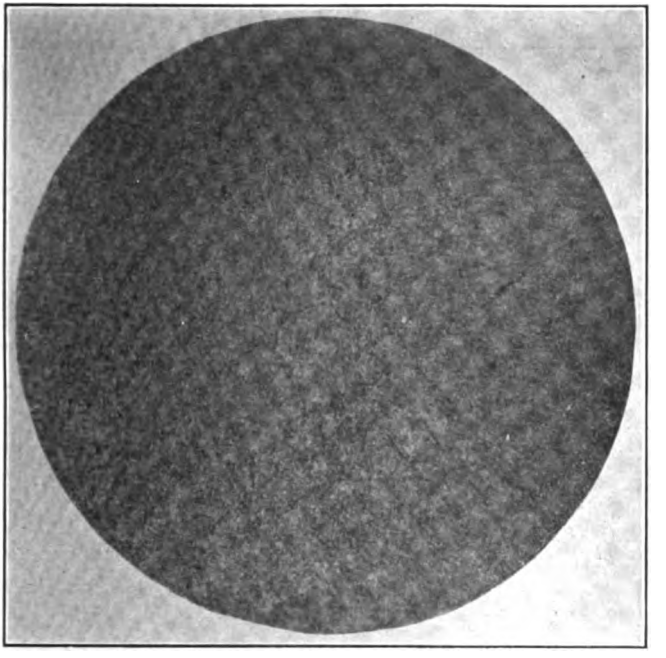

FIG. 8.—AFTER ANNEALING AT 572° F. × 100.

rapidly in size but they grow at the expense of one another and hence become irregular in size; *i.e.*, some crystals are very large while others are much smaller. This irregular growth, and the further fact that the crystal outlines are becoming more and more rounded in the case of specimens annealed at the higher temperatures, probably explain in part the decrease in ductility which we have noticed in the case of the wires annealed at different temperatures. In Figs. 12, 13, 14, and 15 the crystals become successively larger, the crystal faces become more and more curvilinear in outline, and polysynthetic twin crystals make their appearance.

The microstructure of refined copper that has been annealed after

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rolling differs from that of the cast metal by the parallel markings (polysynthetic twin crystals) which appear on the crystal faces (see especially Figs. 14 and 15). The significance of these twin crystals is not entirely clear. The distortion of the metal brought about by rolling results in a refinement of the crystalline structure and an increase in tensile strength. This refinement of the crystal structure is brought about by sliding taking place along the cleavage planes of the polygonal grains. As a result the new crystals become arranged in twinning position with their neighbors. The temperature at which the strained metal is annealed seems to exert a considerable influence upon the formation of these poly-

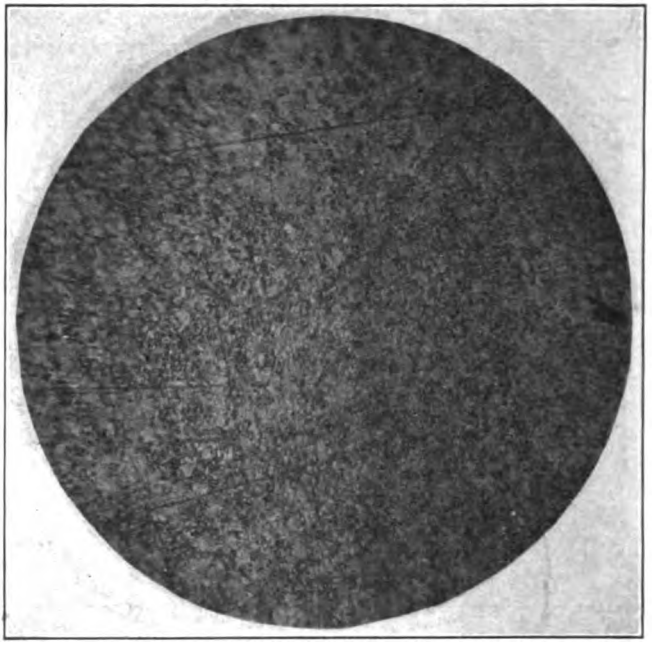

FIG. 9.—AFTER ANNEALING AT 752° F. × 100.

synthetic twin crystals as well as upon the size of the crystals in general. This increase in the size of the crystals is accompanied by a decrease in the per cent. elongation. The decrease in conductivity and in general the changes which take place in the case of the physical properties of wire annealed at the higher temperatures show a reversion of the metal to a condition very much like that of the cast metal. It is of interest to note that the appearance of the so-called polysynthetic twin crystals is coincident with the decrease in conductivity and per cent. elongation. This decrease in the per cent. elongation is of vital interest to the rolling-mill man, it being evident that a piece of the annealed metal having the maxi-

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mum per cent. elongation is in the best possible physical condition for withstanding further mechanical treatment.

The question will probably arise as to what sort of trouble may be brought about by overheating the copper in the annealing furnace. We have seen that overheating facilitates the development of polysynthetic twin crystals and results in a decrease in the per cent. elongation. Sheets which have been overheated in this way give trouble by cracking at the edges. If the annealing temperature is very high, gases which were unable to escape when the metal was cast tend to segregate at the granular boundaries. This results in blisters, which may make themselves evi-

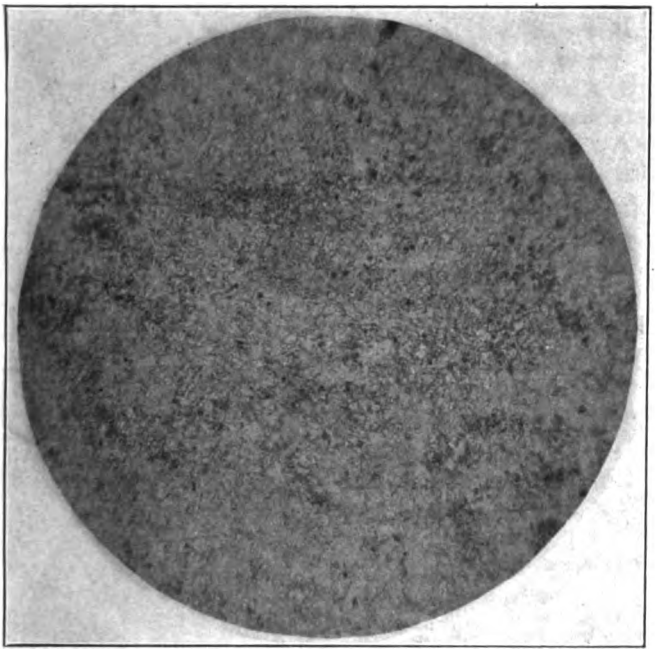

FIG. 10.—AFTER ANNEALING AT 932° F. × 100.

dent only upon subsequent rolling. Overheating of the copper previous to the breaking-down rolls has a similar effect. When the annealing temperature becomes so high that the metal begins to soften, the gas contained in the metal will tend to force the grains apart, causing cracks, into which the oxygen of the atmosphere will penetrate. The copper in this condition is said to be burnt and must be remelted. Overheating does not necessarily mean that the copper is burnt. If the copper attains in the annealing furnace a temperature in excess of 1,100° or 1,200° F. it may be said to have been overheated, but no harm necessarily results if the overheating is inconsiderable. We have already seen that copper

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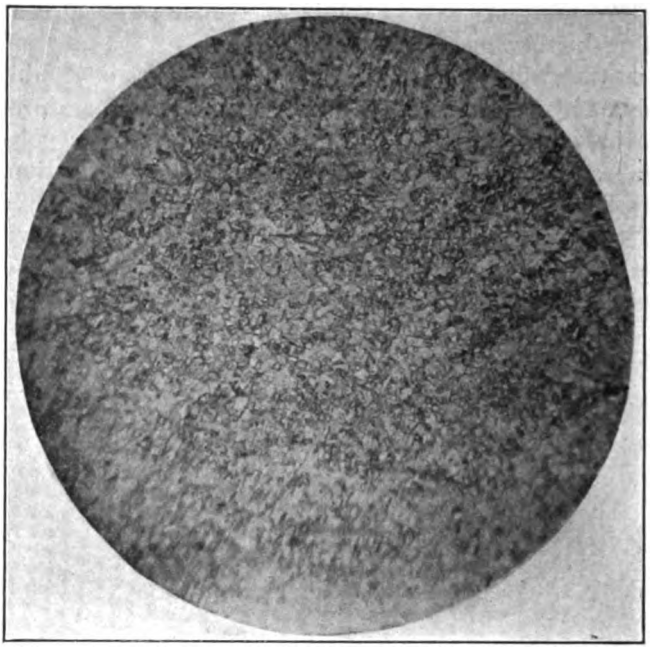

FIG. 11.—AFTER ANNEALING AT 1,112° F. × 100.

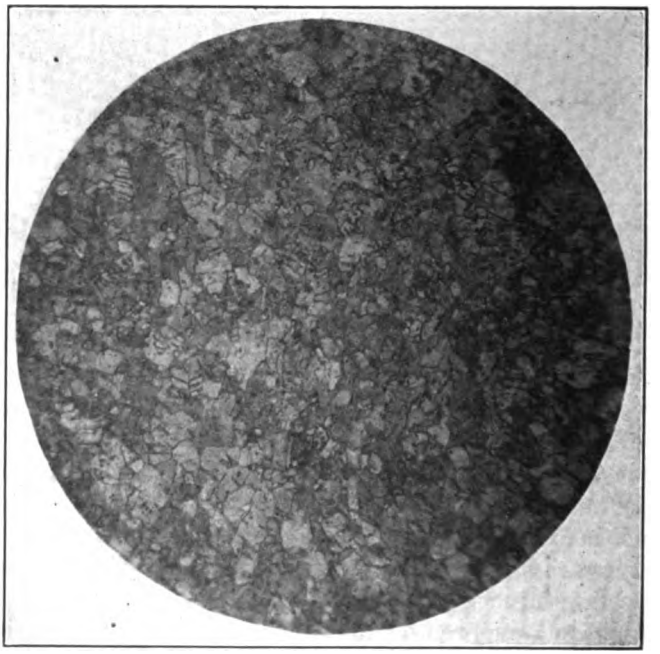

FIG. 12.—AFTER ANNEALING AT 1,292° F. × 100.

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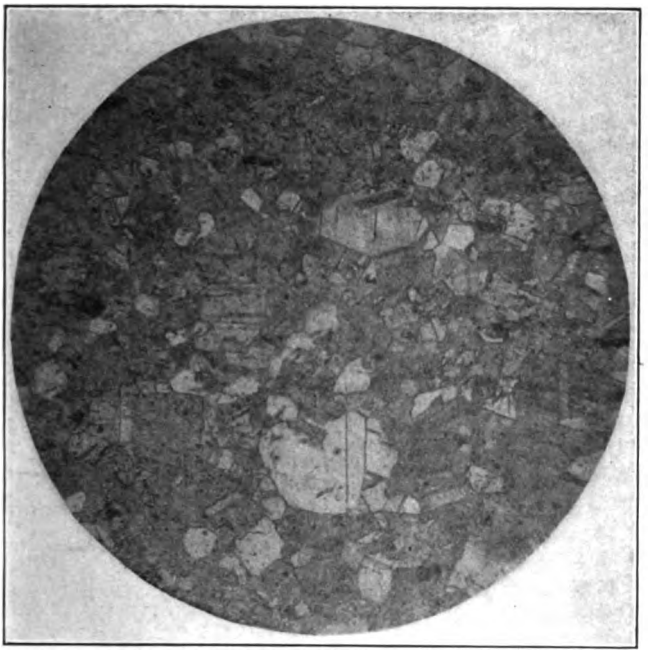

FIG. 13.—AFTER ANNEALING AT 1,472° F. × 100.

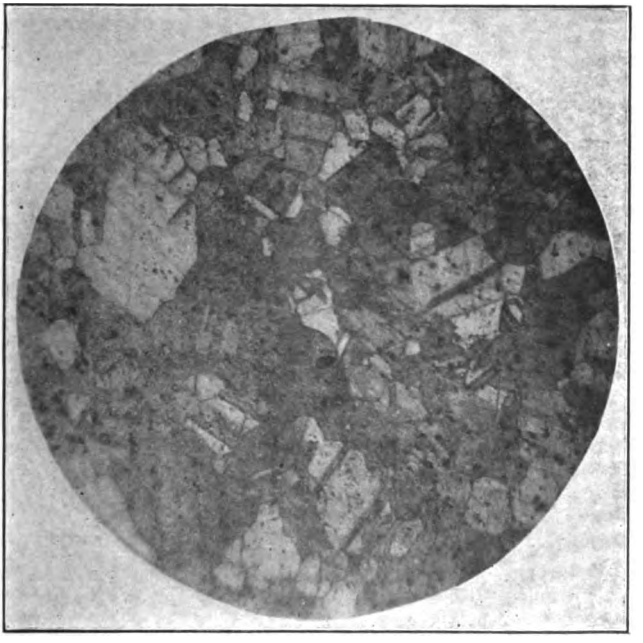

FIG. 14.—AFTER ANNEALING AT 1,652° F. × 100.

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FIG. 15.—AFTER ANNEALING AT 1,832° F. × 100.

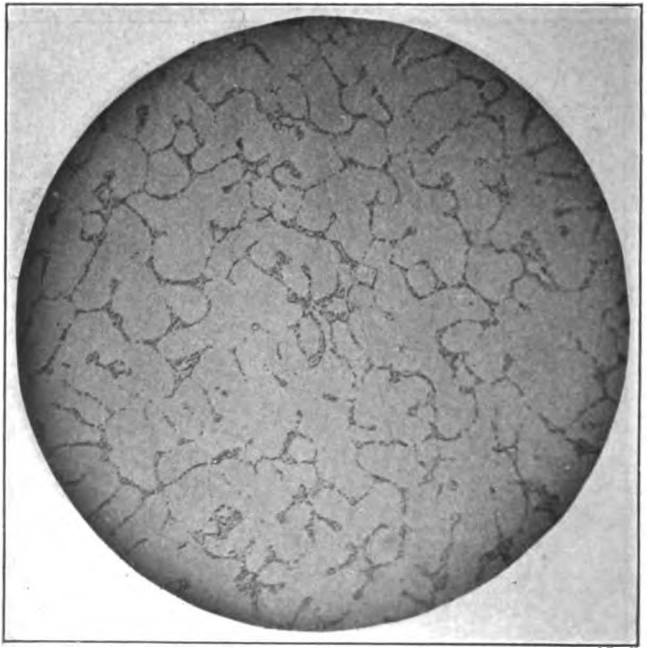

FIG. 16.—CAKE CONTAINING 0.076 PER CENT. OXYGEN. × 41.

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low in oxygen will apparently permit of being heated to 1,300° to 1,400° F. before showing any considerable decrease in either the per cent. conductivity or per cent. elongation.

In order to further illustrate my point it may be of interest to show what happens when a cake of electrolytic copper is rolled into sheet. Fig. 16 is a photomicrograph of a piece of copper cut from a 14 by 17 in. 200-lb. cake containing 0.076 per cent. oxygen. In the photomicrograph the dark network is the eutectic mixture of $Cu_2O$ and copper containing approximately 0.39 per cent. oxygen. The cake was heated to about

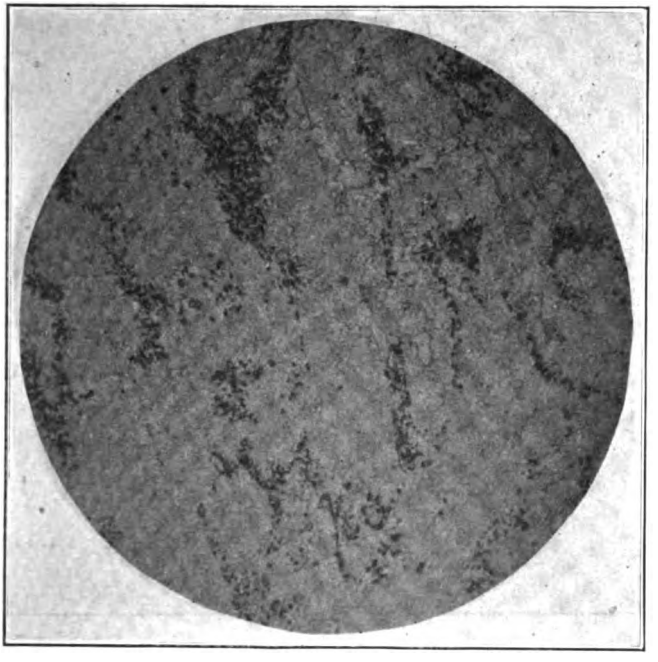

FIG. 17.—3/16 IN. SHEET ROLLED FROM CAKE CONTAINING 0.076 PER CENT. OXYGEN BEFORE SCALING. × 87.

1,560° F. and broken down to $\frac{3}{16}$ in. in thickness by repeated passes through the rolls. Fig. 17 gives a photomicrograph of a piece of the resulting sheet previous to its being scaled. In Fig. 17 it will be noted that while the eutectic has ceased to exist as a continuous network it has not been entirely broken up. A determination of the oxygen made by measuring the area occupied by the dark patches of eutectic and multiplying this area expressed in per cent. by 0.39 gives 0.077 as the oxygen content of the sheet, checking almost exactly the measurement previously made upon a piece of the copper cut from the cake before it was rolled. Note also the small crystals. The copper as it came from the rolls was at a temperature of about 1,100° F., a temperature sufficiently high to anneal

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the sheet. In fact, a piece of this sheet was afterward rolled to 0.03 in. in thickness without further annealing. After trimming and cutting to suitable size for further rolling the $\frac{3}{16}$-in. sheet was scaled and pickled. Fig. 18 is a photomicrograph of the sheet after scaling and pickling, the magnification being the same as in the case of Fig. 17. The specimen was etched more particularly to show the eutectic areas. It will at once be seen that the structure has become very much coarser, which is due to the sheet having been heated to 1,450° F. when it was scaled. The

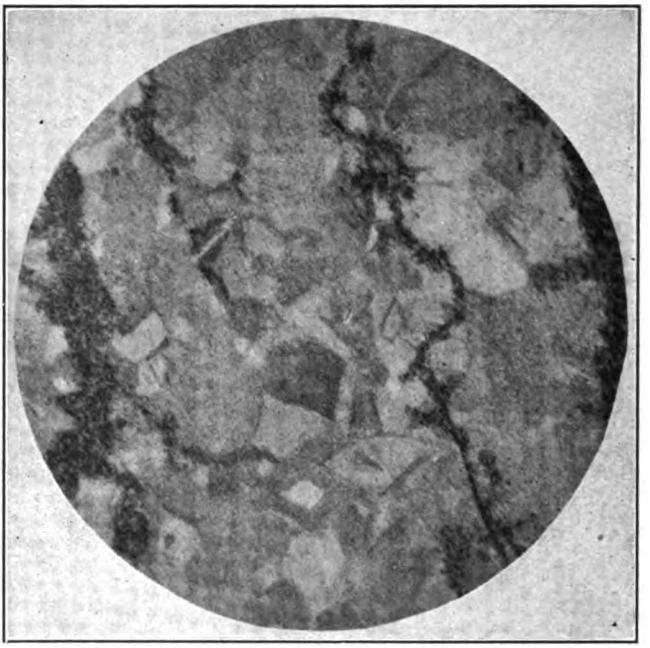

FIG. 18.—SHEET SHOWN IN FIG. 17 AFTER BEING SCALED. × 87.

eutectic has become very much segregated and the crystals are much larger. The crystalline structure is brought out more clearly in Fig. 19. The polysynthetic crystals mentioned in a preceding paragraph are in evidence. The $\frac{3}{16}$-in. sheet was now cold rolled to 0.033 in. The sheet rolled fairly well but cracked at the edges. Fig. 6 shows the structure of the sheet after being rolled to 0.033 in. After being rolled the sheet was annealed at a temperature of about 1,200° F. Fig. 20 is a photomicrograph of the annealed sheet. It will be seen that the eutectic structure has been entirely broken up. The photomicrograph shows a nearly complete absence of twin crystals, indicating that the annealing temperature was about right; in fact, the sheet was subsequently rolled to 0.005 in. gauge without developing defects of any sort. The presence of twin crystals in Fig. 19 shows that the sheet was overheated during

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the scaling process. Had the temperature been kept at about 1,100° F. instead of being allowed to reach 1,450° F. or thereabouts the metal would have been in much better physical condition for the subsequent reduction to 0.033 in. and the sheets would not have cracked at the edges.

The use of pyrometers for controlling the annealing furnace cannot be too strongly urged. In the ordinary reverberatory annealing furnace means should be provided for determining the temperature at several points in the furnace. Careful watch should be kept to see that sheets

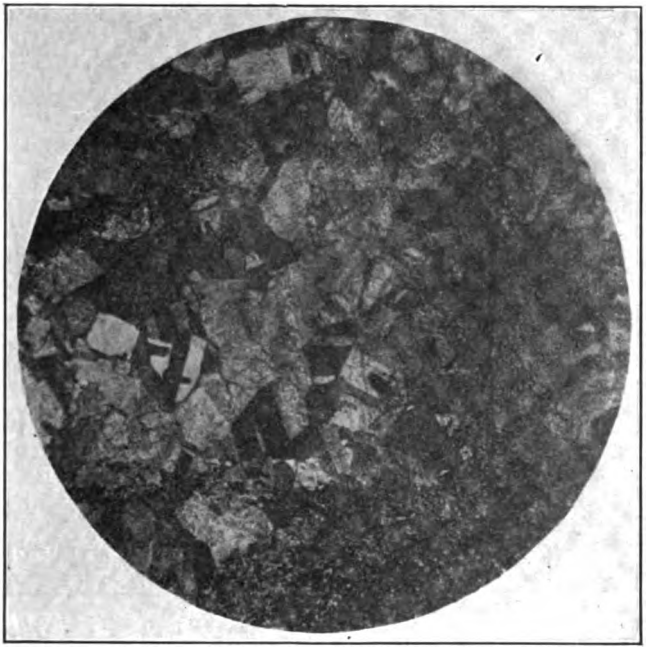

FIG. 19.—SAME AS FIG. 18, BUT ETCHED TO SHOW POLYSYNTHETIC TWIN CRYSTALS. × 87.

nearest the bridge wall are not being overheated. Pyrometers to be of use in controlling the annealing process should be frequently checked against a standard thermo-couple used for this purpose only, otherwise the readings may become misleading. Copper at 1,100° F. is at a dull red heat. If one looks into the furnace the sheet when at this temperature appears almost black, the reddish glow being only faintly visible. A full dull red indicates a temperature of 1,300° F., and a dull cherry-red, which I have observed in many annealing furnaces, indicates a temperature of 1,475° F. Optical pyrometers are very nearly useless for measuring temperatures as low as 1,100° to 1,200° F. Thermo-electric pyrometers unless frequently checked may give readings that are 150° to 200° too low.

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In these experiments the time factor has been neglected; not, however, because it has not been considered as of importance. Baucke[2] has shown that prolonged heating up to 450° C. yields the same results as heating at 1,025° C. for 10 min., and that heating at even as low a temperature as 100° C. for 350 days will result in incipient recrystallization. Results which might be obtained in the laboratory would not be applicable directly in practice. This much, however, may be said: exposure to an annealing temperature of 1,100° F. for 20 min. is sufficient to perfectly

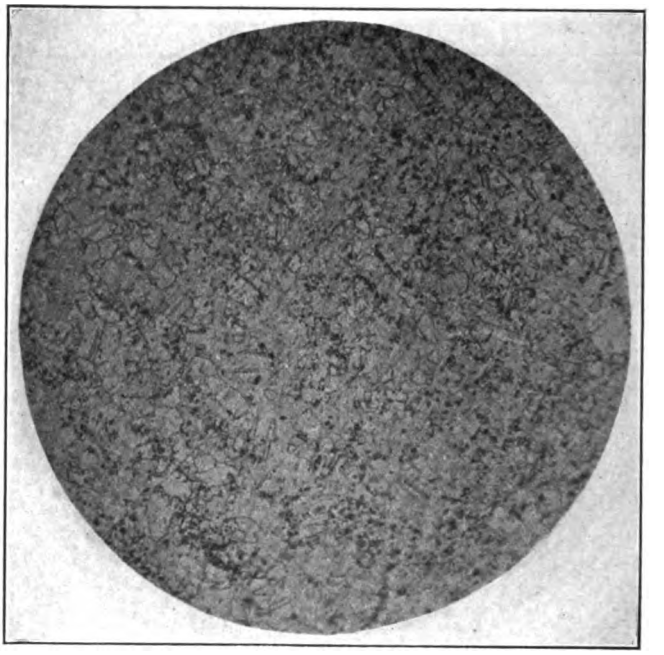

FIG. 20.—SHEET COPPER AFTER BEING COLD ROLLED TO 0.033 IN. AND ANNEALED. × 87.

anneal cold-rolled copper of any gauge up to $\frac{3}{16}$ in., it being understood that the copper shall attain this temperature and be held at this temperature for the stated period; and further, that exposure to this temperature for a period of an hour does not injure the copper. Where annealing is carried out at temperatures higher than 1,100° the time factor has a far greater effect.

I am well aware that it is impossible in a rolling mill to carry out the operation of annealing with the degree of perfection attainable in a laboratory experiment. I am, however, of the opinion that the experiments here described may be suggestive and helpful to those actively engaged in the operation of copper rolling mills.

[2] H. Baucke, Amsterdam: On Some Recent Micrographical Investigations of Copper, International Association for Testing Materials, vol. ii, No. II, 14.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Drumlummon Mine, Marysville, Mont.

BY CHARLES W. GOODALE, BUTTE, MONT.

With notes on other mines of the Marysville district by WALTER McDERMOTT, London, England, and F. L. SIZER, Dos Cabezas, Ariz.

(Salt Lake Meeting, August, 1914)

THE purpose of this paper is to review the history of the Drumlummon, one of the famous old mines of the West.

Mining engineers, when sent to examine new mines in old districts, or to decide whether an old property, under new conditions, will pay to reopen, have had difficulty in learning anything authentic regarding workings which are under water, and historical facts about the working character of the ore, the total production, and the value of the ore with increasing depth. Not many years elapse after the pumps are pulled, before maps, books and records are destroyed, and the value of the publications of the Institute would be increased if they contained the history of many old mines now abandoned. The unsuccessful efforts of the writer quite recently in trying to obtain reliable information about a gold mine which was reported to have produced several million dollars between 1880 and 1888, the workings of which are now partially caved and under water, impressed upon his mind the desirability of having a permanent record of old properties. In this particular case, the owners insisted that under present conditions of improved methods of treatment —particularly by the use of cyanide, which was not an applied process when the mine was operated, the property could be made to pay a handsome profit on ore left in sight in the old workings, but no assay records could be found to confirm these assurances, and even the records of bullion shipments had been destroyed. If surveys and maps were made as the work progressed, no trace of them could be found, and therefore opening of the ground through a new shaft and drifts therefrom would be attended by some risk while exploring near the old workings. Fatal accidents by flooding under these conditions have occurred, and as an evidence that miners have fears of such dangers, it was urged upon the Mining Committee during a recent session of the Montana Legislature, by a delegation of



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miners, that a law should be passed compelling mine operators to file maps of all workings in the office of the State Mine Inspector.

### *Location*

The Ottawa mining district is located in Lewis and Clark County, Montana, and Marysville, distant from Helena about 18 miles in a northwesterly direction, and having an altitude of 5,360 ft. above sea level, is the only town in the district. It is the terminus of a branch line of the Northern Pacific Railway, which leaves the main line at Clough Junction, 8 miles west of Helena. Silver creek, which has its rise at Marysville, runs through the district, and empties into Prickly Pear creek, 6 miles above the junction of that stream with the Missouri river. Beginning at Marysville and for 4 miles down Silver creek, the gravels of the valley were very productive in gold, which was discovered in 1862, but the richer bars were not found until May 1, 1864. During the prosperous days of placer mining there was a lively town called Silver City, near the lower end of the diggings, and it was of such importance at one time that the county records were kept there. In the meantime, Helena was growing in importance, and there was a county-seat rivalry between that town and Silver. It is related that in 1865 the matter was settled when Col. W. F. Sanders of Helena went up to Silver on horseback, took the county records, rode back to Helena, and depositing the books there, established a new county seat.

Records of yield from placer mines are very difficult to obtain, but it is probably safe to say that the Silver creek gravels produced $3,000,000. The gold was not of remarkable fineness (it would sell for about $14 per ounce); that is, it contained considerable silver, and, in view of the silver-bearing character of the ore from the principal vein of the district, the Drumlummon, this is not surprising.

The gold from Last Chance gulch, where Helena is situated, sold for about $17 per ounce, and as this gulch was producing before Silver creek, it is probable that the creek got its name from the more silvery character of its bullion yield.

### *History*

The Drumlummon lode claim was located by Thomas Cruse in 1876, although the ground had been covered by a previous location which had been allowed to lapse. Cruse had been working the placer mines of Silver creek, and after he had opened up some high-grade ore in the Drumlummon claim, he built a five-stamp mill in 1880, and this marked the beginning of prosperity for the town of Marysville. Other locations of lode claims were made by Cruse and by other prospectors as soon as the

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value of the Drumlummon discovery became known. Fig. 1 shows the claims in the vicinity, with the course of the principal veins.

In February, 1883, Cruse sold the Drumlummon and some small adjoining claims to an English syndicate, and the Montana Co.,

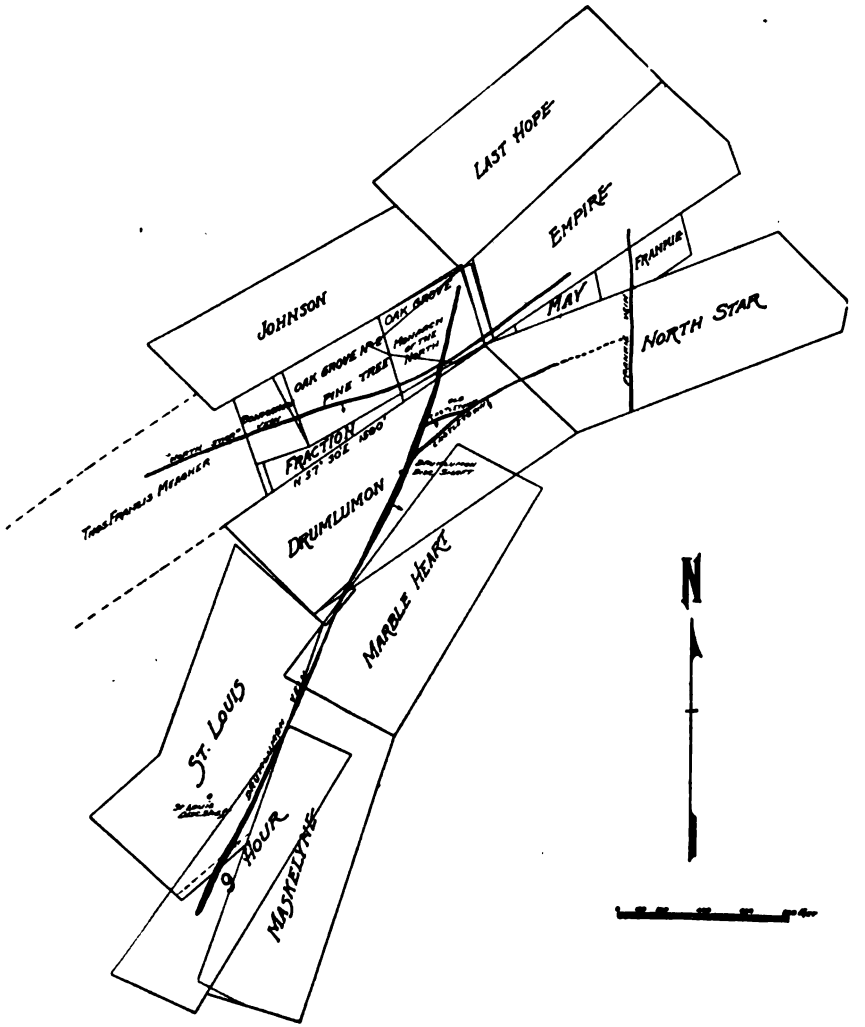


FIG. 1.—THE DRUMLUMMON AND SURROUNDING LODE CLAIMS.

Ltd., was formed under the laws of England, with 300,000 shares at £2 each. Cruse received $1,000,000 and shares worth $500,000. The report on which the mine was purchased was made by John Darlington, an English engineer, in November, 1882, and the workings at that time consisted of shallow prospect holes, a tunnel about 200 ft. long which had

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been run by Cruse single handed, and which cut the vein at a depth of about 140 ft., and 361 lin. ft. of development on the vein. The orebody, as shown in a crosscut 50 ft. from the surface, was 65 ft. wide, but further exploration showed that this crosscut was run at the junction of the Castletown vein with the Drumlummon. The mine had then produced 6,000 or 7,000 tons of ore, part of which had been reduced in arrastres, but 3,780 tons had been treated in a stamp mill, and had yielded $144,539 in bullion.

### *Development and Exploration*

The first work done by the company below the Cruse workings was to sink an incline shaft from the Cruse tunnel, and a new tunnel was started in 1883 at a point near the foot of the mountain. It reached the vein 260 ft. below the Cruse tunnel and 400 ft. below the "discovery" in a distance of about 1,260 ft., and was given the name "Maskelyne," in honor of the chairman of the board of directors. After exploring the vein for some distance northerly and southerly, a large excavation was made in the hanging wall of the vein at the end of the tunnel for a hoisting engine and headframe, and a new three-compartment incline shaft was started in the spring of 1886, having a pitch of about 68° from the horizontal. Two years later this shaft had reached a depth of 800 ft., and the vein had been opened up on the 500, 600, 700, and 800 levels, but ore values had decreased rapidly as depth was gained, and in the spring of 1888 Prof. Joshua E. Clayton was employed by the company to make a thorough geological study of the mine. In his opinion: "The impoverishment of the ore shoots below the 400 level is due to local causes rather than to permanent ones, where surface influences could not permanently affect the lode in so short a depth. . . . There is no geological reason for the lode becoming poor in so short a distance from the present surface, nor is there any chemical cause apparent for such a change in the metal contents of the lode. My conclusion is that the causes are purely mechanical and local, and not due to the original structure or other primal cause."

Sinking was resumed soon after, and by the close of 1891 the shaft, called No. 1, had reached a depth of 1,600 ft., with considerable development on the 1,000, 1,200, and 1,400 ft. levels. Connection had also been made on the 1,200-ft. level with No. 2 shaft, which was started in 1887 from the 400-ft. level, 700 ft. south of No. 1.

In the meantime the 400-ft. level had been driven several hundred feet south, and had discovered the 9-Hour shoot. Exploration of this orebody below the 400-ft. level was by means of No. 3 shaft.

From 1883 to 1910, when the ownership of the mine passed from the Montana Mining Co., Ltd., to the St. Louis Mining & Milling Co., the development and exploration work amounted to 123,500 ft., and as the

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mine produced 1,150,000 tons of ore, the quantity per linear foot was not large—only 9.3 tons. This would indicate one of three conditions: First, that the veins were small; second, that the orebodies were widely separated; or third, that a great deal of unproductive work was done below and beyond the pay ore. The last condition applies to the Drumlummon, and it is doubtful if in any mine in the world the ground has been so thoroughly prospected.

*Geology*

This subject has been thoroughly covered by the U. S. Geological Survey[1] and will not be taken up at great length in this paper. Figs. 2, 3, 4, and 5 have been copied from Mr. Barrell's paper.

Marysville is situated in an amphitheater, with high mountains rising all around it, except on the east, where Silver creek flows out. Quartz-diorite is the bed rock of this basin, and extends up the mountain sides for several hundred feet above Marysville on all sides. Overlying the diorites are the hornstones and slaty shales, and at a point about 400 ft. above the town and southeast of it, is the "discovery" of the Drumlummon vein. This great lode, with a strike of about N. 15° E. and an eastward dip of 65° to 80°, has been opened up for a length of more than 3,000 ft. and a depth of 1,600 ft. Practically, it is entirely in hornstone, the composition of which is shown in Table I; but as it is near the hornstone-diorite contact, which has a very jagged outline, the diorite appears as vein walls at many points, where tongues of this rock extend eastward from the main body.

TABLE I.

| | Barren Vein Matter 1 to 3 ft. wide Per cent. | A Foot-Wall Hornstone Per cent. | B Hanging-Wall Hornstone Per cent. |
|---|---|---|---|
| $SiO_2$..... | 23 | 58.2 | 62.5 |
| FeO..... | 2.7 | 3.4 | 3.4 |
| CaO..... | 33.7 | 7.9 | 5.2 |
| MgO..... | 5.2 | 1.8 | 2.3 |
| $Al_2O_3$..... | 4.6 | 18.0 | 18.5 |

Fig. 2 shows the position of the vein in reference to the contact. Below the 400-ft. level the grade of the ore decreased rapidly in value, and while considerable tonnage was extracted below the 1,000-ft. level, it is doubtful if the bullion yield returned a profit over mining and milling cost, beyond that depth. On the 1,600-ft. level, where there is nearly a mile of exploration drifts and crosscuts, the vein which had yielded so well in upper levels gave assays of only a dollar or two in gold

[1] Joseph Barrell: Geology of the Marysville District, Montana, *Professional Paper No. 57, U. S. Geological Survey* (1907).

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and silver from selected pieces. The ore occurs in distinct shoots, which have a decided trend or rake to the south. This is clearly shown in Fig. 6, with the names given to the various shoots by the management, and it will be noted that two of them do not extend to the surface—the Jubilee and Jubilee No. 2.

In width the lode varies from 15 to 25 ft. in its widest and productive portions, to less than an inch at one observed point in its pinched and barren region. For hundreds of feet along its strike it shows between well-defined walls from 1 to 3 ft. of soft material, almost free from quartz and of the character which is shown in Table 1.

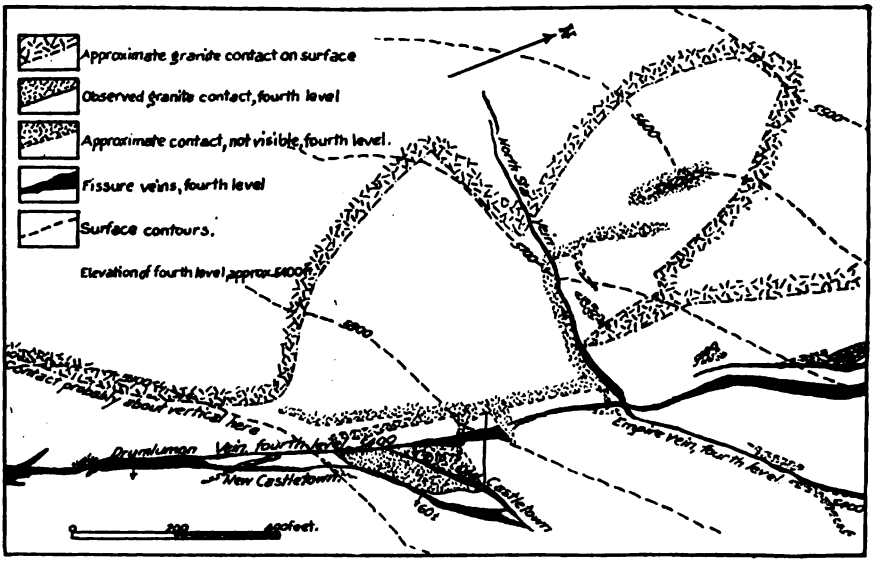


FIG. 2.—APPROXIMATE RELATIONS OF GRANITE BOUNDARY BETWEEN THE SURFACE AND THE FOURTH LEVEL, DRUMLUMMON MINE.

In *Bulletin* No. 213 of the U. S. Geological Survey, W. H. Weed gives a good description of the Drumlummon vein, from which the following quotation is made:

"It is a fault plane with white opaque quartz inclosing angular fragments of black, green and drab slates, which are sometimes distinct and unaltered and at others have been much decomposed. Where the ore-bodies are found the replacement has been complete and the former presence of the fragments is only recognizable by the outlines of the banded quartz. . . . . This vein, which is the largest and the most productive in the district, consists in its lower levels of a mass of angular rubbish, derived from the walls of the fissure, and in places cemented by quartz, in other places still retaining its original character. . . . . It may also be said that the ore shoots were well defined

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and the intervening vein matter barren and unworkable. The pitch of the ore shoots conforms to the usual habit, dipping to the right when looking down the dip of the vein."

The walls are usually indistinct in the productive parts of the lode, but in the barren portions they are well defined. T. A. Rickard in his paper on Vein Walls[2] gives some interesting notes regarding the walls of the Drumlummon vein, and he also comments on the fact that the vein is only productive where it crosses the strata of the slates at an oblique angle, being unproductive where the crossing is at, or near, right angles.

The Castletown lode has a strike about N. 40° E. and dips to the eastward at about the same angle as the Drumlummon. It joins this vein on the hanging-wall side in the "No. 1 South shoot." The walls are slate, or hornstone.

In 1893 an important discovery was made when, after a careful examination of the Castletown workings in and above the 2A level, it was decided to drive a crosscut into the hanging wall of that vein on the No. 3 level. At a distance of about 40 ft. a new orebody was found, which was soon opened up on all levels down to the 1,000, and yielded a large tonnage of good ore (Fig. 7). When fully developed it was found that this vein, which was given the name New Castletown, united with the Castletown on strike to the northeast, and formed a junction on its course southerly with the Drumlummon lode in the Sampson shoot (Fig. 2). The discovery of this new orebody led to further hanging-wall explorations, and one crosscut 210 ft. in length was run into the hanging wall of the New Castletown on the 400-ft. level, but without result.

The North Star vein, with a strike of. N. 80° E. and nearly vertical dip, was very productive and yielded pay ore to a greater depth than the Drumlummon, or to the 1,200-ft. level (Fig. 8). It was given this name in the early history of the mine when it was supposed to be the same vein as the one discovered in the North Star claim, but later developments proved that the latter vein was the same as the Castletown.

The relation of the so-called North Star vein to the others of the system is shown in Fig. 3. In the professional paper by Joseph Barrell, previously mentioned, he says: "The other important vein is the North Star and its continuation, the Empire. This cuts across the Drumlummon and on the seventh level, where the relations are well exposed, it is seen to be younger than the Drumlummon, but has not faulted the older vein, which is here rather poorly developed, consisting of a mesh of quartz veinlets in brecciated slates.' This statement is in conflict with views held by many geologists who have studied the mine; for at the intersection of the two veins on the 400-ft. level, there are evidences that the Drumlummon faulted the North Star, although it must be admitted that their

[2] *Trans.*, xxvi, 193 (1896).

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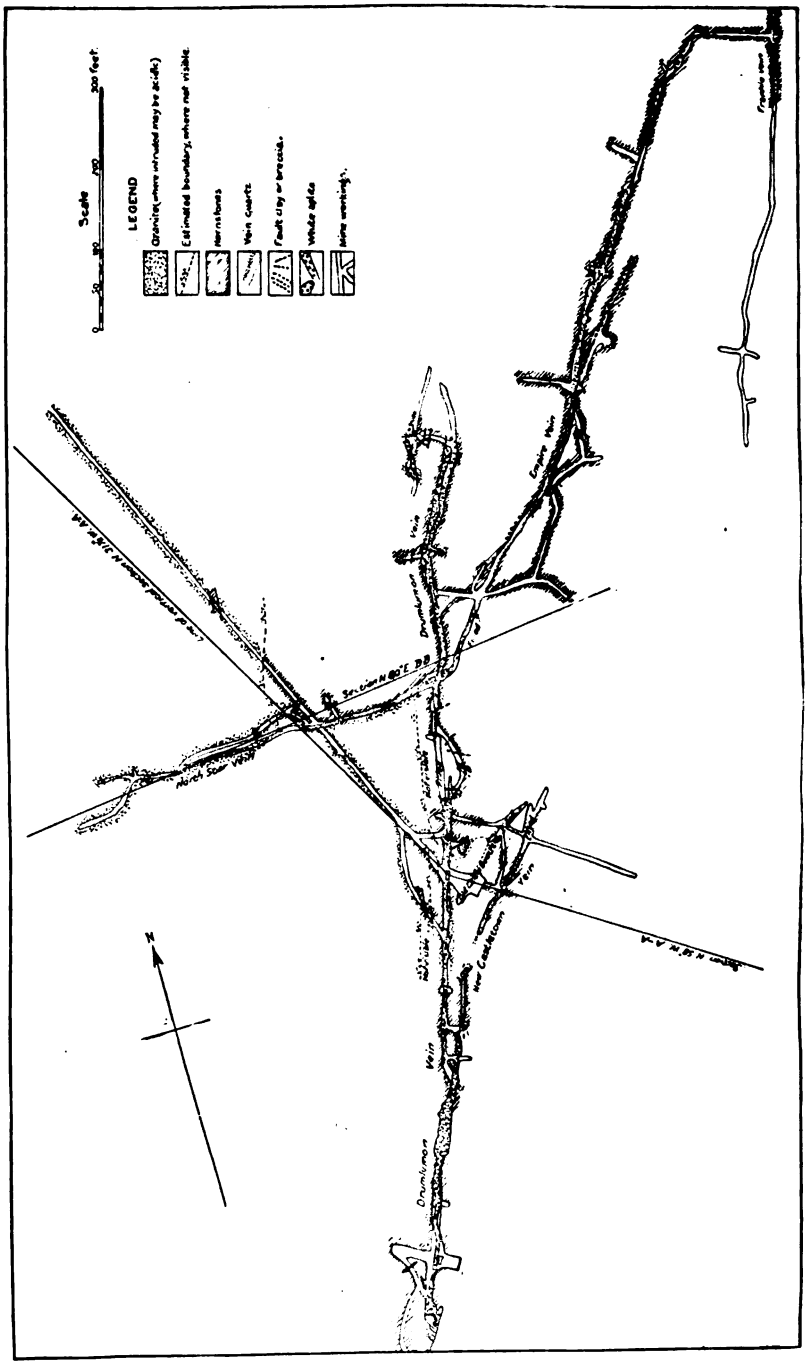


Geology by R. W. Stone, 1901.

FIG. 3.—GEOLOGIC MAP OF FOURTH LEVEL, DRUMLUMMON MINE.

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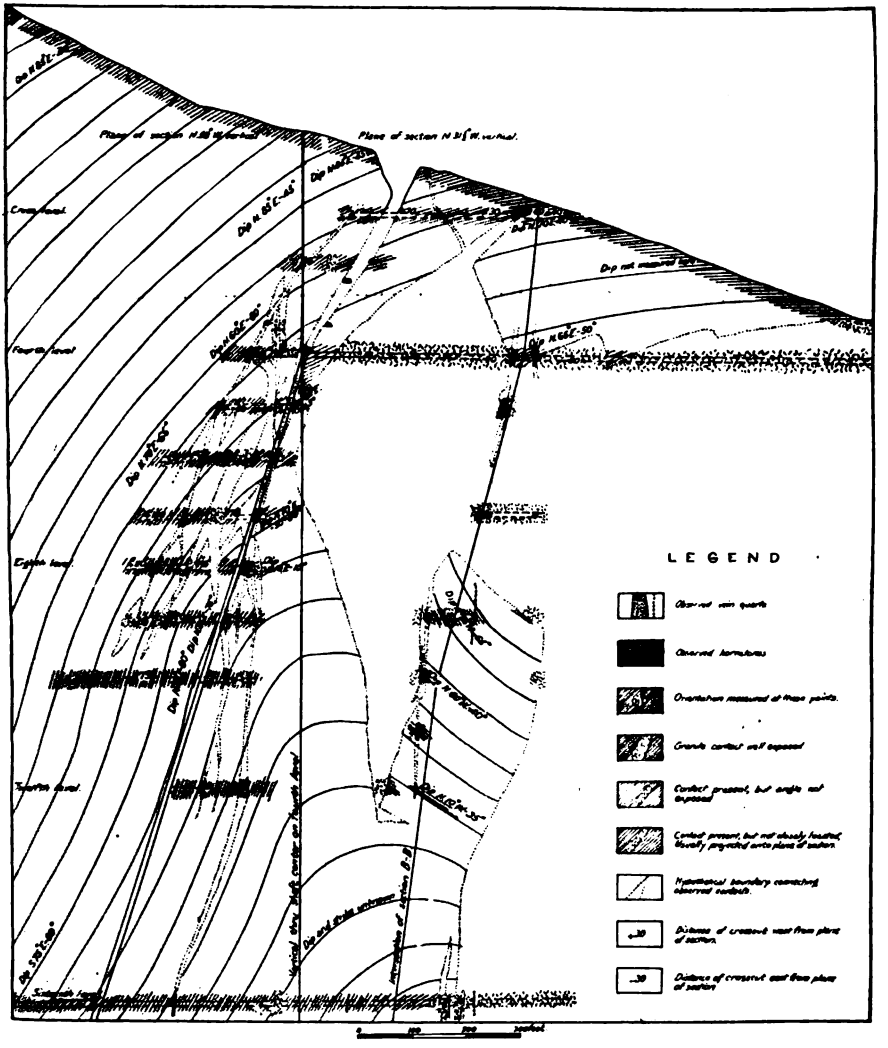
attention was never called to the condition on the 700-ft. level, which led to Mr. Barrell's conclusions. There is some foundation for the belief that the North Star does not extend beyond or into the hanging-wall of




Section through main tunnel and shaft, looking southwest. Planes of section, N. 58° W. and N. 31½° W.

FIG. 4.—VERTICAL SECTION, DRUMLUMMON MINE.

the Drumlummon, and that the Empire vein, like the Castletown, joins the Drumlummon on its hanging-wall side; the occurrence of the two veins in the walls of the Drumlummon at points nearly opposite each other being only incidental. Supporting this view are two facts: First that the strike of the Empire (N. 35° E.) is quite different from that of the North Star

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(N. 80° E.), and is more inclined to be parallel with the Castletown. Second, mineralogically, the Empire resembles the Castletown and the Drumlummon, and is not at all like the North Star. The ore extracted from the latter contained very little silver, and the only sulphide mineral observed was pyrite, while from the former, high-grade silver minerals were obtained, and also some copper in combination with sulphur, antimony, and arsenic. In the oxidized portions a good grade of ore in the Drumlummon, Castletown, and Empire veins was indicated by the presence of carbonate of copper, and some fine specimens of azurite were found. Perhaps the difference above noted will sustain the contention that the veins were not of the same age, but if so, which is the older? If it is argued that the mineral character of a vein is influenced by the nature of the country rock in which it occurs, and not by its age, why was there no change mineralogically in the North Star when it passed out of diorite into hornstone or slate? The ratio of gold to silver in the North Star ore was about 1 to 3 in ounces, and this held true where the vein had slate walls, as well as where the inclosing rock was diorite, while in the other veins above mentioned the ratio varied from 1 : 10 to 1 : 20. It is worthy of note that nowhere along the contact between the diorite and hornstone were bodies of ore found.

The discovery of pearceite, an arsenical polybasite, in the Drumlummon mine is interesting in this connection. In 1895, Dr. Richard Pearce was shown some beautiful crystals of this mineral which had been found in the 500-ft. stopes of the New Castletown vein, and sent a specimen to Prof. S. L. Penfield of Yale, who gave a full description to the Colorado Scientific Society.[3] The analysis of the specimen was as follows:

| | Per cent. |
|---|---|
| Sulphur | 17.71 |
| Arsenic | 7.39 |
| Silver | 55.17 |
| Copper | 18.11 |
| Iron | 1.05 |
| Insoluble | 0.42 |
| | 99.85 |

Professor Penfield says:

"It cannot claim to be a *new mineral*, for as an arsenical variety of polybasite it has previously been recognized, although no special name has been assigned to it. . . . . While recognizing that antimony and arsenic are isomorphous and may mutually replace one another, it is customary and has been found convenient in mineralogy to consider the sulphantimonites and sulpharsenites as distinct species, and to designate

[3] "On Pearceite, a Sulpharsenite of Silver, and on the Crystallization of Polybasite," *Proceedings Colorado Scientific Society*, vol. v, p. 210 (1894–1896).

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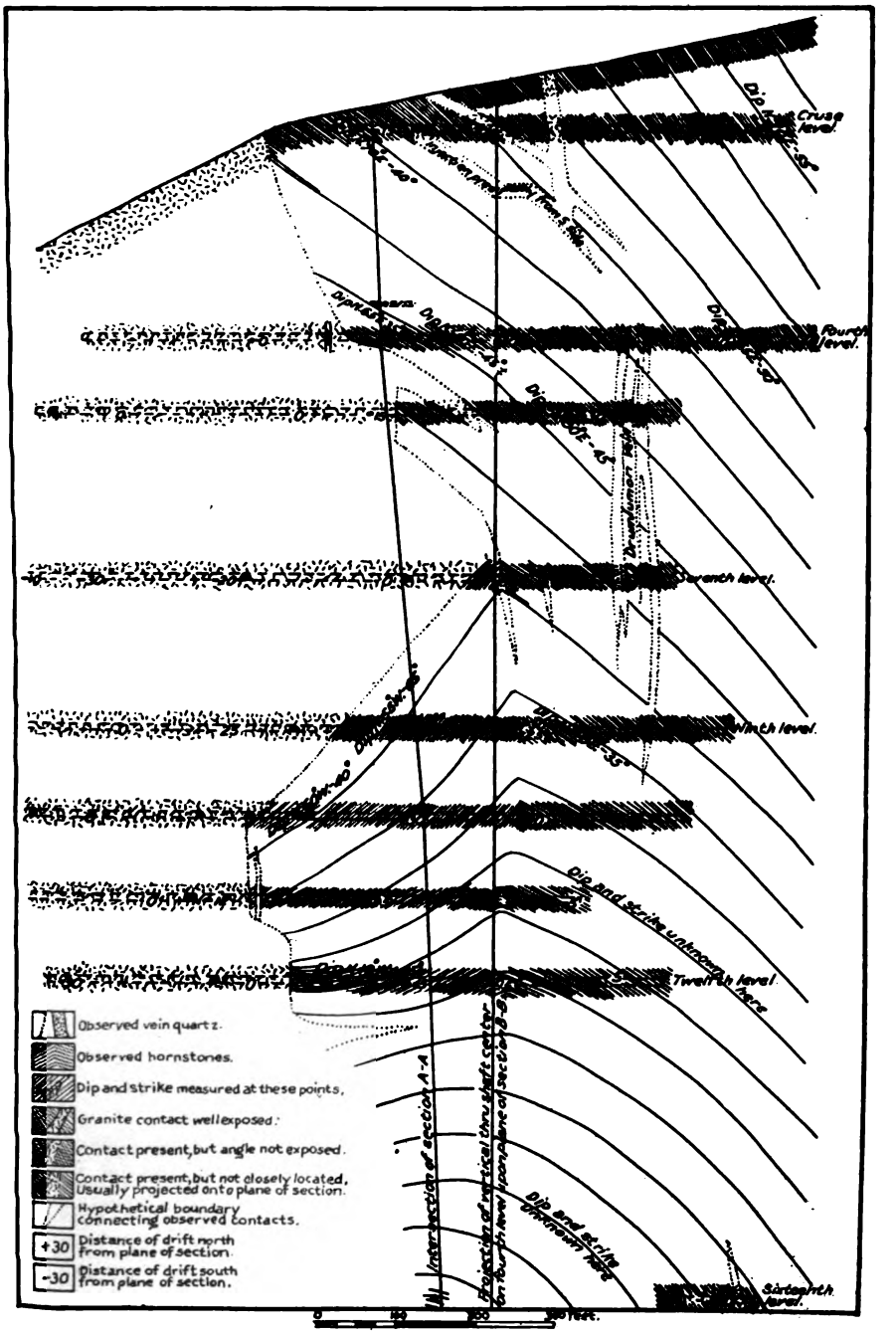
Observed vein quartz.
Observed hornstones.
Dip and strike measured at these points.
Granite contact well exposed.
Contact present, but angle not exposed.
Contact present, but not closely located, usually projected onto plane of section.
Hypothetical boundary connecting observed contacts.
+30 Distance of drift north from plane of section.
-30 Distance of drift south from plane of section.
Intersection of section A-A
Projection of vertical line on earth's surface on the new plane of section B-B
Dip and strike unknown here
Dip and strike unknown here
Fourth level
Twelfth level
Sixteenth level
0
100
200
300 feet.

Section parallel to North Star vein, looking north. Plane of section, N. 30° E.

FIG. 5.—CROSS-SECTION, DRUMLUMMON MINE.

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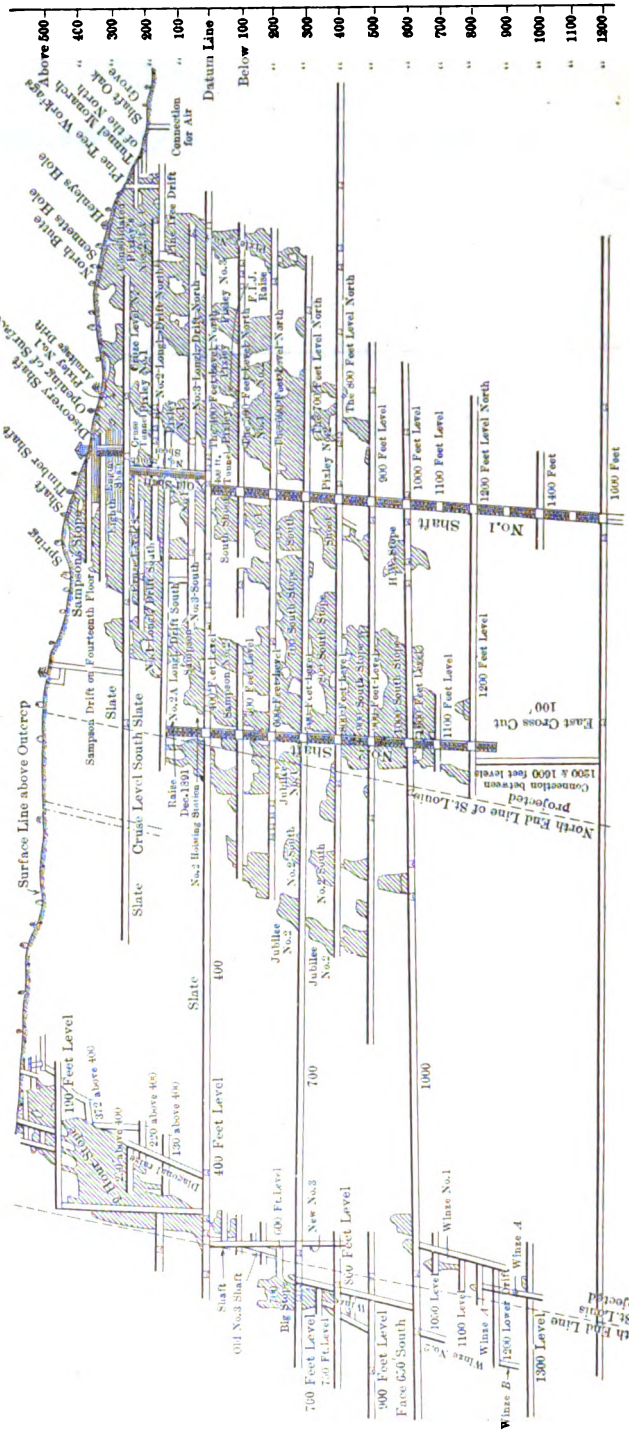
Above 500
400
300
200
100
Datum Line
Below 100
200
300
400
500
600
700
800
900
1000
1100
1200
Surface Line above Outcrop
Shaft
Sprint
Timber Shaft
Discovery Shaft
Opening of Shaft
Pipe Tree Hole
Tunnel Monarch
Shaft Oak
Grove
Connection for Air
Five Tree Drift
Sampson Stope
Fourteenth Floor
Sampson Drift on
Slate
Crude Level South
Slate
No.2A Long Drift South
No.2 Long Drift South
No.2 Long Drift North
Jubilee No.2
Jubilee No.2
Jubilee No.2
Priley No.2
Priley No.2
Priley

them by different names, and the author proposes that hereafter the name polybasite shall be restricted to the antimony compound $Ag_9SbS_6$, and to make of the corresponding arsenic compound $Ag_9AsS_6$ a distinct species. For the arsenical mineral he takes pleasure in proposing then name *pearceite* as a compliment to his friend, Dr. Richard Pearce, of Denver, whose keen interest in mineralogy and connection with one of the large smelting and refining works of Colorado have made him known both to scientific men and to those interested in the development of the mining industries of the Rocky Mountain region."

Some beautiful crystals of quartz replacements of calcite were found in a large vug in the 900-ft. workings of the North Star vein, where the walls were slate. Photographs were taken of the best specimens before they were sent to the Smithsonian Institute, Washington (Fig. 9).

## *Mining*

In developing the mine practically no timbering was required in drifts and crosscuts, as the hornstone was very hard and there was very little

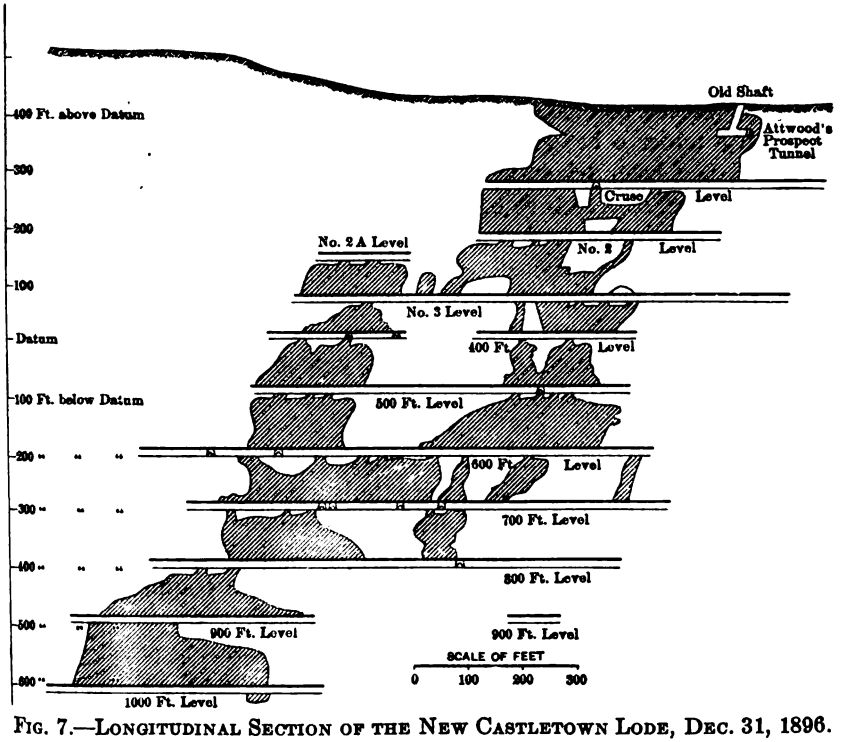


FIG. 7.—LONGITUDINAL SECTION OF THE NEW CASTLETOWN LODE, DEC. 31, 1896.

disintegration on exposure to the air. In the orebodies, after putting in stulls over the drifts and placing the chutes at proper intervals, stoping was continued without timbering, only enough ore being drawn out to

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allow room for ore breaking, and therefore in the most productive era of the mine there was a large tonnage of broken ore in the stopes, the value of which was known from daily samples taken from the faces of the workings. Square sets were used in only one orebody in the mine, the 9-Hour, where the walls were treacherous, and the excavations had to be filled with waste as the stoping progressed.

In 1890, when new ground was being opened up rapidly in depth, it was apparent that provision should be made for a pumping plant of large capacity, and a Cornish pump was purchased, at an outlay of about $55,000, including partial installation at No. 2 shaft, where a large chamber was excavated and timbered in the hard hornstone. As the ground in deep levels drained down, the flow of water diminished greatly, and the installation of the Cornish pump was never completed, but in 1895 a Riedler, having a capacity of 400 gal. per minute, was placed in the 1,600 station of No. 1 shaft, and a great saving was made in draining costs, as this equipment replaced a system of pumps on the 700, 1,000, 1,200, and 1,600 ft. levels. The pump ran at only about one-third capacity at first, but extensive exploration in the 1,600-ft. level during the next two years increased the volume of water so greatly that it was considered advisable as a matter of safety to install another Riedler. This was a larger pump, its capacity being 500 gal., but it easily handled the entire flow, 468 gal. per minute under a head of 1,230 ft. Before this pump was installed, a crosscut had been driven westerly more than 700 ft. for the purpose of developing the North Star vein, and as the flow of water became somewhat alarming, a concrete dam was placed in the crosscut, so that the duty of the pump could be controlled. By the time the new Riedler was ready to run, the pressure at the dam went up to 264 lb. per square inch, representing a head of more than 600 ft.

The hoisting engine at No. 1 shaft was of the first-motion type, arranged to run single or in balance, and its cylinders were 20 in. in diameter by 60 in. long.

No. 1 and No. 2 shafts, in which cages were used, were of the same size, 14 by 4½ ft. in the clear, and were timbered with 10 by 10 in. Oregon fir, which cost $20 per thousand feet at the mine. The sets were 6 ft. apart, each set containing about 1,410 sq. ft. of timber, and the cost per foot for timbers was about $6.50, including labor of framing—the total cost averaging about $100 per foot. In explanation of this high cost it should be stated that it included the cost of excavating and timbering the stations, and that the rock was extremely hard and very difficult to break, owing to the angle at which the shafts crossed the stratification. Adolph Knopf calls the rock wollastonite and diopside,[4] but the analyses in Table I do not support this classification.

[4] Ore Deposits of the Helena Mining Region, Montana, *Bulletin No. 527, U. S. Geological Survey* (1913).

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On May 8, 1892, a fire started in the 1,200 station of No. 1 shaft, and as the draft was strong through the level from No. 2 shaft, the fire raced up through the timbers with great rapidity. Before an adequate supply of water could be poured into the shaft, the flames had reached the 400 station, and 800 ft. of timbering was practically all destroyed. The fire was discovered by the night shift going into the mine. Hose lines were run into the tunnel from the fire pumps in the boiler room, and by stretching a wet blanket across the tunnel and pushing it ahead, the men were

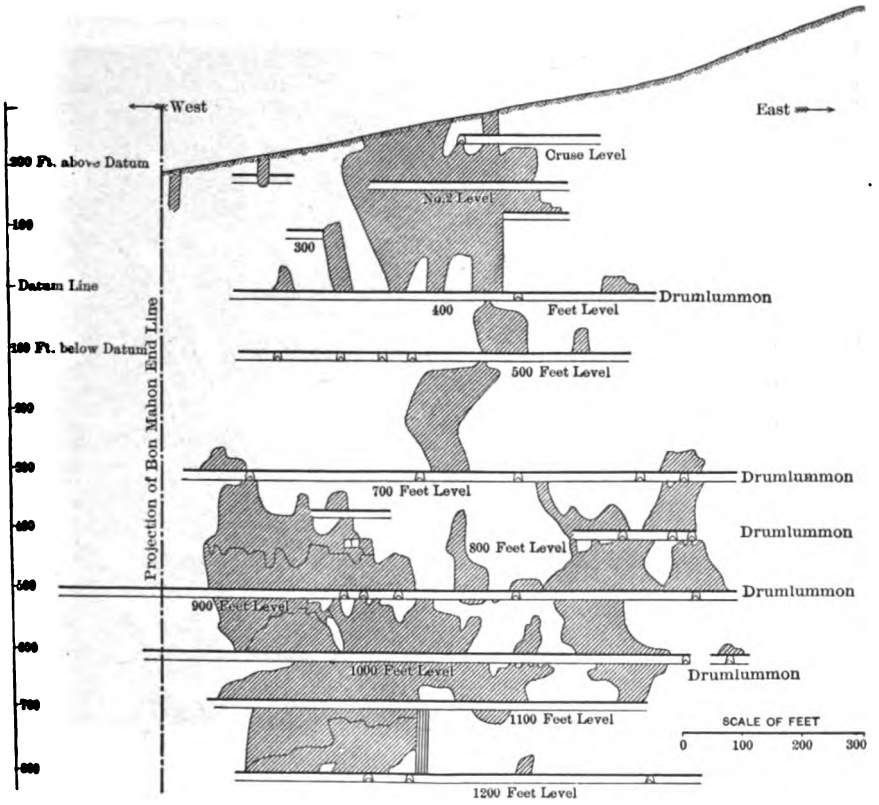


FIG. 8.—LONGITUDINAL SECTION OF THE NORTH STAR VEIN, DEC. 31, 1900.

able, several hours after the fire started, to get close enough to the shaft to put water into it, and also to save the station timbering and head frame. Not less than 5,000,000 gal. went into the shaft and the water rose to the 700-ft. level. The difficulty in extinguishing a fire in an incline shaft will be readily understood, for the water thrown into the shaft could not be effectively used. Nothing was done toward retimbering the shaft until the early part of 1893 and after the reorganization of the company. Owing to interruptions based on financial conditions, the repairs were not

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completed to the 1,600 level until the middle of 1894. The burning timbers above the 1,200 had fallen down below that level, and the shaft was badly damaged all the way down to the 1,600 station. Furthermore, much of the timbering in the stations above the 1,200 had to be renewed, and the removal of the tangled and twisted pipes, which carried air, water, and steam, increased the cost of repairs; but as the shaft was in hard slate all the way, there were no serious caves. The origin of the fire was never known, but afterward while the superintendent was waiting in one of the stations for some blasts to go off, he noticed that the heavy

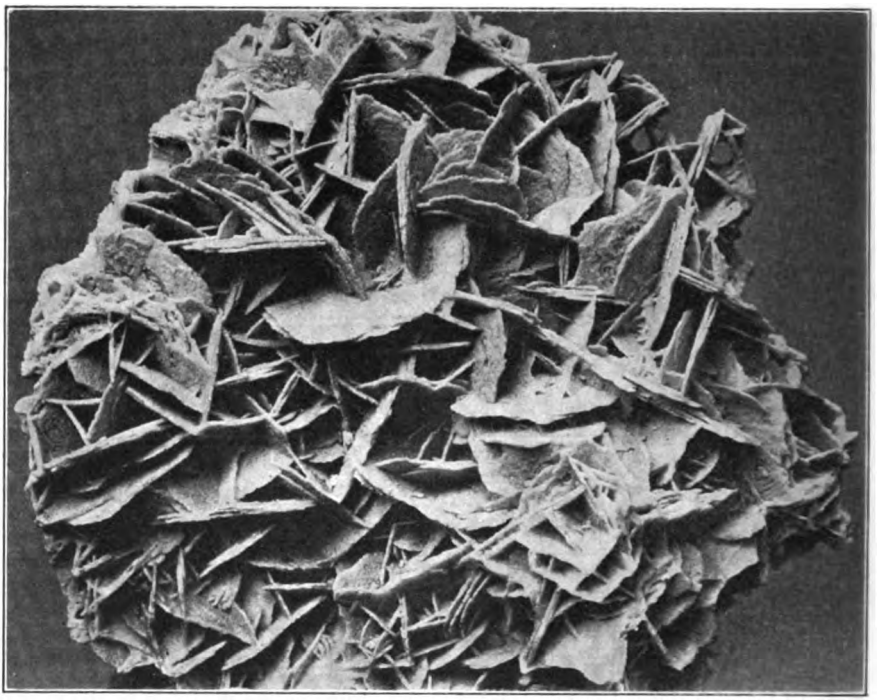

FIG. 9.—CRYSTALS OF QUARTZ REPLACING CALCITE.

concussions caused movement in a defective splice of the electric-light wires, and a fire started in some dry timbers just over the wires—so it was always regarded as quite possible that the fire in No. 1 shaft started in a similar manner.

The cost of drifting and crosscutting varied from $10 to $14 per foot, and the total mining cost per ton of ore was from $3.50 to $4, of which about $1.50 was for development work.

### *Milling*

The first mill for the treatment of Drumlummon ore was erected by Thomas Cruse in 1878, and contained five stamps. Shortly after the

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purchase of the property in 1883 by the English company, another battery of five stamps was added, and two amalgamating pans and a settler. In 1884 a 50-stamp mill was completed and provision was made for extracting the rich sulphides by treating the battery pulp on Frue vanners after it left the amalgamating plates. The feed boards of the vanners also carried amalgamating plates, and the mill was equipped with pans and settlers.

With the ore carrying about ½ oz. gold per ton, and from 7 to 12 oz. silver, the bullion from the plates was of about the following fineness:

| | Gold | Silver |
|---|---|---|
| Battery plates | 540 | 440 |
| Vanner plates | 380 | 600 |

There was a notable accumulation of amalgam on the plates.[b]

Two years later a 60-stamp mill was erected and began operations in November, 1886. It had two vanners for each battery of five stamps, but no pans and settlers, the duty of this mill having been to treat the lowest-grade ore from the mine, and particularly that low in silver. The stamps weighed 750 lb., the drop being from 7 to 9 in., 96 times per minute, the screens were of 30-mesh steel wire, and the average daily duty was 1.9 tons per stamp. The cost of this mill was $128,340.

In the years when the mine was making its best output the yield of the ore was about 1 ton of concentrate from 135 tons of ore, but as this product was very rich, carrying from $300 to $600 per ton in gold and silver, it contained about 22 per cent. of the values recovered from the ore.

The cost of treatment was about $4 per ton in the 50-stamp mill, and $1.15 in the 60-stamp mill.

### *Cyaniding*

In 1896, under the direction of C. W. Merrill, tests were made in a small experimental plant, with a view to working the tailing from the mills by cyanide treatment, and the results were so favorable that a plant was built the next year, at a cost of about $66,000, and having a capacity of 400 tons per day. The equipment included several miles of railway of 36-in. gauge, with Porter locomotive and cars, the tailing having been impounded by five dams located along Silver creek for about 4 miles. Additions were made to this equipment a few years later,

[b] *Trans.*, xxvi, 33 (1896).



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involving an expenditure of about $38,000, and by the close of 1907 all the tailing had been treated, amounting to 824,570 tons.

| | Total | Per Ton |
|---|---|---|
| Value of product | $1,954,050.78 | $2.37 |
| Expenses | 1,111,106.57 | 1.35 |
| Profit | $842,944.21 | 1.02 |

| | Tailing | | Recovery | | Residue | |
|---|---|---|---|---|---|---|
| | Total Contents | Per Ton | Total | Per Ton | Total | Per Ton |
| Gold, ounces | 113,178 | 0.137 | 80,195 | 0.097 | 32,983 | 0.04 |
| Silver, ounces | 1,424,513 | 1.728 | 682,400 | 0.828 | 742,113 | 0.90 |

Percentage of recovery: gold 71; silver 48 per cent.

The average cost per ton was $1.35, but this included a charge of $0.125 for redemption of the capital invested in the plant. A summary of the costs per ton for the year 1904, when 95,490 tons were treated, follows:

| | | | |
|---|---|---|---|
| Loading | | | $0.15 |
| Transportation | | | 0.10 |
| Impounding | | | 0.02 |
| Supplies: | Cyanide | $0.375 | |
| | Zinc | 0.060 | |
| | Lime | 0.025 | |
| | Fuel | 0.115 | |
| | Sundries | 0.105 | |
| | | | 0.68 |
| Operating labor | | | 0.20 |
| Superintendence | | | 0.04 |
| Total | | | $1.19 |

Composition of average precipitate:

| | Per cent. |
|---|---|
| Gold | 2.27 |
| Silver | 15.00 |
| Copper | 6.70 |
| Zinc | 38.00 |
| Calcium carbonate | 25.00 |
| Insoluble | 1.10 |

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*Production*

The best year of the mine was 1887, when the yield of the ore averaged $27.21 per ton, and dividends amounting to nearly $920,000 were paid, but the grade of the ore did not keep up as depth was gained, and in 1892 a combination of misfortunes—the fire in No. 1 shaft, a flood, a bad cave in the 9-Hour stopes, and expensive litigation, made reconstruction of the company a necessity. This was effected Dec. 15, 1892, and a new name was adopted, the Montana Mining Co., Ltd.

The following table shows the production of the Montana Co., and of the Montana Mining Co., up to the time when the property passed to the St. Louis Mining & Milling Co.

| | Ore Tons | Tailing Tons | Total Net Yield | Yield per Ton of Ore | Dividends |
|---|---|---|---|---|---|
| 1883–1892 Montana Co..... | 575,809 | | $8,498,794 | $14.76 | |
| 1883–1892 Montana Co..... | | 14,925*a* | 59,849 | | £537,057 |
| 1893–1910 Montana Mining Co. | 572,641 | | 4,719,131 | 8.24 | |
| 1893–1910 Montana Mining Co. | | 14,975*a* | 34,000 | | |
| 1893–1910 Montana Mining Co. | | 824,570*b* | 1,954,051 | | 90,355 |
| | 1,148,450 | | $15,265,825 | | 627,412 |

*a* Tailings treated in amalgamating pans.
*b* Tailings treated in cyanide plant.

| Total Production | Oz. |
|---|---|
| Gold..... | 568,898 |
| Silver..... | 4,982,942 |

## The Penobscot and Belmont Mines

BY WALTER MCDERMOTT, LONDON, ENGLAND

Mr. Goodale has asked me for some notes on the other mines of the Marysville district, particularly the Penobscot, but it is more than 30 years since I was connected with that mine and the Belmont, and I am afraid I can contribute very little of interest. The following is mostly taken from old reports in my possession:

The Penobscot was located in 1872 by Nate Vestal, and created a great excitement a few years later from development of very rich ore near the surface. The first ore was worked in two small arrastres and a little five-stamp mill on Silver creek, a mile and a half below the mine. With this equipment $80,000 had been produced from ore yielding $20 a ton as an average.

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In 1878 a 14-ft. vein of good ore was found. On the strength of this find the mine was purchased for $400,000 by the Penobscot & Snowdrift Consolidated Mining Co., the capitalization being $500,000. William Frue, inventor of the Frue vanner, with friends in New York, raised the capital for purchase and equipment, and controlled the operations for some time; he also purchased the Belmont mine.

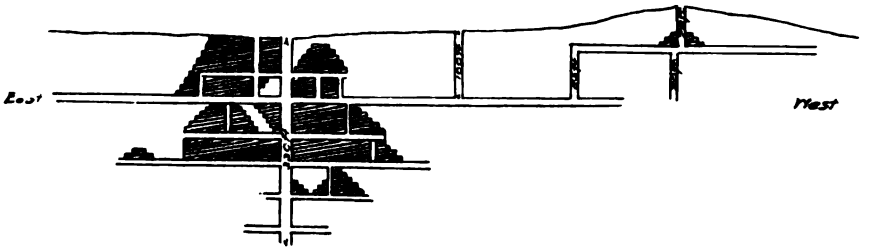


FIG. 10.—LONGITUDINAL SECTION OF PENOBSCOT MINE, MAR. 15, 1880.

The first ore taken from the big vein was very rich, 685 tons yielding $80,798 in the first five months of 1878. Mining and milling costs were about $10 a ton. This ore was treated in the arrastre, and the tailings from it assayed 1 oz. in gold and 2.05 oz. in silver. Some samples assayed from vein widths of 6 ft. yielded better than $1,000 gold per ton. The average silver content was about 10 oz.

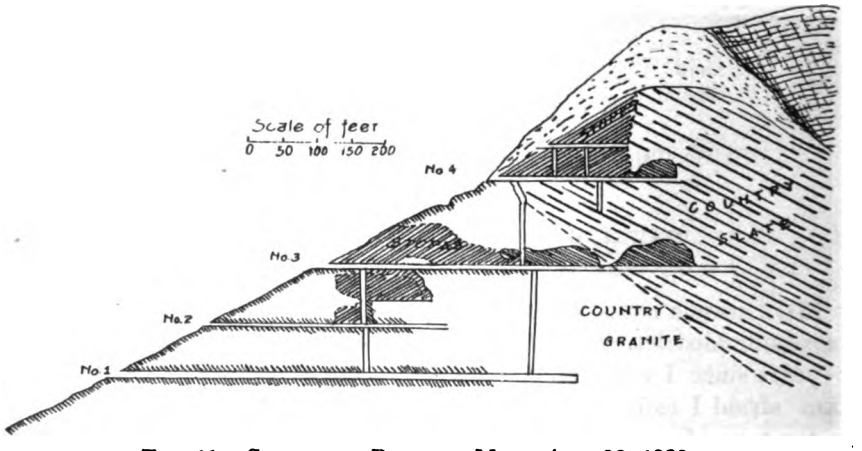


FIG. 11.—SECTION OF BELMONT MINE, APR. 22, 1880.

Reports made at the time the property was purchased were very flattering. The average assays of the workings were given as $30 gold and 3 oz. silver per ton.

In spite of this encouraging start, the results proved disappointing to the stockholders. The value did not persist in depth and the ore shoots

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proved to be very erratic and pockety. The Bonanza shaft was sunk to a depth of 330 ft. but yielded unsatisfactorily below a depth of 160 ft. In addition to erratic values, the vein widths also varied rapidly. The vein was in silicified shale near a granite contact. Its strike was northeast-southwest and the dip was 75° to 80°.

During the period from September, 1878 to January, 1880, 14,307 tons of ore were mined and milled, yielding $251,661 in bullion. The expenses for that period, outside of purchase of mine, were $268,513, showing a loss of $15,852. From the above, the average bullion yield per ton is seen to have been $17.59.

Soon after this the property was closed down, remaining idle for several years. Later it was worked on tribute for a time.

Operating costs at that time were not so excessive as one might think, considering the cost of transportation and supplies. It was 204 miles from Helena to the terminus of the Utah & Northern Railway, 52 hr. by stage. The average mining costs, including extensive prospecting, were $8.60 per ton, and milling costs were $2.98 per ton.

The mine water, about 35 gal. per minute, was used in the mill. The 35-h.p. hoisting engine was also used to run the Blake crusher.

The ore was hoisted in iron buckets of 750 lb. capacity, sliding on skids in the inclined shaft. With a mine yield of 1,100 tons of ore per month, hoisting and pumping cost 68c. per ton.

A system of tram lines brought the ore from mine to mill. From the shaft the ore was trammed 200 ft. to the crusher, from which it was carried on a double-track gravity tram 3,300 ft. long to the ore house, and from there it was trammed 300 ft. to the mill. The gravity tram had cars of 2½ tons capacity, and could keep the mill going by running 2 hr. out of the 24. Tramming costs were figured at 29c. per ton.

The mill was considered a very fine one for those times. It was furnished by the Fraser & Chalmers Co. of Chicago, and had 40 stamps, each weighing 650 lb. The stamps dropped 70 times per minute with a 7-in. drop. Forty-mesh screens were used and the capacity was rated at 1½ tons per stamp per 24 hr.

Below the plates there were four Frue vanners, and the vanner tailings were run through blanketed sluice boxes. About 75 tons of concentrates, containing lead as sulphide and carbonate, and averaging $50 per ton, had accumulated up to 1880, but there was no smelter near enough to handle this product profitably.

A 68 per cent. saving was made by the mill, the tailings assaying $8.30 per ton.

The 40-stamp mill cost about $40,000. After the 40-stamp mill was in operation, the arrastre was no longer used. The arrastre plant was a rather pretentious affair, consisting of four grinding tubs, three amalgamating tubs, and one settler.

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In 1880 the Belmont mine had a 20-stamp mill. The property looked very encouraging at this time, having yielded $104,238 from December, 1878, to April, 1880. The yield per ton was between $8 and $20. Mine and mill costs were about $6.50 per ton.

The Belmont vein was in silicified shale in the upper levels, but ran into the granite with depth. The ore paid fairly well in the upper levels, but petered out in value and regularity below. The change of country rock in depth appeared unfavorable in the case of the Belmont.

The Belmont mill had shaking copper plates in addition to stationary plates. These were attached to the Frue vanners as feed aprons, and, fitted with two low riffles (top and bottom), not only made an additional regular saving of gold after the fixed plates, but were effective in catching hard amalgam when careless work at the batteries in feed of quicksilver caused occasional losses. Four vanning machines concentrated the tailings from the plates. The concentrates were treated in an amalgamation pan and settler.

In general it may be said that all the early day mines of the Marysville district, except the Drumlummon, were disappointing in depth.

## THE EMPIRE MINE

BY F. L. SIZER, DOS CABEZAS, ARIZ.

The Empire Mining Co., Ltd., of London, England, was organized early in 1886, to take over the Empire mine, owned by Hickey and Cotter, and the Whipperwill mine, owned by Nate Vestal, who had discovered the Penobscot mine a few years earlier and sold it for a half million dollars. The claims in the Empire group are shown in Fig. 10.

The Empire mine was little more than a prospect, opened by cuts and shallow shafts, none of them over 70 ft. deep, but already showing a good body of high-grade gold-bearing quartz.

The Whipperwill mine had been worked to a depth of 350 ft., and had produced largely, but, at the time of the organization of the Empire Mining Co., had not been in operation for nearly 10 years, and most of the workings were caved and inaccessible.

The writer became the first manager for the new English company and, after a few necessary repairs to the old 10-stamp mill, in Coombes gulch, a mile below the mine, started milling operations on ore which averaged about $20 per ton. This ore was readily amalgamated and a high percentage was usually saved. Inside amalgamation was the method in use in the old mill.

The locality on the west side of Belmont mountain, 5 miles from Marysville and 1½ miles south of Gloster, was a heavily timbered country, cut by deep gulches and subject to a heavy snowfall, which made the spring months difficult ones in which to start an operation.

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Repair of old roads and building of new ones was the first order of business, and, while Marysville afforded some supplies, Helena, 27 miles distant, was the principal supply point, everything then coming by wagon over a heavy grade from Marysville, or the longer route, by Silver City and the Little Prickly Pear creek, a distance of 35 miles.

A telephone line to Marysville was one of the first things constructed and by midsummer of 1886 Empire was a bustling camp.

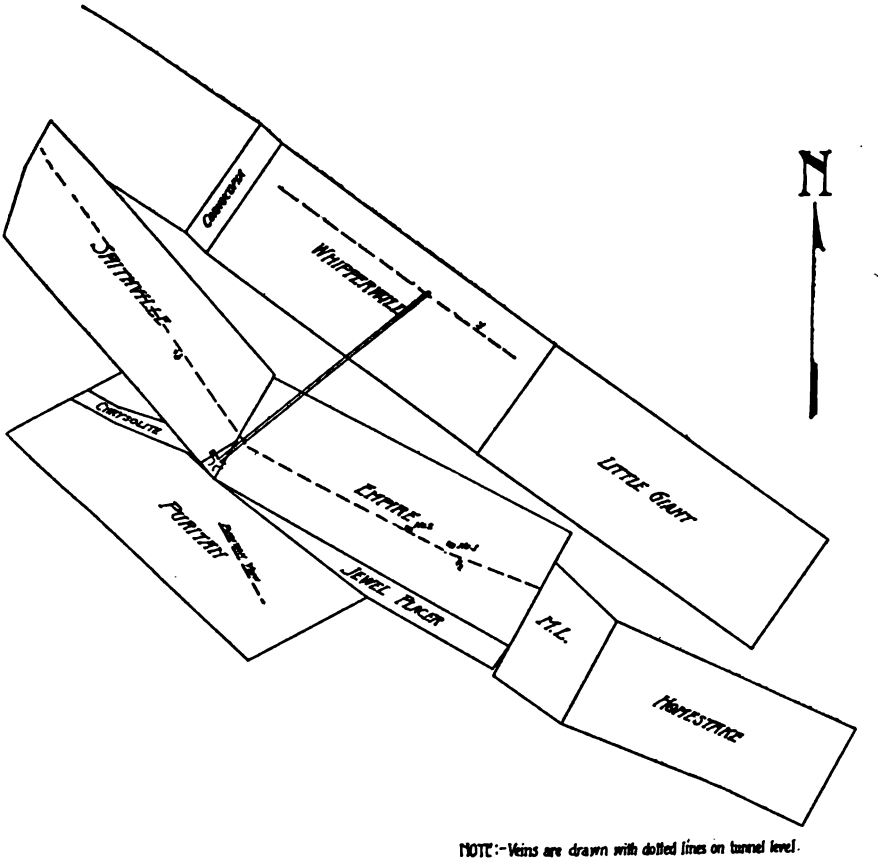


FIG. 12.—EMPIRE GROUP, STEMPLE MINING DISTRICT, MONTANA.

A daily stage to Marysville brought mail and passengers, and as much freight as a four-horse team could handle, sometimes obliging the passengers to walk up the steep grade. Occasional visitors who drove out by team from Helena thought the descent into Coombes gulch altogether too steep for wheeled vehicles and sometimes tied their teams at the top of the hill, coming down to the mill on foot.

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The Drumlummon mine was in full swing, with the 50-stamp mill in operation and the 60-stamp gold mill under construction. A number of the same Englishmen were large owners in both properties.

The initial development of the Empire mine was the driving of an adit at the west end of the claim, through the hard metamorphosed black slate which, at a distance of 120 ft., cut the Empire vein. From this point levels were driven, both east and west on the vein, and this constituted the main working level.

I well remember that this rock was so hard that a good crew of miners who had taken a contract for driving the adit at $20 per foot, threw it up in despair at less than 40 ft. in. This adit was afterward extended straight into the mountain, in a northerly direction, and at 1,100 ft. cut the Whipperwill vein at a depth of 500 ft. where the vein was practically barren. The entire length of this 1,100-ft. working tunnel was in the metamorphosed black slate, without any intrusion of quartz diorite or porphyry and, so far as my development of the Empire vein went, there was no change in the country rock.

The higher-grade Empire quartz was generally slightly copper stained, and all the surface ore highly oxidized, iron predominating. There was enough copper in the tailings to prove a serious drawback to their treatment by the cyanide process, which was attempted several years later, but not enough for us to receive pay for the copper in the concentrates, except in one shipment. There was very little silver in the ore, generally an ounce of silver to the ounce of gold, and the bullion ran about 700 in gold fineness, 298 in silver and 2 in copper. The Whipperwill quartz contained no copper, was generally harder, and the better grade of ore was the rose quartz and purple-stained amethystine quartz, some of the crystals obtained there being beautiful cabinet specimens.

The Empire vein had a nearly east-and-west strike, dipping to the south at 70°, while the Whipperwill, parallel to the Empire vein in strike and occupying the crest of the ridge, was dipping north at about 80°. There were no cross veins intersecting the Empire and Whipperwill veins and no faulting of either of them, although the Smithville (the western extension of the Empire vein) had a strike of about N. 60° W., and there was a radical change in the character of the vein quartz on this west extension of the Empire.

The richer portion of the Empire was between the surface and the 300-ft. level (the adit above mentioned being the 420-ft. level), and on the 300-ft. level, and just above it, was some of the richest gold-bearing quartz ever taken out of any mine in the district.

A peculiar feature here was a slab of galena ore on the hanging-wall, very rich in gold and associated with copper-stained quartz, which extended for a length of 150 ft., a height of 40 or 50 ft., and a thickness of from 2 or 3 in., on the edges, to a thickness of 10 in. in the middle. At

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no other point in the mine was galena ore ever encountered. The Empire vein had an average width of 6 ft., some of the stopes opening to a width of 16 ft.

Below the adit mentioned a winze was sunk to a depth of 130 ft., and levels run from the bottom, east and west, making this depth from surface 550 ft. below the highest point of outcrop at the east end of claim. At this depth there was no pay ore, the vein showing for the most part as a fissure filled with crushed slate, devoid of value, and what quartz there was tending to sprangle out into the slate walls as insignificant seams. No pay ore was taken out below the tunnel level, with the exception of about 40 ft. in depth, and this was a short shoot.

The Whipperwill vein was about 4 ft. in width, opening to 10 ft., but at a depth of 500 ft. was so uniformly low grade as to preclude the possibility of working it at a profit. This was also true of the Smithville, the western extension of the Empire vein, which was devoid of copper-stained quartz and was generally of a value below $6. There was no exploration of the Whipperwill vein below the working-tunnel level, during my time, though I have some recollection of having heard that a shallow winze was sunk, in later years, with no profitable results.

During 1886, amalgamation, both inside and outside the mortars, was tried and a five-stamp battery of light stamps, high drop, for an experiment, and Frue vanners for saving the concentrates, were added to the old mill. With the high-grade surface ore a fairly prosperous year resulted; so that before the end of the first year's existence of the Empire company a 15 per cent. dividend was paid, and authorization made for a new mill.

The concentrates did not constitute more than 5 per cent. of the total mine output; and were usually below $100 per ton in value, and from only one locality, that above the 300-ft. level, as mentioned, did I ever receive pay from the smelters for lead in the concentrates, although all the concentrates carried a considerable percentage of lead, much of it in the form of carbonate. One shipment gave the following assay: Gold, 4.2 oz., silver, 29.4 oz.; lead, 36.5 per cent. The total output in two years, under my management, was a little over a half-million dollars—all from the Empire and Smithville veins, but none from the Whipperwill.

Doubtless because of the fact that the Montana Co., Ltd., at Marysville, had just completed a 60-stamp mill, the Directors declared in favor of a mill of the same size at Empire, and by April of 1887 the work of excavating for foundations was begun. Deliveries of lumber and machinery were slow, owing to the long wagon haul, so that it was late in November before the first 40 stamps were in operation and the remaining 20 stamps, about the end of January, 1888. A substantial and up-to-date mill was the result, and a monthly crushing capacity of 5,000 to 6,000 tons was the average.

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To supply this larger mill very rapid work in the mine was necessary, and, because of the failure of the lowest levels to develop any new ore, a lower average grade ore was supplied to the mill. A rail tramway connecting the main adit level with the mill proved a cheap method of delivering the ore, and the whole operation was systematized and constituted a model operation, stimulating other prospecting and mining work in the district.

Marysville became a lively center and, for the succeeding eight or ten years, was a prosperous mining camp, obtaining rail connection late in the year 1887.

The Stemple district has always been one prolific of profitable gold-quartz mines and now, after the lapse of 25 years, has again taken on new life due to renewed activity in the Bald Butte, Piegan-Gloster, and others of the old mines, re-opened, and it is quite likely that further prospecting will uncover some new bonanza orebodies in the mines of this district.

The Empire mine is one of those which has been operated at intervals since 1893, by a number of different owners, but has been idle for a major portion of the time in the past 20 years. At one time it was reported that a payable body of copper ore had been discovered in the lowest level, but it proved to be only a limited amount of the heavily copper-stained quartz, and even that not specially high-grade gold ore.

It has many times been suggested to me that the lowest level in the Empire was in a barren zone, and that deeper sinking would result in penetrating the continuation of the ore shoots, which were so rich and profitable above.

If any operation in the district is continued long enough to test this surmise, it will be a distinct gain for the district, as a whole, and I hope that some purse will be long enough to give this matter a fair trial.

I confess that the very hard and highly metamorphosed slates at the head of Coombes gulch and general underground conditions in the Empire mine, with many quartz veins which, at shallow depths, 200 ft. or less from the surface, die out in the hard slates, suggests the thought that some of the quartz veins higher up on Belmont mountain than the Empire are likely to be more lucrative to re-operate than this mine, which was my first love in the grand old State of Montana.

Regretting that my scattered notes of earlier years of experience do not permit of my giving a more detailed account of this very interesting property, and hoping that these brief notes will stimulate some of my contemporary engineering friends in Montana to add to this line of historical record, I herewith submit my contribution.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Methods and Economies in Mining

BY CARL A. ALLEN,* DENVER, COL.

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* Non-member.

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## Introduction

In any discussion of mining one is repeatedly confronted with the difficulty of dealing with so many variable conditions. It is not an exact science and in the choice of a method each varying factor has a certain weight, which, in many cases, experience alone can determine. In mathematical terms, it is a function of many variables.

A discussion of mining also loses much of its value unless costs are considered, because the expectation of profit is the only excuse for carrying on mining at all. As conditions vary they cause fluctuations in cost and there are few operations to which a definite cost can be assigned. The character of the ore may make it difficult to drill, yet because of the ease with which it breaks the total cost of drilling and blasting may be low.

In preparing this article the attempt has been, not to cover the whole field of mining, but to describe the different methods of stoping and mining which have distinct principles. In addition to this an effort has been made to show the advantage of dissecting the methods into their detailed operations and applying to these a mathematical study as an aid to the judgment in determining which is the best method to adopt, or in attempting to reduce the cost of a method already in use.

Methods of mining include stoping, caving, and various methods of working large deposits which, in addition to the method of actually breaking the ore, require elaborate and definite plans of development of the orebody. The ordinary methods of stoping are too familiar to all for any elaborate discussion, but it has seemed advisable to review the subject and give the principal advantages and disadvantages of the different methods.

## Factors Affecting Choice of Method

In addition to the various external factors, such as the supply of labor and the financial status of the operating company, the principal items that influence the method of mining to be adopted are:

The size and shape of the deposit:
Character of the ore, whether high or low in grade.
Whether the values are regularly or irregularly distributed.
Physical character, whether hard or soft, tough or brittle.
Character of the country rock.
Immediate and future demands for ore.
Amount of development work done or that may be necessary.
Amount of water to be handled.
Cost of power, timber, and supplies.
Ventilation.
Whether drilling is to be done by hand or with machine drills.

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## Review of Stoping Methods

Depending on these factors, the following methods of stoping may be employed:

Underhand stoping:
- Ore is hoisted to the level above; Cornish stoping.
- Ore is drawn from the level below.

Overhand, or back stoping:
- Starting stopes.
- Drift stoping.
- Cutting out, or lead stoping.
- Raise stoping.
- Longitudinal back, flat-back, or long-wall stoping.
- Rill-cut stoping.
- Saw-tooth back stoping.
- Shrinkage stoping.

Combination stoping.
Breast stoping.
Side stoping.
Sublevel stoping.
Square-set stoping.
Filling methods.

### *Underhand Stoping*

The method of underhand stoping in which the ore is drawn to the level above (*B*, Fig. 1), is called Cornish stoping. It finds application only when it is necessary to mine a lens of good ore below a level and it is not practicable or financially possible to do the necessary development to come up from underneath. Its disadvantages are the excessive cost of raising both ore and water.

Underhand stoping where the ore is drawn from the bottom (*A*, Fig. 1), has more merit than is usually accorded it, especially in the Western States. On the Rand it has been used almost exclusively. Its advantages and disadvantages as compared chiefly to overhand stoping are as follows:

Advantages:
- Ease in drilling down holes when drilling is done by hand.
- Holes are drilled wet and dust is eliminated.
- No platforms are required on which to drill.

Disadvantages:
- Limited to steeper pitches than overhand stoping because the ore does not work straight down the foot wall to the raise; in flat pitches this would necessitate more shoveling.

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Limited to good walls if the vein is steep. Loose rocks sluff off from poor walls and endanger the workmen below.

Levels ordinarily must be driven closer together to reduce the height of the unsupported walls; this necessitates an added cost of development.

Raises must be put up at frequent intervals. This work in some cases amounts to 35 or 40 per cent. of the total development.

No ore reserves are possible.

The waste that can be sorted in the stope if the vein is steep is limited to what can be thrown on lagging supported by stulls.

Certain efficient types of stoping drills cannot be used.

The great advantage if, of course, the drilling of down holes when drilling is done by hand. Few men to-day can or will drill very many

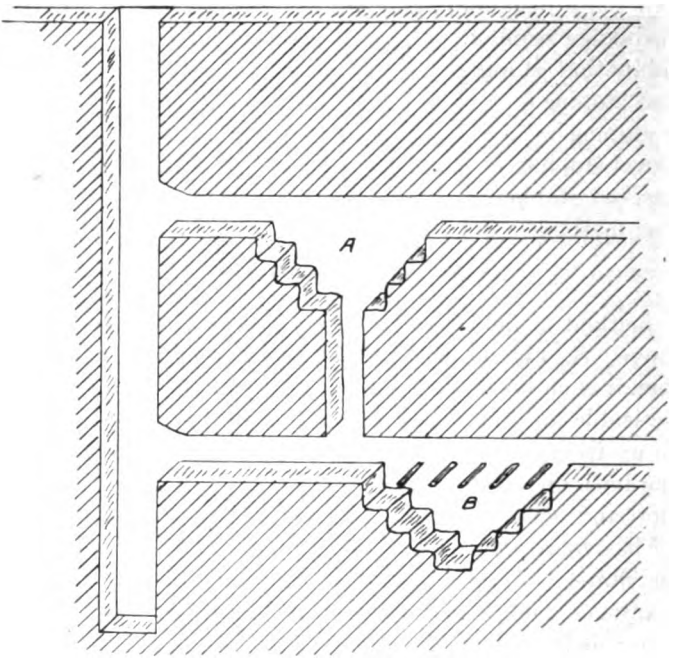


FIG. 1.—UNDERHAND STOPING.

"uppers" in a shift. South Africa had its native labor which could not be taught to drill them. It might seem on first thought that the disadvantages are so numerous that they preclude any chance of the method being adopted under ordinary labor conditions, but I have examined mines where the combination of conditions in narrow veins indicates very strongly that the method would be more economical than overhand stoping and quite as safe.

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*Overhand Stoping*

Overhand stoping, in general, has the following advantages and disadvantages:

Advantages:

Stoping can be started directly from the level without any raises or winzes.

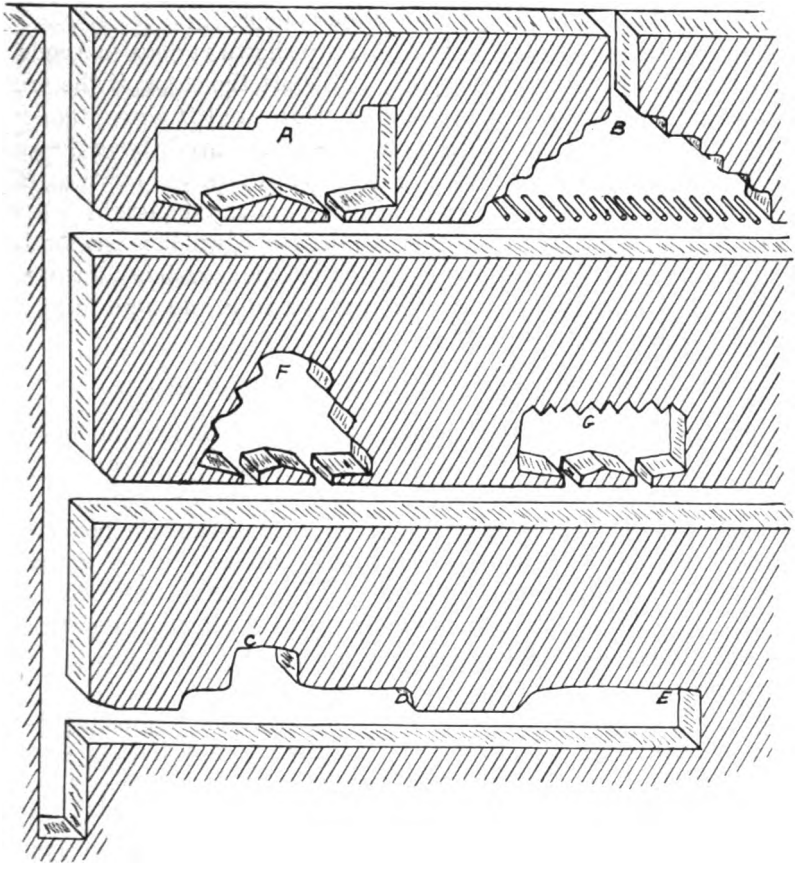


FIG. 2.—OVERHAND STOPING.

Levels can be driven far apart. These two considerations mean less cost for development.

Advantage is taken of the force of gravity in breaking the rock.

Miners work at the back and can inspect it so that the danger of falling rocks is largely eliminated.

Either ore or waste can be stored in the stope.

A flatter seam can be worked than in underhand stoping without

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the necessity of shoveling, as the ore runs directly down the pitch and is given an impetus by the blast.

Water takes care of itself.

Pillars of ore or waste can be left easily.

Disadvantages:

In most cases holes must be drilled up or flat.

If the stope is not filled with ore or waste, stulls and platforms must be used, even though the walls require no support.

If the raises are far apart ventilation is poor.

*Starting an Overhand Stope.*—In Fig. 2 are shown various methods of working the back in overhand stoping. To start the stope, if the ore is low grade and timber is scarce, a pillar or pillars are left above the level as shown in *A*, and if the ore is high grade it is all removed above the level and stulls are placed as shown at *B*. To start a stope as shown at *B*, a cutting out or lead stope is broken immediately above the level as shown at *D*, Fig. 2. *C*, Fig. 2, is a wide raise or raise stope, which is one method of opening an overhand stope, and *E* is a drift stope, which is a term used in the Lake Superior copper mines and means a wide and high drift as a start for the overhand stoping.

### *Longitudinal Back, Flat-Back, or Long-Wall Stoping*

After starting an overhand stope the shape in which the back is carried is often of prime importance. In *A*, Fig. 2, is illustrated the method of carrying a flat or longitudinal back (the term long-wall is used sometimes when the vein lies nearly flat). In this method the benches or breasts are frequently made of a height sufficient to allow of a square set being placed. This is done at Butte and it is often referred to as breast stoping. In general, a flat-back stope as illustrated has the following advantages and disadvantages:

Advantages:

If the stope is filled with waste or ore it is more convenient for men to work on a level surface. Tramming, shoveling, or sorting in the stope can be done to better advantage, and waste raises can be driven farther apart.

Square sets may be conveniently placed by making the benches the proper height.

Timber struts or cribs can be used between the filling and the back, in case the latter is weak.

A long stope distributes the broken ore well along the level, and tramming is thus facilitated.

Disadvantages:

The principal disadvantage of a flat-back stope is that if filling is used, and kept close to the back, it must be distributed in wheelbarrows or cars.

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### *Rill-Cut Stoping*

Rill cut, or rill stoping, as shown at *B*, Fig. 2, has the principal advantage that when filling is used it can be run down through a raise at the apex of the stope and will fill the stope without shoveling. Its disadvantages, on the other hand, are as follows:

When waste filling is run in from the raise it has a sloping surface which makes it difficult to keep ore and waste from mixing, and is unhandy for men to work.

Raises must be put in at frequent intervals, and the added cost of these raises may exceed the cost of spreading the filling in a flat-back stope.

Ore chutes must be carried up through the filling, and the timber used in their construction cannot be recovered. This is a disadvantage as compared with shrinkage stoping.

At *F*, Fig. 2, is shown a method of rill stoping in which the benches are inclined so that down holes can be used. Down or water holes give off no disagreeable dust, and can be drilled faster with piston machines.

### *Saw-Tooth Back Stoping*

This method of carrying the back in an overhand stope is shown at *G*, Fig. 2. It is claimed that it makes drilling more convenient if machine drills mounted on a bar are used.

### *Shrinkage Stoping*

Shrinkage stoping refers to any overhand method in which the stope is kept full of broken ore until it is completed. The miners stand on top of the broken ore and work at the back. As broken ore takes up more space than ore in a solid mass, about 35 to 40 per cent. of it must be drawn to leave room for working.

Advantages:

Raises may be placed far apart.

The broken ore serves as a support to the walls. This does away with the necessity for much timber.

The miners work on top of the broken ore; timber platforms are eliminated, and the work is made much easier. It is also convenient to work a larger number of men in the stope.

A large ore reserve is maintained.

Large rocks can be broken with sledges in the stope and blocking the chutes is avoided.

No ore passages are required from the level up to the back of the stope. Manways are necessary up through the broken ore, but the



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timber used in their construction can usually be recovered when the ore is drawn.

Ventilation is better than in an empty stope.

Disadvantages:

Stoping must be kept ahead of the demand for ore. This requires additional capital.

There is practically no opportunity to sort ore in the stope.

Ore filling is not permanent and the stope may cave when it is withdrawn.

Scaling or unstable walls may cause waste to mix with the ore or prohibit the use of the method altogether.

Although shrinkage stoping is rather generally used, it would be used more if it were not for the fact that many mines are not in a condition to keep an ore reserve, but must draw the ore for the mill as fast as it is broken. The added efficiency to be obtained from the miners when working on a firm floor of ore, and not on loose lagging laid on stulls, is a very important advantage of the method.

### *Combination Stoping*

In a discussion of stoping methods, steeply pitching veins are usually referred to, because the majority of veins in nature are steeper than 45°, and also because to refer each method of stoping to veins of all dips causes confusion. Before considering combination stoping, however, a brief *résumé* of the methods of handling ore in veins of different dips is necessary. In veins dipping from 35° or 40° up to 90° the ore runs down by gravity. From 20° to 35° it must be assisted by shoveling, or by using chutes with smooth bottoms or placed at an angle steeper than that of the vein. From 10° to 25° the ore may be taken down to the level by shoveling, shaking chutes, mono-rail trams, gig-back railways, or conveyors. From veins that are horizontal up to those having dips of 10° to 15° tracks are usually laid on the foot wall and cars are pulled up to the face by men or animals. An animal can pull an empty car up a 6° slope and a loaded car up a 3° slope. So in dips greater than 6° the track must be laid at an angle with the line of the dip.

In combination stoping, which is illustrated in Fig. 3, and which is a combination of underhand and overhand stoping, it is possible to keep the working face more nearly in a line parallel to the raises and perpendicular to the level. This is a distinct advantage if a stationary or shaking chute is being used to carry the ore to the level, because a large part of the face of the stope is accessible to the chute. This becomes a double advantage if a large output is required. In combination stoping development work is also reduced because levels may be driven far apart, and fewer raises are required than in simple underhand stoping.

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### *Side Stoping*

This is an extreme of combination stoping in which the working face of the stope is vertical. The face is parallel to the raises just as the face in flat-back stoping is parallel to the levels.

### *Breast Stoping*

Breast stoping refers to the working of a flat orebody, or a flat section of an orebody, just as coal is mined from a flat coal seam. That is, a slice is worked in a horizontal direction. The assumption is that there is no open stope either above or below the slice, or else the method

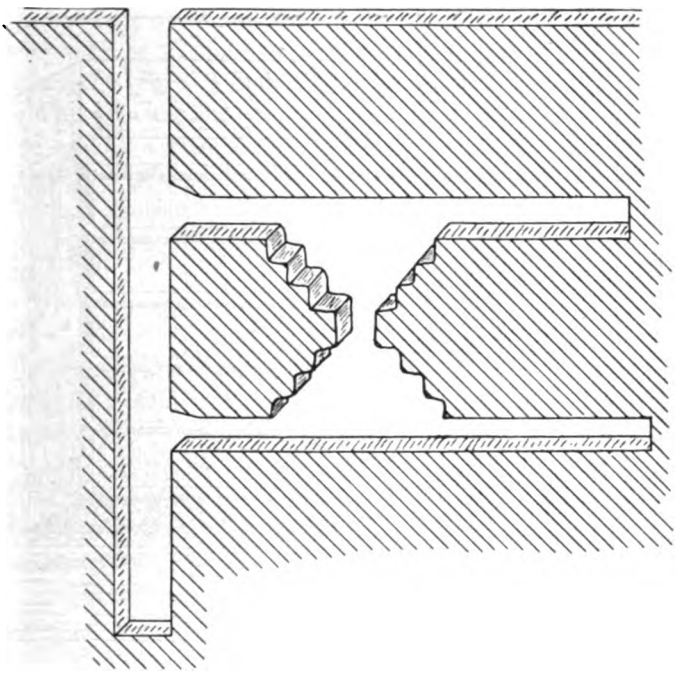

FIG. 3.—COMBINATION STOPING.

becomes either underhand or overhand stoping. In some cases the benches in underhand or overhand stoping are called breasts, and the method breast stoping, but I believe that is not the usually accepted definition. Breast stoping, in the strict sense, is the only name applicable to the mining of a horizontal slice, such as the sill floor of a large overhand stope, or the slices in the top-slicing method.

### *Sublevel Stoping*

This method, described by F. W. Sperr in the *Engineering and Mining Journal* of June 5, 1912, and by P. B. McDonald in the *Mining and*

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*Scientific Press* of July 5, 1913, is really a combination of several different methods of stoping. Fig. 4 is an illustration of the method, and shows half of the vein cut away where the stoping is being done. The miners go from the haulage level up the raise into the sublevels. On the first sublevel they will blast into the shape of funnels the raises that come up from the haulage level, and after working back the slice, *s*, by breast stoping will drill holes at *a* and shoot part of this bench down into the raises. On the second sublevel the miners work back the slice, *s*, and then with down holes at *b* will blast the remainder of the first bench down into the raises, and with uppers at *c* shoot off part of the second bench. Both sets of holes will be fired at the same time. In this way

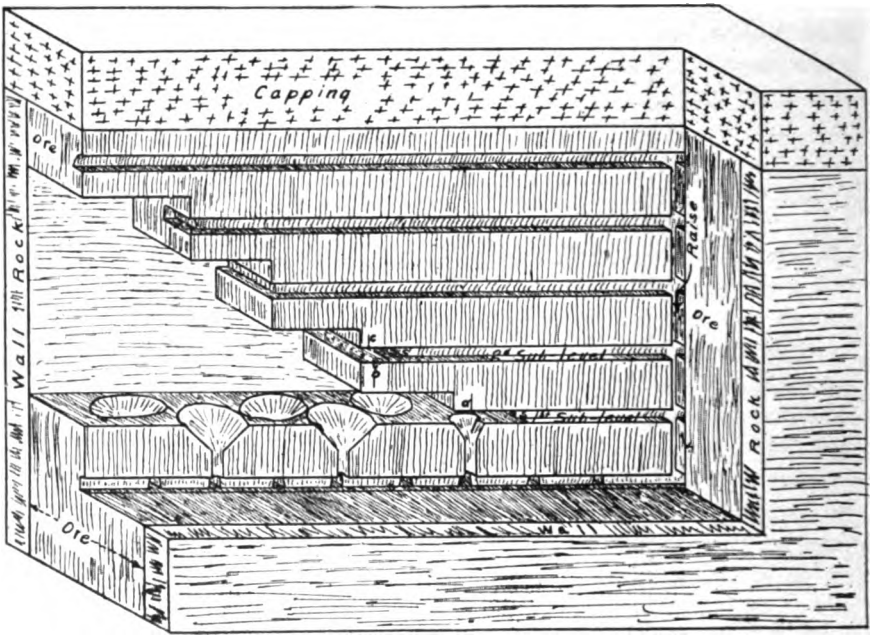


FIG. 4.—SUBLEVEL STOPING.

each sublevel is drawn back; the broken ore is drawn off through the funnel-shaped raises, which leaves a large open stope. The sublevels are driven about 25 ft. apart vertically, and 8 to 10 ft. are shot off from each bench from the sublevel below. Obviously where there is a capping above the ore, as shown in the cut, part of it will fall when the upper bench is blasted and part or all of this ore may be lost. "When the ore is from 50 to 100 ft. wide, a narrow stope of about one-third of the width of the ore is drawn back through the middle, leaving a pillar of ore standing on each side." These pillars or slabs left against the walls are then drawn back in the same manner. The method is applicable to

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veins from 12 to 100 ft. wide, or wider, if pillars are left between stopes, and evidently comes into competition with shrinkage stoping, square-set stoping, and top slicing. It requires expensive development but permits a large tonnage to be broken in a relatively small stope. The working back of the slices on the sublevel is expensive mining, but after that is done the rest of the ore blasted from the benches requires a comparatively small amount of powder.

### *Square-Set Stoping*

Square-set stoping may refer to any method of stoping in which square-set timbers are used. Unless otherwise specified, however, the term is limited to overhand flat-back stopes in which square sets are used, either with or without waste filling. Square-set mining is advantageous when the vein is wide and the walls will not stand without timbering and shrinkage stoping cannot be used; when, on account of surface conditions, caving cannot be allowed, or where caving might cause the loss of other orebodies; when the ore changes rapidly in grade and requires frequent sampling; when the orebody is irregular in outline; and when old stopes may have to be approached or passed through at some later time.

### *Filling Methods*

Mine workings are filled with waste as an aid to timbers in supporting weak walls or back, or to avoid fully or in part the use of timbers. Workings that must be prevented from caving for a length of time in the future are best protected by waste filling. The filling may be waste rock from development, rock blasted from the walls or surface for the express purpose of filling, or, if available, sands from concentration mills make excellent filling and are cheaply placed in the stopes.

## Caving Methods of Mining

There are three distinctive methods of working large deposits that involve, in some way, the factor of caving. Slightly different names are in use, but those that are simple as well as descriptive are top slicing, sublevel caving, and block caving. In a discussion of these methods, three things must be borne in mind: that the methods are usually applied to massive deposits; that the deposits are usually divided into blocks or panels, and the description of mining one panel is practically a complete description of the method; and that, in most cases, the orebodies do not come to the surface but are covered with a capping. This capping may be glacial drift, as in some instances in the iron regions around Lake Superior, or it may be rock from which the ore values have been leached, as is the case at some of the large copper deposits.

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### *Top Slicing*

Top slicing, illustrated in its ideal form in Fig. 5, consists in the working of an orebody in horizontal slices, beginning at the top. Levels, for haulage, are established at proper intervals and, from these, raises are put up to the top of the orebody about every 50 ft. Starting at the tops of these raises drifts are run out and then, retreating toward the raise, the slice is worked back by breast stoping. The overburden or

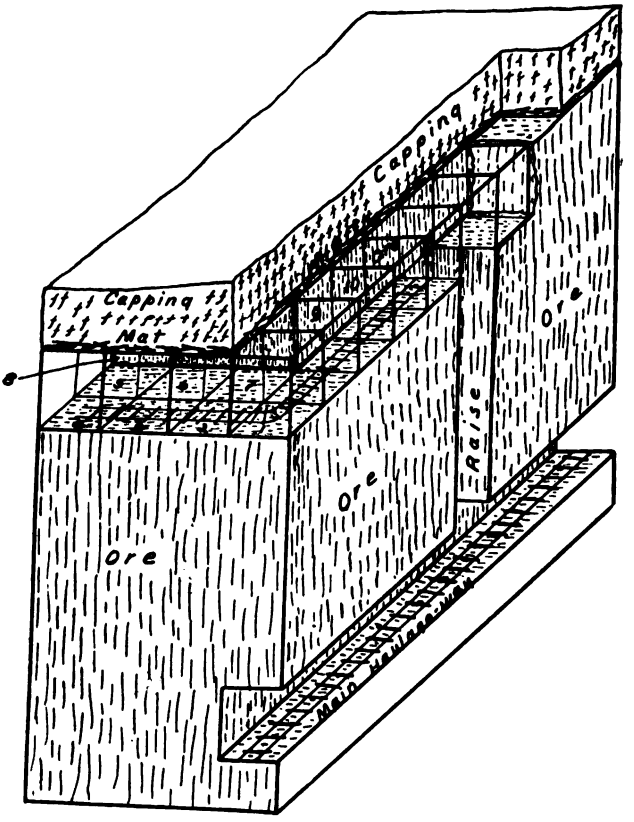


FIG. 5.—TOP SLICING.

capping lying above the ore is supported on square sets or posts until a slice, or part of a slice, has been worked. The floor of the slice is then covered with slabs or plank and the supporting timbers are shot out and the capping is allowed to cave on to the timber floor. This floor and the broken posts form what is called a mat, which keeps ore and capping separated. The miners then start in the raise and work out another slice of the ore just under the previous one and catch up the timber mat with posts or square sets. When this slice is completed another

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floor is laid and the supports again blasted. In this manner the orebody is worked in successive slices from the top downward. The broken ore from each slice is run to the raise in wheelbarrows or cars and dropped to the haulage level. The capping caves and follows down on top of the mat.

In this method the ore is not caved at all, but the ground above the ore does cave, and the surrounding country will cave more or less according to the amount of ore removed. If there are any other orebodies in the region affected by the cave, they will be lost or their recovery made more difficult. This is one of the reasons which may prohibit the use of caving methods. Also, unless the deposit is small compared to the distance to the surface, surface subsidence will take place and the surface must be free from buildings or roads. Top slicing is adaptable to heavy ores that can be mined easily, and require heavy timbering and filling if worked by overhand stoping. Deposits of large extent, over which the overburden will cave readily, are customary conditions. In some cases, top slicing may require as much or even more timber than an overhand stope with square sets, but for very heavy ore, which if worked overhand necessitates strong sets, reinforced and braced, it takes less timber, and in any case poorer timber may be used.

The following are the advantages and disadvantages of top slicing, chiefly as compared with overhand square-set work or with other methods of caving.

Advantages:

Cheaper timber, and for heavy ores less timber, is required than for square-set stoping.

No waste filling is necessary.

Ore can be sorted as mined.

Very little waste from the capping becomes mixed with the ore.

Less skilled labor is required.

Any rich fine ore produced in breaking will be saved on the slice below.

Complete extraction of the ore is possible.

Disadvantages:

Caving may cause injury to surface structures or render unworkable other orebodies in the region.

Mining the ore by breast stoping requires more drilling and blasting.

Broken ore on the slice must be shoveled into barrows or cars and taken to the raises.

Development must be kept ahead of the demand for ore.

It is difficult to leave bodies of low-grade ore which may be found in the deposit.

Ventilation is difficult, and in some cases, the decaying timbers in the mat give off heat and obnoxious gases.

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It is possible to work only on the top of the ore, and although several slices may be worked at the same time, it is not possible to establish work at different levels. This may make it difficult to obtain a required tonnage from a deposit of small horizontal section. Capping should cave readily, and not arch, or the method may become very dangerous.

### *Sublevel Caving*

This method, otherwise referred to as sublevel drifts and back caving, or sublevel slicing, resembles top slicing in that the orebody is worked from the top down, and the ore is taken out in horizontal slices. A block

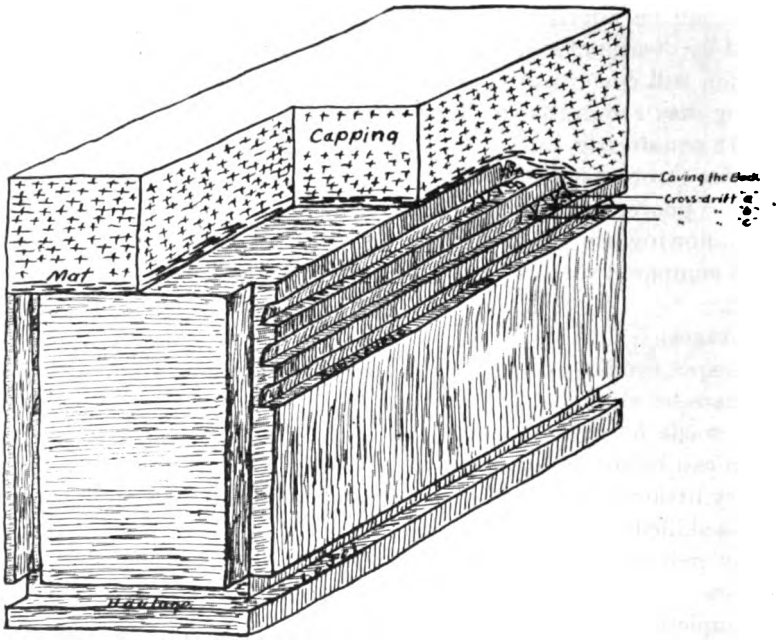


FIG. 6.—SUBLEVEL CAVING.

of ore is left above the slice, however, and when a small portion of the slice has been mined out, this back is allowed to cave. Fig. 6 shows a block or panel of ore worked by sublevel caving. Raises are put up from the haulage level, and from these sublevel drifts are driven 14 to 20 ft. apart vertically. When these drifts reach the boundary of the property, or of the panel to be worked, cross drifts as *a*, *b*, and *c* are driven across the block and timbered. The back of ore is thus undercut but is supported by the drift timbers. The timbers of cross drift *a* are next blasted and the weight of the capping caves the back of ore down on to the floor

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of the slice, where the miners shovel it into cars and push it to the raise. A timber mat may be used as in top slicing. Sublevel caving is adaptable to massive deposits or wide veins in which the ore is not difficult to break, yet is firm enough to hold the capping while supported by the drift timbers. An overburden that will readily cave is necessary.

Advantages:

The cost of mining is low, because a large percentage of the ore is broken with little or no blasting, and the amount of timber used is small.

A large output is possible.

A large percentage of the ore can be saved.

The ore can be kept freer of waste than in block caving.

Disadvantages:

It is limited in application to certain ores.

A large amount of development is required which must be kept ahead of the demand for ore.

A large part of the ore must be shoveled.

Caving endangers surface and other ore deposits.

There is some danger to the miners.

Ventilation is difficult.

*Block Caving*

Block caving is an extreme case of sublevel caving in which instead of a back of ore 5 to 10 ft. thick, one 50 ft. thick is undermined and allowed to cave. The method is illustrated in Fig. 7. After the bottom of the block, or panel, is cut up into pillars by drifts and cross drifts, the pillars are robbed, and then the remaining stumps are blasted out with one large blast. This allows the entire block above to cave. In settling it disintegrates so that it can be shoveled with very little additional blasting. For a block to cave it is usually necessary for it to be freed on one or more sides. This is done, as shown in the cut, by narrow stopes, called isolating stopes. After the ore has settled for from two to six months timbered drifts are driven through it. The broken ore is allowed to run into these drifts, starting farthest from the shaft, is shoveled into cars and trammed out. As soon as capping shows at any point shoveling is stopped.

The advantages of this method are:

The cost of mining is low because very little drilling and blasting, or timbering is required.

The amount of development is small.

A large output is possible.

Disadvantages:

The method is limited in application to low-grade ore of such character that it will cave and disintegrate.

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A large amount of ore becomes mixed with capping and lost. There is no opportunity to select or sort the ore.

*Back Caving into Chutes*

There is another principle that should be mentioned under mining methods. It is called back caving into chutes or chute caving and in some ways resembles block caving. (The description of a mine employing this principle is given in *Mining Without Timber*, by Brinsmade, p. 181.) Large overhand stopes are worked, not by drilling and blasting the entire back, but by blasting out narrow isolating stopes around the edges of the

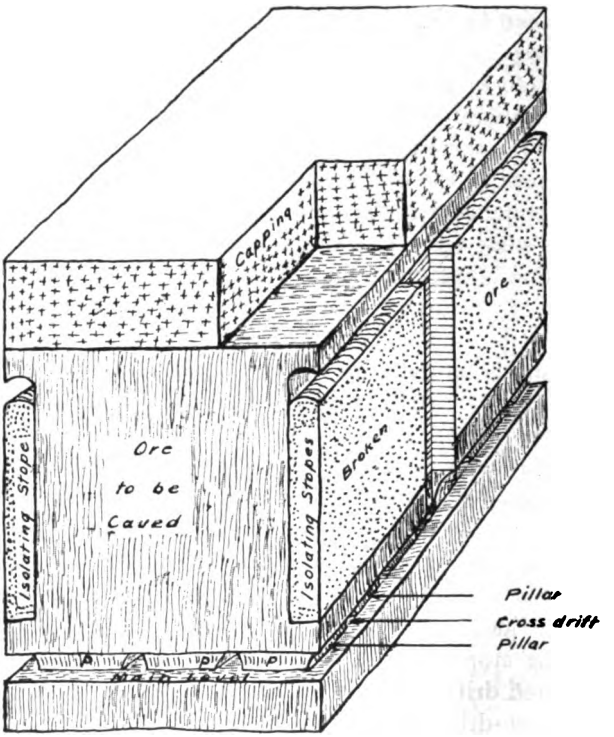


FIG. 7.—BLOCK CAVING.

main stopes and allowing the rest of the ore in the center to fall of its own weight. That is, a large percentage of the ore is mined by caving. The broken ore is drawn off through chute raises in the bottom of the stope.

As far as I am aware, the preceding pages cover every method of mining that has a distinctive feature and can be readily classified. There are innumerable different systems of mining but they all involve only these principles, or modifications and combinations of them.

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## Costs of Mining Methods

To compare properly two different methods of mining, it is not wise to attempt to calculate the total cost of each, but rather to compare the

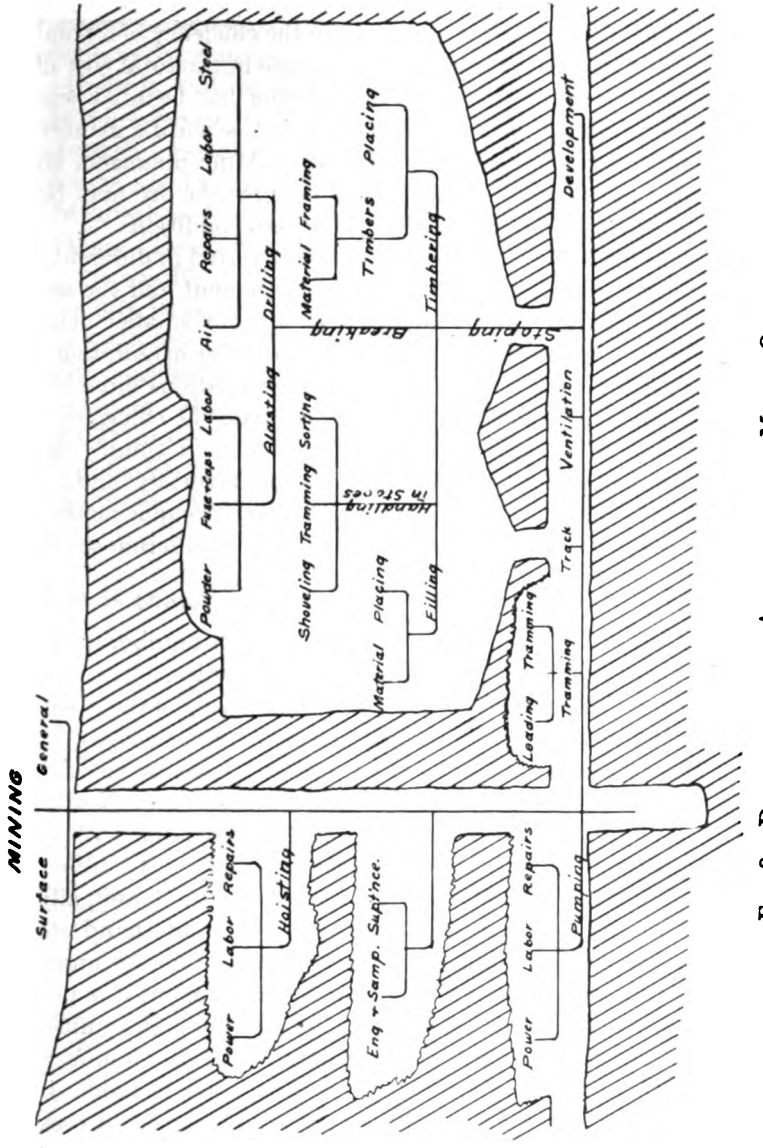


FIG. 8.—DIAGRAM SHOWING ARRANGEMENT OF MINING OPERATIONS.

costs of those operations that are differently executed in the two methods under consideration. When any method is in operation the proper way to reduce the total cost is to dissect it into its different operations and see

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if the cost of one or all of these cannot be reduced. Keen competition has forced large manufacturing and other industrial enterprises to give careful attention to cost accounting and efficiency engineering. This calls for the investigation of the smallest details; conditions are adjusted so that a workman will not lose time in going after a tool or in walking from one machine to another. An increase in the efficiency of a number of small operations means a corresponding decrease in the total cost of production. The reasons why efficiency engineering has been so generally adopted are two: First, it attacks its problem in a scientific manner and, second, it brings results in dollars and cents. Mine managers are not excusable from applying its principles just because the ore may be high enough in grade to pay dividends even under lax conditions.

The different operations of mining can be separated in different ways. Fig. 8 is a diagram which shows a logical arrangement and the smallest operations shown in it could be subdivided into yet smaller details *ad infinitum*. In determining upon a new method or in attempting to reduce the cost of one already established, these operations must be carefully studied. If modifying a method will increase the efficiency of one operation without affecting the rest, the problem is a simple one, but usually more than one operation is affected, some adversely, and some to advantage, in which case to determine which is the cheaper method it is necessary to apply to each operation its cost under the different conditions.

*Costs of Drilling and Blasting. Comparison of Stoping Methods*

As an appendix to this article I have collected a number of costs of mining as published in the reports of mining companies.

Obviously but few of these reports show the costs of the operations given in Fig. 8, and in order to bring out their detailed costs it is necessary to make certain assumptions. To bring out the costs of the different operations of drilling and blasting, consider the data on North Star, Alaska-Treadwell, and the Rand as given in the appendix, with general data on Cripple Creek and the Michigan copper mines. All are straight overhand stoping methods with the exception of the Rand, and, although not so stated, I assume that these data refer to underhand stoping. The first consideration is drilling. Up to the present time for underhand work, mounted machines were the only ones available, but it is probable that in the future they will be largely replaced by light unmounted machines of the Jackhamer type. For overhand work, the hammer stoping drill, unmounted, has demonstrated its superiority in reduced labor cost, increased ease of handling, and greater drilling speed in rock that is not too hard. While actually at work machines will drill from 0.25 to 15 in. a minute, usually 1 to 3 in. At the North Star mine, evidently, 25 to 30 ft. are drilled for each machine shift. Light stoping drills are used and break 7½ tons in a 4-ft. stope. In Cripple Creek the average

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of several mines is 60 ft. for each machine shift. Light stoping drills are used and the machine runner is using the machine about 6 out of every 8 hr. In a 5-ft. stope 14 4½-ft. holes will break about 15 tons. In the Lake Superior copper region, in the amygdaloid lodes, machines drill from 30 to 47 ft. a shift in different mines.[1] In the conglomerate, on account of more massive copper and higher quartz content, about 24 ft. for each machine shift is the average (9-hr. shift) and seldom is more than 28 ft. drilled. Holes in both cases are usually 8 ft. deep and a burden of 2 to 3 ft. is placed on them. In the conglomerate, although harder to drill, 54 tons are broken for each machine shift, whereas in the amygdaloid only about 38 tons are broken. The stopes are 10 to 30 ft. wide. At the Alaska-Treadwell each machine in the stopes drills 28 to 34 ft. and breaks 32½ to 45 tons. Heavy piston drills are used and the stopes are about 60 ft. wide. On the Rand large machines break 7.2 tons in a 55-in. stope, 10 tons in a 6-ft. stope, 19½ tons in a 7-ft. stope, and small machines break 3.7 tons in a 4-ft. stope. To arrive at the actual cost of drilling it is also necessary to consider the cost of compressed air, machine labor, repairs, and drill steel and sharpening. The cost of compressed air varies from $0.40 to $2.50 for each machine shift. It depends on the cost of power, size and efficiency of the compressor, size and efficiency of machine drill, and the conditions of the drill. The accompanying table taken from *Ore*

| | | | Cost per 1,000 Cu. Ft. Free Air Compressed | | | Cost per Drill Shift. | | |
|---|---|---|---|---|---|---|---|---|
| Style of Compressor | Maximum Capacity Cu. Ft. Free Air per Minute | Total Cost per H. P. Hour | Sea Level | 5,000 Ft. Altitude | 10,000 Ft. Altitude | Sea Level | 5,000 Ft. Altitude | 10,000 Ft. Altitude |
| Simple steam (non-condensing..... | 200 | 2.2c. | 5.9c. | 5.3c. | 4.8c. | $2.07 | $2.22 | $2.40 |
| Compound steam non-condensing)..... | 300 | 1.5 | 4.0 | 3.6 | 3.3 | 1.40 | 1.50 | 1.65 |
| Simple steam (condensing)..... | 2,500 | 1.0 | 2.7 | 2.4 | 2.2 | 0.95 | 1.01 | 1.10 |
| Compound steam (condensing)..... | 3,000 | 0.8 | 2.2 | 1.9 | 1.8 | 0.76 | 0.81 | 0.88 |

Table based on 8-hr. shift with coal at $5 a ton.

[1] Claude L. Rice: *Engineering and Mining Journal*, vol. xciv, No. 5, p. 217 (Aug. 3, 1912).

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*Mining Methods*, by W. R. Crane, gives the cost of operating a 3-in. drill in granite.

At the North Star mines machine men cost $3 a shift or 43c. a ton; machine power, 40c. a shift or 6c. a ton; repairs and lubrication, 42c. a shift or 6c. a ton; drill steel and sharpening, 75c. a shift or 10c. a ton; total for drilling, $4.57 a shift or 65c. a ton. To this must be added 25c. a ton for powder, 4c. for fuse and 1c. for caps, which makes a grand total of 95c. a ton. Allowing for rock left in the stopes this would be reduced about 15 per cent.

At Cripple Creek machine men cost $3.50 a shift or 24c. a ton; air, $2 a shift or 13c. a ton; repairs and lubrication, 25c. a shift or 2c. a ton; drill steel and sharpening, 50c. a shift or 3½c. a ton. Total for drilling, $6.25 a shift or 42½c. a ton. To this add 24c. a ton for powder, 3½c. for fuse, and 1c. for caps, which gives a grand total of 71c.

At the Alaska-Treadwell machine men and helpers average $6.85 a shift or 21c. a ton; air, 75c. a shift or 2c. a ton; repairs, 50c. a shift or 1½c. a ton; drill steel and sharpening, 45c. a shift or 1½c. a ton. Total for drilling, $8.55 or 26c. a ton. Adding 15c. for powder, 1c. for fuse, and ½c. for caps gives 42½c., to which must be added an additional 6c. for extra labor which includes powder men. This gives a grand total of 48½c.

The data given in the appendix on operations on the Rand are too incomplete to analyze as the others are analyzed. From other data which I have, I assume that the costs are about as follows: Machine labor, $4.50 a shift; air, 75c.; repairs, 50c.; drill steel and sharpening, 50c. This is a total of $6.25. An average of 12 tons were broken each shift, which gives a cost of 52c. a ton for drilling. To this must be added 25c. a ton for explosives, which gives a grand total of 77c.

On the Rand, despite the greater stoping width, the cost of explosives is nearly as much as at North Star or Cripple Creek and the drilling cost per shift is much greater. The influence of the width of the stope on the tonnage broken is well illustrated in the Alaska-Treadwell and the added cost of machine labor is minimized. The long holes used and the character of the ore, no doubt, preclude the possibility of using one-man machines.

From the above, it is seen that an item worthy of consideration is that of drill steel and sharpening. It varies greatly but in any case is a factor worthy of attention. Not only the cost of new steel and the wages of the blacksmith, but the delivery of sharp drills to the stope and the removal of dull steel, should be considered. By comparing cost items occurring in the reports, which are given in the appendix, other features influencing drilling and blasting might be shown, but the greatest value results in making comparison when conditions are similar and are known with exactness.

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*Lighting*

It is stated in one of the mining books that the cost of lighting for each ton of ore mined is about the same in all mines. This is not the case. The cost varies according to the light used and the number of men underground. The entire subject of mine lighting has been well covered by Frederick H. Morley in the *Mining and Scientific Press* of April 11, 1914. The following table taken from this article gives the cost of acetylene lighting in 10 of the large metal mines.

| Name of Company | Number of Men Employed Underground | Number Using Lamps | Carbide Consumption per Lamp, Oz. Shift | Cost Carbide per Lb., Cents | Cost per Lamp Shift, Cents | Cost of Candles per Shift, Cents |
|---|---|---|---|---|---|---|
| Homestake Mining Co..... | 1,025 | 1,025 | 8.0 | 3.50 | 1.75 | 7.00 |
| Ray Con. Copper Co..... | 1,400 | 1,200 | 9.0 | 4.50[a] | 2.50[a] | 5.00[a] |
| Quincy Mining Co..... | 1,389 | 575 | 6.7 | 3.50[a] | 1.46[a] | ..... |
| Osceola Con. M. Co. ..... | 625 | 625 | 6.0 | 3.50 | 1.38 | ..... |
| United Verde Copper Co.... | 600 | 575 | 6.5 | 5.50 | 2.23 | 5.40 |
| Bunker Hill & Sullivan Co.. | 460 | 208 | 7.0 | 5.25 | 2.30 | 6.18 |
| Calumet & Arizona M. Co... | 1,000 | 60 | 7.0 | 5.50 | 2.40 | 6.64 |
| Ohio Copper M. Co..... | 96 | 37 | 8.0 | 5.80 | 2.90 | ..... |
| Nevada Con. Copper Co.... | 200 | 20 | 4.0 | 4.67 | 1.12 | 3.27 |
| Mammoth Copper M. Co.... | ..... | 12 | 10.0 | 5.86 | 3.66 | 5.15 |
| Average..... | ..... | ..... | 7.22 | 4.76 | 2.17 | 5.52 |

[a]Estimated.

*Timbering and Handling Ore in Stopes. Square-Set vs. Top-Slicing Methods.*

There is often a question as to whether square setting or top slicing is more economical. For instance, in massive deposits of heavy sulphides the ore is easily drilled and breaks well and if overhand stoping with square sets is used the weight of the ore necessitates very heavy timbering. In order to compare the two methods and also to show cost data on timbering and handling ore in stopes, assume the methods applied to a block of ground 50 ft. square and assume that the development work is the same in each case. The sets to be 6 by 6 by 8 ft. In the top-slicing method a drift one set wide is started at the top of the raise and run to the boundary of the block and then across the end of the block. This will give 10 sets of drift. It will take about eight holes, each 4½ ft.

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deep, to break a round, or 12 holes to break a set. All work in top slicing is breast work, and the drilling must be done with a drill mounted on a column. A mounted drill will drill slower and require more time for moving than a light, unmounted, stoping drill, which can be used in square-set mining. The cost of each set for the first 10 sets of drift in top slicing will be about as follows:

| | |
|---|---|
| Drilling, labor, 2 shifts @ $3.50 | $7.00 |
| Drill repairs @ 0.30 | 0.60 |
| Drill steel and sharpening @ 0.20 | 0.40 |
| Air @ 1.00 | 2.00 |
| Powder, 25 lb. @ 12½c. | 3.00 |
| Fuse, 60 ft. @ ½c. | 0.40 |
| Caps, 12 @ 1c. | 0.12 |
| Cost for each set | $13.52 |

Each set contains 288 cu. ft. or approximately 28 tons; this makes the cost of drilling and blasting 48c. a ton. The distance to tram the ore to the raise will average 50 ft. and the cost for mucking and tramming may be estimated as follows: A man will shovel this ore at the rate of about 2½ tons an hour, or 1 ton in 24 min. A wheelbarrow will hold about ¼ ton and to wheel seven barrow loads 50 ft. will require from 10 to 15 min., say 11 min; this makes the time consumed in shoveling and wheeling 1 ton 35 min. or 22c. a ton if shovelers, wages are 37½c. an hour. This is equal to 14 tons handled for each 8-hr. shift and a cost for each set of $6. Exclusive of the raise there will be a total of 63 sets in a slice, of which 10 sets are mined for 70c. a ton (48c. for drilling and blasting plus 22c. for mucking). The remaining 53 sets have two free faces to break to. The cost of breaking will be 30 per cent. less or 35c. a ton. The average distance from the raise will be less, which will reduce the cost of shoveling, say to 20c., which gives a cost of 55c. for the remaining 53 sets or an average cost of 57c. a ton to mine the whole slice. On one slice there will be required 72 posts, 64 caps, and 72 girts. A post 8 by 8 in. by 8 ft. will contain 42 ft. B.M. and costs 84c. if lumber is figured at $20 per M. Caps and girts 8 by 8 in. by 6 ft. will cost 64c. each, figured on the same basis. Framing the posts will cost about 10c. each by hand or 5c. each by machines, say 6c. Framing the caps and girts will cost about the same. This gives:

| | |
|---|---|
| 72 posts @ 90¢ | $64.80 |
| 136 caps and girts @ 70¢ | 95.20 |
| | $160.00 |

which is $2.55 a set. The cost of taking the timber into the stope and placing it will amount to from $1 to $2 a set, say $1.50. Other timber for lagging and blocking will cost about 85c. a set, which gives a total

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for timbering of $4.90 a set or 17c. a ton. This added to the cost of breaking and mucking gives 74c. a ton.

To mine this same block of ground with square sets, it would be necessary to divide it into two stopes four sets wide and eight sets long. The ore has been assumed to be heavy and it would be impossible to carry a wide stope. As soon as a stope is mined it should be filled. For drilling, a light stoping drill can be used which, under the assumed conditions, will drill easily 50 per cent. more footage than the machine on the slice. One-third less explosives per ton will be required, so the reduction in cost of breaking will be at least 30 per cent. or to 30c. a ton. A flooring of plank will have to be laid on the top set and about 75 per cent. of the broken ore will not fall into the chutes but must be shoveled. A shoveler will handle 3 tons an hour at a cost of 12½c. a ton for 75 per cent. of the ore. This is equivalent to 9c. for the total tonnage. We have, therefore, reduced the cost of breaking and shoveling from 57c. in top slicing to 39c. in overhand stoping with square sets, a saving of 18c. a ton.

Consider the timbering. It is not necessary to go again into detail but first assume that the ground can be held up by square sets of 10 by 10 in. timber. A post 10 by 10 in. by 10 ft. long contains 66 ft. B.M. and at $20 per M. will cost $1.32. Since it is heavy timber and framed on both ends, the framing will cost about 12c., making its total cost $1.44. Caps and girts will cost $1 each plus 8c. for framing, or $1.08 total. This makes a total of $3.60 for timbers in the square set instead of $2.30 in the set used in top slicing, an increase of $1.30. There will also be an added cost of placing of about 50c., which makes a total additional cost for timbering in the square-set method of $1.80 per set or 6½c. per ton.

These figures show very plainly that the timbering used in square set must be very heavy to make the increased cost over top slicing greater than is the saving in breaking. Here the saving in the latter is 18c. but the increased cost of timber is only 6½c., leaving a balance of 11½c. in favor of square-set mining. This is in accord with experience, but when timbers will not support the ground and many braces are required, or filling must be resorted to, top slicing becomes cheaper. Filling will cost at least 20c. a ton and probably 50c. One mine that I have had in mind in preparing these figures has a cost of timbering and filling of more than $1 a ton. At the Esperanza mine, Mexico, in 1907, 35 ft. B.M. of timber were used for each ton of ore, which at $20 per M. is 70c., and that for timber alone. It must be remembered, however, that in ore that does not break easily the additional cost of breaking in the top-slice method becomes much greater.

I do not want to give the impression that these figures are to indicate the total cost of mining by the different methods. There are many other items such as superintendence and ventilation that enter into the cost of mining. There are also incidental expenses in breaking and timber-



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ing, such as repairing, timbering chutes, and the like, the cost of which will enter into the total cost of any method, and the amount can be judged only by experience. *The figures given bring out only the relative costs of different operations in different methods under assumed conditions.* In my opinion, a few figures are necessary to aid in determining the merits of different methods or in reducing the costs of methods in use. With the figures, experience and judgment must be added. These are probably more necessary in mining than in any other business, because there are so many variable conditions.

### *Tramming*

The costs of tramming as given in published reports nearly always include not only tramming, but loading also, either from chutes or from the floor of a drift. The following interesting data are from the *Engineering and Mining Journal* of Mar. 8, 1913.

"At the Elkton Consolidated Mining & Milling Co., Cripple Creek, Colo., the 1911 cost of tramming was 14.6c. per car of approximately 0.7-ton capacity. The South Utah Mines & Smelters, Newhouse, Utah, reports its tramming cost for the year ended June 30, 1912, at 15.76c. per ton of ore, which evidently includes the cost of handling waste removed. In Goldfield, Nev., tramming has averaged about 18c. per ton of ore produced from stopes and has ranged from about 14 to 25c. These figures are for actual tons trammed and do not include any shoveling in stopes. At the North Star mine, Grass Valley, Calif., observations show that a man pushes an 18-cu. ft. car about 150 ft. per minute and shovels about 3 tons per hour from a plat into car against 2 tons when shoveling off a rock bottom. According to this a shoveler's efficiency is increased about 50 per cent. by using a plat. The Wolverine Co., Houghton, Mich., reports tramming costs at 17.4c. per ton of ore, and the Wettlaufer-Lorain, Cobalt, Ont., 21c. per ton of ore."

*Data on Tramming*

| Mine | State | Size Car Used | No. of Men | Length of Tram Ft. | Amount Trammed per Man-Hr. |
|---|---|---|---|---|---|
| | | | Shoveling from Rock Bottom | | |
| Erie Consolidated | Calif. | 1½ ton | 2 | 1,000 | 1.17 ton |
| Erie Consolidated | Calif. | 1½ ton | 1 | 1,000 | 1.6 ton |
| Pittsburg-Silver Peak | Nev. | 1.1 ton | 1 | 700 | 1.52 ton |
| Cananea Consolidated | Mex. | 16.8 cu. ft. | 1 | 300 | 35.8 cu. ft. |
| | | | Shoveling from Plat | | |
| Erie Consolidated | Calif. | 1.0 ton | 1 | 100 | 1.75 ton |
| Pittsburg-Silver Peak | Nev. | 1.1 ton | 1 | 1,000 | 1.575 ton |
| Cananea Consolidated | Mex. | 16.8 cu. ft. | 1 | 300 | 42.0 cu. ft. |
| Ohio Copper | Utah | 20 cu. ft. | 1 | 100 | 41.0 cu. ft. |
| | | | Loading from Chute | | |
| Erie Consolidated | Calif. | 1½ ton | 1 | 1,500 | 3.12 tons |
| Pittsburg-Silver Peak | Nev. | 1.1 ton | 1 | 700 | 6.19 tons |
| Cananea Consolidated | Mex. | 16.8 cu. ft. | 1 | 300 | 84.2 cu. ft. |
| Ohio Copper | Utah | 20.0 cu. ft. | 1 | 150 | 206 cu. ft. |
| Mother Lode | B. C. | 2.15 tons | 1 | 450 | 8.4 tons[a] |

[a] Tramming with horses and locomotives.

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If a man shovels from a plat at the rate of 2½ tons an hour and wages are 37½c. an hour the cost of shoveling is 15c. a ton. If a 1,000-lb. car is being used he will fill it in 12 min. If the tramming distance is 1,000 ft. and the trammer walks 200 ft. a minute the trip will require 10 min., allowing 2 min. to dump, the tramming will take the same time and cost the same as the loading, that is 15c. a ton or a total of 30c. a ton. Three things are very evident: Tramming is an important item of cost in nearly every mine, the size of the car makes a decided difference, and the resistance to traction of the car itself and the grade and condition of the track are important factors. I remember seeing one stretch of track 1,000 ft. in length over which it took two men to push a car of 1,000 lb. capacity. Figuring as above this increased the cost of tramming 15c. a ton, and in fact it was even more because the two men could push the car but slowly.

Henry Nagel, Superintendent of the Vindicator mine at Cripple Creek, has made some interesting observations in regard to tramming. He noted in one case that tramming on 800 ft. of level track cost 3c. a ton more than tramming on 800 ft. of track which had a grade of ½ per cent. in favor of the loads. In another case a man who had been shoveling and tramming in a drift with bad air, did 50 per cent. more work when a good circulation of air was secured. In his opinion the greatest efficiency is secured from a trammer when the track grade is such that he can ride the car when going in loaded. As to the latter assertion a few figures may show how it would not be true under all conditions. A car weighing 500 lb. and holding 1,200 lb. of ore will weigh 1,850 lb. when a 150-lb. man is riding. If it has good bearings and has a resistance to traction of only 20 lb. to the ton it will run on a 1 per cent. grade. If it has very poor bearings and has a friction of 60 lb. to the ton it will require a 3 per cent. grade for coasting. On this grade the empty car returning weighs 500 lb. and would require one-fourth of 120 lb. or 30 lb. force to push it. This is too much for the average man. The car with the good bearings would take only a push of 10 lb. to bring it back up the grade, which shows the importance of good bearings, as well as of good track.

## Mining of the Massive Porphyry Copper Deposits

The past decade has witnessed the development and successful operation of a half dozen or more immense deposits of low-grade copper ore that occur in the Western States. In every case sufficient capital was available to carry out the development of the mines in the manner decided upon, and it is interesting to note the different methods that were employed.

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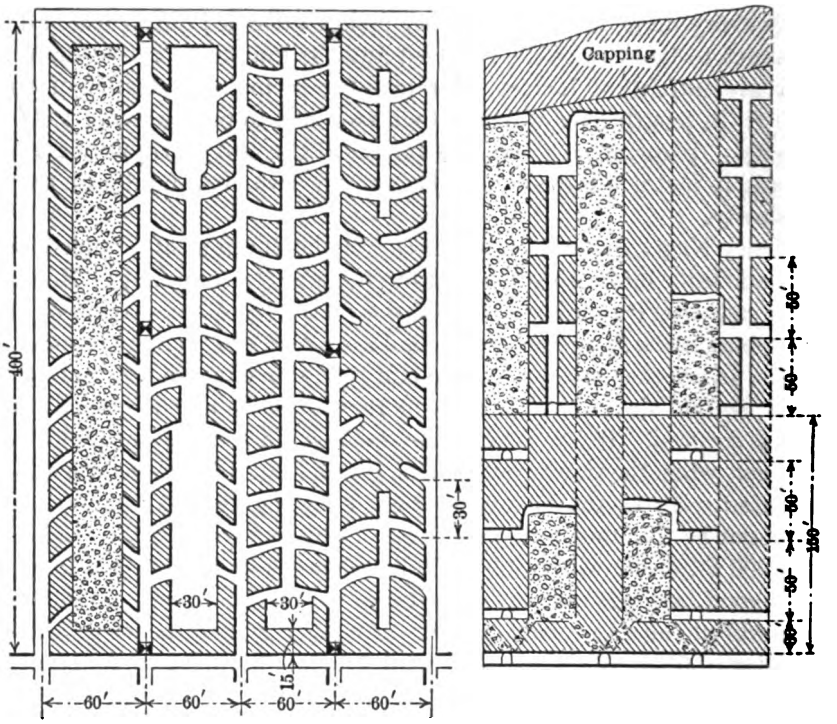
*Utah Copper and Boston Consolidated Mines*

The first two "porphyry coppers" were the Utah Copper, and Boston Consolidated, at Bingham, Utah. The Utah Copper started out with a modification of the chute-caving method described in Crane's *Ore Mining Methods*, p. 141. The method was not satisfactory and was




PLAN END VIEW OF STOPES

FIG. 9.—STOPING SYSTEM, BOSTON MINE.

discarded for stopes and pillars, although very little mining is done by this method because the ore will be handled by steam shovels. The Boston Consolidated mine, now called the Boston mine of the Utah Copper Co., was originally laid out into stopes and pillars. Fig. 9 illustrates the method.[2] Above the main haulageway there is a 30-ft. pillar of ore to protect it. Above this the deposit is divided into a series of vertical stopes 30 ft. wide and 150 ft. high alternating with vertical pillars of the same dimensions. Above these comes another and similar

[2] Figure taken from article by C. T. Rice in *Mines and Methods*, September, 1910, and copied in *Mining Without Timber*, p. 172.

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series of stopes and pillars, but the pillars in the upper series come immediately above the stopes in the series below. The plan was to work all the stopes on both levels as shrinkage stopes. When the stopes were all completed drawing would commence from the bottom of the lower stopes. The upper pillars being over stopes below would settle and crumble so that they would pass down with the ore from the upper stopes when the lower pillars were weakened by blasting. It will be noted that in this method the capping must cave and follow down on top of the broken ore. Just what alterations in the plan of working would have been necessary if the method had remained in use it is difficult to state. The stopes are still being worked but only the excess ore is drawn off; the remaining ore is left to be mined by steam shovels after the capping is removed. The width of stopes has been reduced from 30 to 18 ft., and the pillars increased to 42 ft.

*Ray Consolidated Mine*

The Ray Consolidated Copper Co. at Ray, Ariz., is controlled by the same interests as the Utah Copper Co. and when their system of mining was laid out they no doubt profited by the earlier experience at Bingham. The orebody is flat and averages a little more than 100 ft. thick. The main haulageways are at the bottom of the ore and above them the ore is divided into vertical stopes 15 ft. wide leaving 10-ft. pillars between as shown in Fig. 10[3]. The stopes are worked as shrinkage stopes to the top of the orebody, then the bottoms of the pillars are blasted and drawing is commenced from under both stopes and pillars. The pillars, being undercut, settle and break up as the ore is drawn. By taking care to draw equally from all the chutes under a block of ground the capping will cave and follow down uniformly on top of the ore and there will be little mixing of the two. The Ray system, from all reports, has been eminently successful. Stopes 15 ft. in width are wide enough so that a large tonnage will break compared to the footage drilled, and all the advantage is had of the shrinkage method of stoping. After the ore in the stopes is broken, the pillars, which contain more than one-third of the total ore, are available for extraction with only a slight expense for drilling and blasting. The cost of mining for the last quarter for which reports have been published was 71c. a ton. This includes a proportion of all general and fixed charges, but does not include an allowance of 12½c. a ton for the retirement of mine-development suspense account.

*Miami Mine*

The orebody of the Miami Copper Co. is much deeper than that at Ray. In horizontal cross-section it is roughly circular, about 1,000 ft.

[3] Abstract of article by L. A. Blackner, *Mining and Scientific Press*, Jan. 3, 1914.

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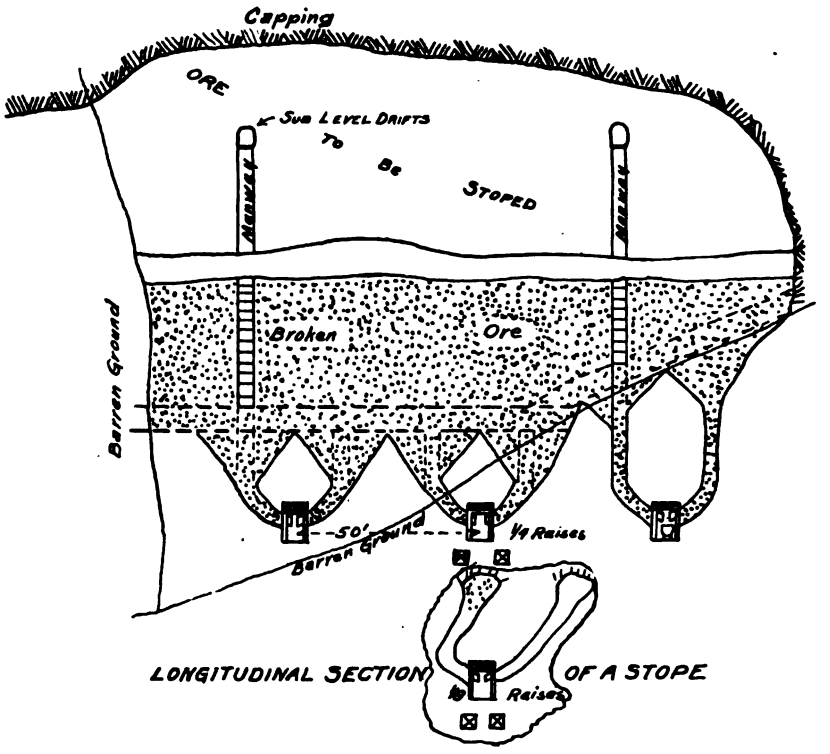


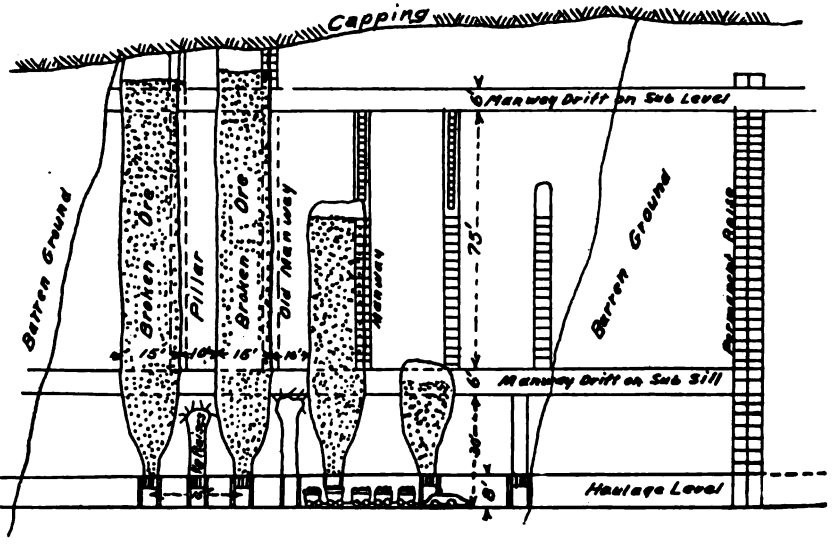


FIG. 10.—SECTIONS OF STOPES, RAY CONSOLIDATED MINE.

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across. A tongue of waste rock intrudes into the ore from one side. The irregularities in the orebody under the capping are first worked by square-set stopes and timber is placed to form a mat between ore and waste. Through the main body of ore sublevels are spaced 25 ft. apart vertically as shown in Fig. 11[4] and on these sublevels drifts and cross-drifts are driven each way 50 ft. apart. The main body of the ore is to be mined with shrinkage stopes and pillars similar to the Ray method except that

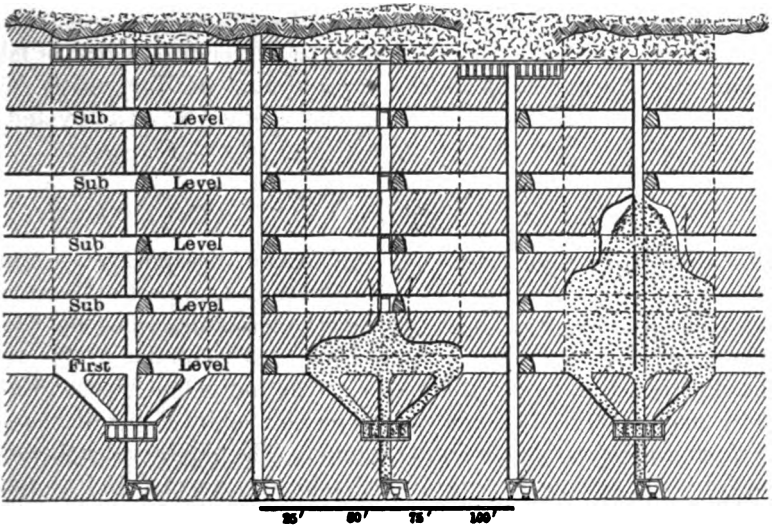


FIG. 11.—CROSS-SECTION OF STOPE, MIAMI MINE.

at Miami the stopes were planned to be 60 ft. wide and the pillars 40 ft. In working the stopes the miners do not work on top of the broken ore but approach the stopes through the drifts on the sublevels and break the ore into the stopes by the method of sublevel stoping previously described. The ore breaks readily into small pieces and obviously a 60-ft. shrinkage stope would be very dangerous if worked in the usual way, but by the sublevel method the danger is eliminated. Even under these conditions the last report states that it has been found advisable to make the stopes narrower. After the shrinkage stopes are carried up to the mat, which is next to the capping, the pillars are worked from the top down by top slicing. As this is done the ore from the stopes is drawn and a uniform settlement of the capping results.

The cost of mining at Miami has been about $1.20 a ton. Last year a premature crushing of square sets under the capping started a cave

[4] R. L. Herrick in *Mines and Minerals*, July, 1910; *Mining Without Timber*, p. 167.

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which extended through into the stopes below. The repair of this damage caused an increase in the cost for 1913. The Miami orebody is of higher grade than that at Ray and the Miami method will save a greater percentage of clean ore. The cost per ton is greater and there is a question as to how much the cost can be reduced in the future. To date, a large amount of ore has been taken from the square-set stopes, an expensive operation, but, on the other hand, in the future there will be an increased amount of ore to be mined from the pillars, which is also expensive. It would be interesting to know how much of the difference in cost between the methods at Ray and Miami is due to the difference in cost of labor. At Miami the miners receive $3.50 a day and up. At Ray mostly Mexicans are employed and they are paid, I believe, about $2.50 a day and up.

### *Inspiration Mine*

The Inspiration mine, adjoining the Miami, has not yet begun to produce on a large scale, but it has been developed, in part at least, with the idea of using block caving. The method was described by Claude T. Rice in *Mines and Methods*, June, 1909, and is similar to the block caving illustrated earlier in this paper. The individual blocks were to be 75 by 200 by 200 ft. high and were to be isolated by stopes on the sides and breast stopes between ore and capping. The pillars under the blocks instead of being as shown in Fig. 7 were to be very narrow, 75 ft. long and 75 ft. high. Between the pillars narrow shrinkage stopes filled with ore would prevent the pillars from early crushing. This method, I believe, has never been given a trial and now is to be replaced by the method of block caving which has been so eminently successful at the Ohio Copper mine at Bingham. This method is worthy of a fuller description. The following is abstracted from a description of the method by Clarence G. Bamberger in the *Engineering and Mining Journal* of Apr. 6, 1912.

### *Ohio Copper Mine*

"In *résumé* the conditions are: An orebody opened 400 ft. in width, 450 ft. in length, 1,300 ft. in depth, dipping from the horizontal at an angle of 50°. The entire mass being a broken shattered quartzite containing copper, chiefly in the form of chalcopyrite disseminated throughout, inclosed by foot and hanging wall of the same formation with boundaries not clearly defined.

"The preparation of this ground for a caving system of mining has been carried out as shown in Fig. 12. An incline shaft was sunk in the foot wall on the dip of the vein connecting with the different levels for the transportation of men and supplies. Two main ore chutes No. 1 and No. 2 were driven in the foot wall on an angle slightly greater than the dip of the vein. . . . . These chutes connect by diagonal raises through the foot wall, with the different levels where extraction is taking place and

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deliver into bins of large capacity at the Mascotte tunnel or main haulage adit which in turn leads to the concentrator situated at the portal about 3 miles distant.

"Lateral main levels were driven across the orebody at the 100-, 300-, 400-, 500-,

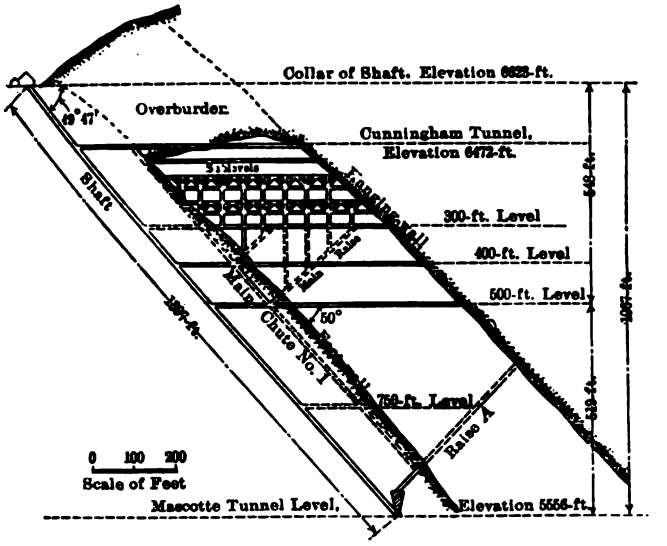


FIG. 12.—CROSS-SECTION, OHIO COPPER MINE.

750-, and 1,000-ft. or haulage level. By a network of crosscuts and drifts these different levels have been cut up into blocks 200 ft. square which in mining will again be subdivided. Between the several levels the ground has been blocked out into sublevels

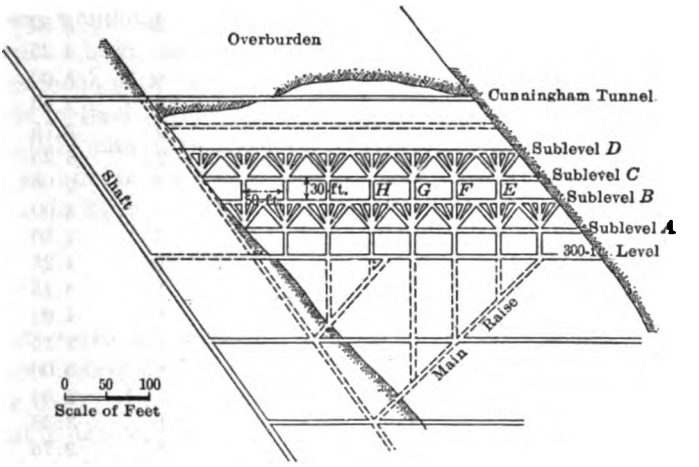


FIG. 13.—CROSS-SECTION, OHIO COPPER MINE.

30 ft. apart and again subdivided by crosscuts, drifts and raises into blocks approximately 30 x 50 x 25 ft.

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"The actual procedure of extraction can be clearly understood by reference to Fig. 13 which shows in detail the sublevels above mentioned. Thus we have for example the four sublevels, *A*, *B*, *C*, and *D*. Raises are driven from the crosscuts and drifts on the 300-ft. level through sublevels *A* and *B*. From these vertical raises at the various points *E*, *F*, *G*, and *H* raises are run into sublevel *C*, radiating like fingers from the palm of the outstretched hand. At the breast of these several raises sublevel *D* is blasted down, the ore falling by these leads into the cross inclined chutes connecting with the main ore chute, which delivers into bins at the loading station on the haulage-tunnel level.

"The same procedure is carried out on sublevel *B* which in turn is blasted down; at the same time sublevel *C*, which has already been cut up by this finger-like network of raises, comes with it as the solid ground below is blasted down. Thus each alternate sublevel is cut up by the numerous raises and the corresponding sublevel above and below is blasted down.

"The actual cost per ton for the different phases of the operation is shown in the accompanying table which is an average covering a period of 31 days in October, 1911, during which period 56,311 tons were mined.

*Typical Daily Labor Report. Ohio Copper Co.*

| Description of Labor | Number | Rate | Amount |
|---|---|---|---|
| Superintendent | 1 | $11.66 | $11.66 |
| Foremen | 2 | 5.33 | 10.66 |
| Shift bosses | 3 | 4.00 | 12.00 |
| Timekeeper | 1 | 3.00 | 3.00 |
| Sampler | 1 | 3.00 | 3.00 |
| Mine engineer | 1 | 5.00 | 5.00 |
| Hoist engineer | 2 | 3.25 | 6.50 |
| Stationary engineer | 2 | 3.50 | 7.00 |
| Chainman | 1 | 3.50 | 3.50 |
| Tool sharpener | 2 | 3.50 | 7.00 |
| Machine men | 21 | 3.25 | 68.25 |
| Machine men | 8 | 3.00 | 24.00 |
| Muckers | 23 | 2.50 | 57.50 |
| Muckers | 10 | 2.75 | 27.50 |
| Loaders | 2 | 3.25 | 6.50 |
| Loaders | 3 | 3.00 | 9.00 |
| Chute tappers | 3 | 3.00 | 9.00 |
| Nippers | 2 | 2.50 | 5.00 |
| Blacksmith | 1 | 4.25 | 4.25 |
| Blacksmith helper | 1 | 3.25 | 3.25 |
| Carpenter | 1 | 4.00 | 4.00 |
| Carpenter's helper | 1 | 3.25 | 3.25 |
| Trackmen | 1 | 3.00 | 3.00 |
| Trackman's helper | 1 | 2.50 | 2.50 |
| Pipeman | 1 | 3.25 | 3.25 |
| Pipeman's helper | 1 | 2.75 | 2.75 |
| Timbermen | 7 | 3.25 | 22.75 |
| Timbermen's helpers | 7 | 2.75 | 19.25 |
| Total | 110 | | $344.32 |

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| | |
|---|---|
| Total labor charge for Oct., 1911 | $11,582.78 |
| Total stores consumed for period | 3,743.82 |
| Total power consumed for period | 476.00 |
| Total operating expense for period | $15,802.60 |
| Total development and equipment for period | 3,429.00 |
| Total actual operating expense for period | $12,373.60 |
| Average number of feet driven per day | 31 |
| Average number of feet raised per day | 42 |
| Average tons mined per day | 1,817 |
| Total number of feet driven for period | 961 |
| Total number of feet raised for period | 1,302 |
| Total tonnage mined for period | 56,311 |

All calculations based on net weight.

"From the above data several interesting conclusions can be drawn. Considering the whole working force of 110 men as producing the ore, approximately 17 tons of ore are delivered to bins per day per man. Considering, however, only the men who are actually breaking the ore, approximately 63 tons of ore are delivered to bins per day per man. From the "total operating expense for period" and "total tonnage mined for period," the cost per ton delivered to bins is shown to be 28.06c., which figure includes development and equipment charges. From "total actual operating expense" and "total tonnage mined for period," the actual cost per ton for mining delivered to bins is shown to be 21.97c."

This method of mining is similar to the ideal case of block caving already described, only instead of having to shovel the ore after it is caved a large number of branching raises are brought up from underneath and the ore runs into these and on down to bins at the bottom of the mine without any handling. The raises contain chutes and the ore is drawn evenly from a large area so that the capping will follow down without mixing too much with the ore. While considerable ore is lost the cheapness of the method is remarkable. I have in my possession the figures for April, 1913, almost two years after the above figures were taken, and the cost of mining including development and equipment is reduced from 28.1c. to 22.2c.

## Development

The extensive development necessary for the successful operation of some methods of mining naturally brings to mind the question of what this work costs. In the appendix, I have placed a collection of figures on the cost of development as published in the reports of mining companies Easily driven drifts and crosscuts will cost $3.50 to $4.50 a foot. Average-sized drifts in hard rock with a small amount of timbering cost $6 to $7 and under unfavorable conditions the cost will be $10 or more a foot. In a 5 by 7 ft. drift costing $6.50 a foot the labor for drilling and blast-

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ing will be about $2.15, labor for shoveling and tramming about $1.20, air about 30c. a foot, explosives about $1.25, and miscellaneous 60c. a foot. Raises are as a rule less expensive than drifts and, in very favorable ground, can be driven for $1.50 to $2.50 a foot. The average 4 by 5 ft. raise in hard rock will cost $4 to $5 and raises difficult to drive or requiring close timbering will cost $8 a foot and up. Winzes are not driven often but when they are they cost from $10 a foot up.

Space permits of a consideration of only a few methods from the point of view of development. At Kalgoorlie (J. Cheffirs, *Transactions Australian Institute of Mining Engineers*, vol. xiii) in a block of ore 500 ft. long and 200 ft. high (the distance between levels), rill stoping required four raises and shrinkage stoping required one; 800 ft. of raises cost about $5 a foot or $4,000. If we assume the vein to average 8 ft. wide a block of ore would contain about 60,000 tons, which means a cost for development of almost 7c. a ton. With shrinkage stoping this cost would be less than 2c. a ton. Of these methods touched upon in the preceding pages the Miami mine has the greatest footage of development work. The scheme of development allows a good many different methods of mining to be used without any changes; it also permits a large tonnage to be produced from a limited area. A block of ore 50 by 50 by 50 ft. contains approximately 50 ft. of raise and 200 ft. of drifts and crosscuts. Estimating raises at $3 a foot and drifts at $5 a foot gives a total of $1,150 for the block. It contains about 10,000 tons of ore, which would make 11½c. a ton as the cost for development; this excludes the cost of shafts, stations, and similar requirements.

At Ray 12½c. a ton is allowed for the retirement of mine-development suspense account but just what this covers is not stated. It evidently covers more than development for stoping.

The method used at the Ohio Copper mine requires considerable development of which a large percentage is raising. Due to the fact that many of these raises were inclined, so that a miner could work at the face without using timber, and that all the ore, after being blasted, runs by gravity to the bins, the cost of the raises was only at the rate of 50 or 60c for each ton of ore mined in driving them.

## APPENDIX

### *Mininy Costs*

*Montana-Tonopah Mine.* (*Mining and Scientific Press*, Mar. 22, 1913)

Silicified and mineralized veins in altered andesite, 3 to 5 ft. wide, steeply pitching. Sometimes thrown over by faults through the stopes. Overhand stoping.

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Year ended Aug. 31, 1912; 53,874 tons mined.

| Labor: | |
|---|---|
| Ore breaking | $0.579 |
| Mine machines | 0.031 |
| Hoisting and dumping | 0.197 |
| Boilers | 0.034 |
| Shoveling and sorting | 0.675 |
| Tramming | 0.224 |
| Timbering | 0.209 |
| Tool sharpening | 0.026 |
| Surveying | 0.028 |
| Foreman and bosses | 0.078 |
| Sampling | 0.016 |
| Storekeeper | 0.011 |
| Assaying | 0.016 |
| Watchman | 0.011 |
| Superintendence | 0.046 |
| Maintenance and repairs | 0.001 |
| Total labor | $2.182 |

| Supplies: | |
|---|---|
| Water | $0.005 |
| Ore breaking | 0.292 |
| Compressed air | 0.111 |
| Hoisting and dumping | 0.059 |
| Hoisting (elec. power) | 0.129 |
| Timbering | 0.197 |
| Total supplies | $0.793 |
| Total mine cost | $2.975 |

*Tonopah-Belmont.* (*Engineering and Mining Journal*, May 11, 1912)

Year ended Feb. 29, 1912; 182,000 tons dry ore and waste; 152,550 tons dry crude; 115,560 tons dry, sorted.

Fracture zones in andesite and rhyolite-dacite, filled with quartz. Overhand stoping.

| Development: | Per Ton Dry, Sorted, Cents |
|---|---|
| Miners | 47.4 |
| Muckers and trammers | 22.2 |
| Timbermen and helpers | 8.4 |
| | 78.0 |
| **Stoping:** | |
| Miners | 44.5 |
| Shovelers | 33.9 |
| Trammers | 19.2 |
| Timbermen and helpers | 97.8 |
| Filling | 4.2 |
| Piston-drill repairs and maintenance | 5.0 |
| Stoping-drill repairs and maintenance | 2.9 |
| Steel and sharpening | 7.1 |
| Explosives | 28.6 |
| Hoisting to surface | 30.9 |
| Auxiliary hoisting | 9.4 |
| Ore sorting and loading | 27.3 |

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| | | |
|---|---|---|
| Sampling and assaying | 4.7 | |
| Surveying | 5.6 | |
| Supt. and shift bosses | 13.7 | |
| Mine office | 12.9 | |
| Surface and plant | 16.2 | |
| Lighting | 4.6 | |
| Heating | 4.1 | |
| Drayage | 6.1 | |
| Maintenance and repairs to buildings | 2.5 | |
| Maintenance and repairs to machines and machine tools | 2.3 | |
| Maintenance and repairs to pipe lines and tanks | 1.6 | |
| Maintenance and repairs to railroad spurs | 0.8 | |
| Maintenance and repairs to pole lines | 0.2 | |
| Pumping | 6.2 | |
| Ventilation | 1.7 | |
| | 394.0 | |
| | | $3.94 |
| | | 0.78 |
| | | $4.72 |
| Administration, taxes, safety, and depreciation | | 0.719 |
| Grand total | | $5.439 |

*West End, Tonopah.* (*Mining and Scientific Press*, Aug. 16, 1913)

| | |
|---|---|
| Superintendent and foreman | $0.135 |
| Breaking | 0.802 |
| Timbering | 0.090 |
| Tramming | 0.372 |
| Hoist, etc | 0.200 |
| Ore loading | 0.233 |
| Ore sorting | 0.366 |
| Assaying, sampling, surveying | 0.091 |
| Surface, ore dump, drayage | 0.195 |
| Development | 0.862 |
| General expense | 0.554 |
| Miscellaneous | 0.262 |
| Total | $4.162 |

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*Bunker Hill and Sullivan.* (*Engineering and Mining Journal*, June 14, 1913)

Large replacement deposits of sulphide ores in quartzite. Overhand stoping with stulls or square sets.

| | Labor | Supplies |
|---|---|---|
| Superintendents, blacksmiths, and supply men | $0.177 | ..... |
| Timbering | 0.086 | $0.215 |
| Miners | 0.367 | ..... |
| Carmen | 0.052 | ..... |
| Shovelers | 0.366 | ..... |
| Power | 0.027 | 0.045 |
| Repairs | 0.026 | ..... |
| Explosives | ..... | 0.075 |
| Illuminants | ..... | 0.020 |
| Lubricants | ..... | 0.003 |
| Iron and steel | ..... | 0.012 |
| Miscellaneous supplies | ..... | 0.035 |
| Wood | ..... | 0.034 |
| Stable | ..... | 0.001 |
| Total | $1.101 | $0.440 |

Grand total $1.541

*Stewart Mining Co., Cœur d'Alène.* (*Mining and Scientific Press*, Apr. 26, 1913)

| | |
|---|---|
| Cost of mining and development | $2.15 |

*Ferreria Mine, Rand, South Africa.* (*Mines and Minerals*, March, 1911)

Overhand rill stoping, shrinkage.

| | |
|---|---|
| Tons for each machine shift | 10.48 |
| Pounds of explosives per ton, 57, cost | 16c. |
| Total cost of stoping, 77c., plus 25c. for timbering, | $1.02 |
| Stoping on contract, per ton | 55c. |

*Brakpan Mine, Johannesburg.* (*Mining and Scientific Press*, May 17, 1913)

Presumably underhand stoping.

| Mining: | |
|---|---|
| Stoping | $1.11 |
| Timbering and packing | 0.24 |
| Shoveling and tramming mine and dump | 0.66 |
| Transport, underground | 0.11 |
| Transport, surface | 0.01 |
| Hoisting | 0.20 |
| Pumping | 0.16 |
| Other charges | 0.17 |
| Development | 0.36 |
| Total | $3.02 |

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*Rand Mining Costs.* (E. M. Weston: *Engineering and Mining Journal*, Feb. 8, 1913)

Presumably all underhand stoping.

New Kleinfontein Mine.—With hand drilling each native breaks 1.2 tons a shift at a cost of $1.03. Of this amount the white labor cost $0.16, the native labor nearly three times as much, while food, etc., for the latter costs 22c. to 24c. per ton. Owing to easy nature of ground, explosives cost only a little over 14c. a ton. The average footage drilled per native each shift is 40 in. and the stoping width 4½ ft.

At the same mine large machines broke 10 tons a shift in a 70-in. stope at a cost of 97c. a ton broken; of this 97c., white drillers cost 23c., natives 11½ c., while their food, etc., cost 30c. Explosives cost 21c. a ton.

One of the Deep-Level Mines.—Hammer boys drilled 33 in. per shift and broke ½ ton in a 44-in. vein at a cost of $2.73 a ton. Explosives cost 30.3c. a ton. Large machines broke 7.2 tons a shift in a 55-in. vein at a cost of $1.87 a ton. Explosives cost 36.3c. a ton. Small machines broke 3.7 tons in a 4-ft. vein at a cost of $2.43 a ton. Explosives cost 40.5c. a ton.

One of the Large Outcrop Mines of the Central Rand.—Each native drilled 48 in. and broke 1.5 tons a shift in a 63-in. vein at a cost of $1.22 a ton. Of this explosives cost 18.2c. Large machines broke 19.5 tons a shift in an 83-in. vein at a cost of 77c. Of this, explosives cost 20.2c.

The above costs are for stoping only. The total mining costs ran from $2.25 to $5 a ton.

### *Portland Gold Mining Co., Cripple Creek, Colo.*

Overhand stoping with stulls and square sets.

| | Per Ton Broken | |
|---|---|---|
| Tramming | $0.17 | |
| Hoisting | 0.10 | |
| Sorting | 0.14 | equal to 30c. per ton of ore through ore house. |
| Assaying, engineering, superintendence, and general | 0.30 | |
| Stoping | 1.63 | |
| Total | $2.43 | |

About 44 per cent. of total ore broken is trammed; 29 per cent. of ore trammed is sorted, equal to 12.7 per cent. of total.

The costs for the year 1906 at the Portland mine as given in *Ore Mining Methods*, by W. R. Crane, were as follows:

| | Cost per Ton |
|---|---|
| Labor | $1.142 |
| Machines | 0.270 |
| Tramming | 0.029 |
| Explosives | 0.380 |
| Hoisting | 0.230 |
| Supplies | 0.036 |
| Superintendence, assaying, etc. | 0.450 |
| Total | $2.537 |

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The labor cost was subdivided as follows:

| | |
|---|---|
| Machine men | $0.4761 |
| Trammers | 0.3214 |
| Pipe and track men | 0.0357 |
| Timbermen | 0.1666 |
| Timber helpers | 0.1428 |
| Total | $1.142 |

*Temiskaming Mining Co., Cobalt, Canada.* (*Engineering and Mining Journal*, May 25, 1912)

Narrow veins in schist. Overhand stoping.

Mining and timbering $1.85, of which labor was 66.5 per cent. and power 13.5 per cent. Dynamite cost 33.7c. a ton, fuse 2.6c., candles 3c., drill repairs 21c., steel 14.7c.

*Hollinger Mine, Porcupine, Canada.* (*Mining and Scientific Press*, May 3, 1913)

Country rock, fine-grained compact schist. Veins 2 to 6 ft. wide, containing quarts in stringers and bunches. Overhand stoping.

| | |
|---|---|
| General and superintendence | $0.179 |
| Diamond drilling | 0.027 |
| Stoping and driving | 1.969 |
| Timbering stopes | 0.219 |
| Tramming | 0.551 |
| Drainage and pipes | 0.092 |
| Hoisting | 0.170 |
| Dumping | 0.063 |
| Drill steel | 0.298 |
| Assaying, sampling, and surveying | 0.064 |
| Change house and lights | 0.013 |
| Handling explosives | 0.025 |
| Handling waste | 0.016 |
| | $3.686 |

In the Kalgoorlie district, Western Australia the ore occurs in strong veins in quartz-dolerite. The width of the veins varies but averages about 12 ft. The stoping methods employed are overhand rill stoping with filling, flat-back stoping, and shrinkage stoping. The following costs are of mines in this district.

*Great Boulder Perseverance.* (*Engineering and Mining Journal*, May 25, 1912)

Average stoping width 12.94 ft.

| | |
|---|---|
| Wages and contracts | $0.92 |
| Explosives | 0.156 |
| Drill parts and air lines | 0.0434 |
| Candles | 0.0182 |
| Air for drilling | 0.1056 |
| Not specified | 0.4682 |
| Total cost | $1.71 |



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*Ivanhoe Mine.* (*Mining and Scientific Press*, May 24, 1913)

| | |
|---|---|
| Ore breaking | $1.38 |
| Filling stopes | 0.27 |
| Tramming and hoisting | 0.58 |
| Development | 0.52 |

*Kalgurli Mine.* (*Mining and Scientific Press*, Mar. 8, 1913)

| | |
|---|---|
| Labor: | |
| Superintendence | $0.04 |
| Breaking ore | 0.66 |
| Timbering stopes and chutes | 0.04 |
| Loading and tramming | 0.42 |
| Filling stopes | 0.16 |
| Tool sharpening, etc | 0.05 |
| Sundries | 0.02 |
| Total labor | $1.39 |
| Stores: | |
| Tools, steel, drill renewals | $0.03 |
| Candles | 0.02 |
| Explosives | 0.18 |
| Timber | 0.04 |
| Assays | 0.01 |
| Sundries | 0.01 |
| Total stores | 0.29 |
| Hoisting, machine drills | 0.24 |
| Grand total | $1.92 |

*Oriental Consolidated, Unsan, Korea.* (*Mining and Scientific Press*, Apr. 30, 1910)

| | | | |
|---|---|---|---|
| Mining timbers | $0.254 | Picks, shovels, hammers | $0.009 |
| Firewood | 0.133 | Miscellaneous (rope, pipe, cars, etc.) | 0.046 |
| Lumber | 0.105 | Total supplies | $0.823 |
| Dynamite | 0.097 | Assays | 0.006 |
| Candles | 0.094 | White labor | 0.117 |
| Fuse | 0.094 | Korean labor | 0.612 |
| Detonators | 0.011 | Outside expense, shops, stables, etc | 0.030 |
| Charcoal | 0.013 | Total | $1.590 |
| Lubricants | 0.010 | | |
| Drill steel | 0.011 | | |
| Bar, sheet, track iron | 0.010 | | |

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### *Consolidated Mercur Gold Mining Co., Mercur, Utah.* (*Mining and Scientific Press*, Sept. 25, 1909)

Fifteen to 70 ft. vein of soft ore in hard cherty limestone. Dip 10° to 30°. Sublevel caving, with sublevels 14 ft. apart.

Total cost of mining $1.53.

### *Alaska Treadwell*

From the costs of 1912:

| | | | |
|---|---|---|---|
| Machine drillers | $0.143 | Iron and steel | 0.003 |
| Laborers, powdermen, etc | 0.128 | Lumber and timber | 0.003 |
| Foremen | 0.009 | Compressed air | 0.018 |
| Blacksmiths | 0.006 | Power | 0.007 |
| Machinists, carpenters, timbermen | 0.005 | Mechanical repairs | 0.002 |
| Powder | 0.146 | Blacksmith shop | 0.005 |
| Fuse and caps | 0.016 | Miscellaneous | 0.041 |
| Candles | 0.006 | Total | $0.544 |
| Machine drill supplies | 0.006 | | |

Cost of stoping only

### *Anaconda Co., Butte, Mont.*

Cost of mining by overhand stoping with square sets and filling was $3.77 in 1911.

### *Braden Copper Mine, South America.* (*Mining and Scientific Press*, Dec. 4, 1909)

Ore breaking including superintendence and general charges was 41c. Machine drilling with 2¼-in. drill was at the rate of 31 ft. per man-day at a cost of 2.025c. a foot. Hand drilling was done at the rate of 13.8 ft. per man-day at a cost of 8.2c.

### *Cost of Development*

### *Bunker Hill and Sullivan, Cœur d'Alêne.* (*Engineering and Mining Journal*, June 14, 1913)

11,050 ft. of vertical and horizontal development:

| | Per Foot | Per Cent. |
|---|---|---|
| Foremen, blacksmiths, etc | $0.312 | 4.4 |
| Miners | 2.500 | 35.0 |
| Shovelers | 1.650 | 23.1 |
| Explosives | 0.990 | 13.8 |
| Timber and lagging | 0.400 | 5.6 |
| Power, labor and supplies | 0.524 | 7.33 |
| Not specified | 0.774 | 10.8 |
| Total | $7.15 | 100.0 |

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### *Portland Gold Mining Co., Cripple Creek, Col.*

Cost of drifts or levels, $6.12; crosscuts, $6.23; winzes and raises, $8.60. In the *Cost of Mining* by Finlay, p. 380, the cost of 896 ft. of 5 by 7 ft. drifts is given as follows:

| | Per Foot |
|---|---|
| Tramming | $1.00 |
| Pipe and trackmen | 0.14 |
| Machine men | 1.88 |
| Total labor | $3.02 |
| Other costs: | |
| Use of machines, air, etc | $0.97 |
| Repairs, cars, etc | 0.08 |
| Explosives | 1.43 |
| Hoisting | 0.46 |
| General expense, surveying, assaying, bosses | 0.58 |
| Grand total | $6.20 |

### *Montana-Tonapah Mining Co.* (*Mining and Scientific Press*, Oct. 9, 1909)

Drifting, $6.56 a foot; crosscuts, $5.44; raises, $4.65; winzes, $11.92. Cost of development subdivided as follows:

| Labor: | Per Cent. |
|---|---|
| Breaking | 27.10 |
| Timbering | 4.38 |
| Hoisting and dumping | 6.32 |
| Foremen and shift bosses | 1.75 |
| Blacksmith, sharpening | 1.75 |
| Shoveling and tramming | 22.90 |
| Surveying | 1.35 |
| Watchman | 0.52 |
| Storekeeper and timekeeper | 0.35 |
| Diamond drill hole | 4.38 |
| Supplies: | |
| Breaking | 21.20 |
| Timbering | 1.23 |
| Hoisting and dumping | 2.86 |
| Hoisting, electric power | 3.56 |

*At a Western Gold Mine* 302 ft. of development work took 238 miner's shifts, 122 shifts of muckers and trammers, and 105 machine-drill shifts. Explosives cost $1.41 a foot and air for drills 25c. a foot or 75c. for each machine-drill shift.

*At the Standard Mine, Bodie, Cal.*, drifts cost $3.07 a ft. with an average of 1.3 ft. a shift, and raises cost the same. At the *Commercial mine at Bingham, Utah*, in 1912, drifts cost from $5.83 to $8.13 a ft., and raises cost $5.56. At the *Nevada Hills mine, Fairview, Nev.*, drifts and crosscuts cost $9.75 a foot, raises $12.50, and winzes $30. At the *Great Boulder Perseverance, Kalgoorlie*, drifts cost $12.90 a foot, and crosscuts $13.80. At the *Braden Copper Co., South America*, development cost $3.54 a foot.

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*At Cananea* in hard quartz a 4½ by 6½ ft. drift took 7.8 lb. of powder for each foot, and a 6 by 11 ft. raise took 8.3 lb. At the *Pittsburg-Silver Peak* a 4½ by 5 ft. raise took 9 lb. powder for each foot, and at the *Erie Consolidated* in slate and quartz a 5 by 7 ft. raise took 6.65 lb. In drifts driven by hand work in medium-hard ground the cost was $4.50 to $5.50 a foot of which the labor was 77 to 85 per cent. and explosives 15 to 23 per cent. The rate of advance was 1 to 1½ ft. a shift.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Experimental Leaching at Anaconda

BY FREDERICK LAIST AND HAROLD W. ALDRICH, ANACONDA, MONT.

(Salt Lake Meeting, August, 1914)

THE object of the construction and operation of the 80-ton leaching plant was to test out the leaching of sand tailings on a large scale and, if possible, determine a definite method of operation, and the best construction for the larger unit which is now being built. It was also of importance to obtain some practical data as regards cost items. The 80-ton plant was not expected to be a commercial success. A leaching process on such low-grade material necessitates the treatment of a very large tonnage. Such a cost item as labor, for instance, is entirely out of proportion in such a small plant. The amounts of acid, salt, scrap iron, and fuel for roasting, however, are the same per ton in an 80-ton plant as in a 2,000-ton plant. These costs were definitely established. Also the percentages of extraction and grade of tailings were definitely determined for a large-sized unit.

Following will be found the description of the plant, the method of operation and the results obtained. A general view of the plant is given in Fig. 1.

The bins consisted of one coal bin with a capacity of 30 tons, one salt bin with a capacity of 20 tons, and a "mill tailings" or feed bin which held 70 tons. The coal and salt bins were placed with their floor level even with the firing floor of the roasting furnace. The feed bin, to give it greater capacity, discharged about 12 ft. below this point, to a 12-in. conveyor belt, which, in turn discharged into a pug mill. This fed another conveyor belt, which discharged into the top hearth of the furnace.

The furnace was an ordinary six-hearth MacDougall roaster, 20-ft. in diameter. It was equipped with two fire boxes, on opposite sides, the flames entering on the third hearth. An induced draft was obtained with a Buffalo blower, worked as an exhaust fan. Just above the top hearth was an annular flue around the whole circumference of the furnace. Slots at intervals of about 18 in. led from this flue down into the top hearth. The slots were 2 in. wide by 6 in. long. This annular flue proved rather unsatisfactory, as it tended to fill up with flue dust and choke the slots, thus cutting down the draft. The volume of gas through

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the furnace was 3,500,000 to 4,000,00 cu. ft. per 24 hr. at standard conditions of temperature and pressure. The temperature of the outgoing gases was about 180° C.

The furuace was air cooled, 15-lb. air entering the shaft through a 2½-in. pipe. The pressure in the arms themselves was 2½ lb. per square inch; 850,000 cu. ft. of cooling air at standard conditions of temperature and pressure was used for this purpose. This, we believe, was a great deal more than was necessary, but we did not want to take any chances on losing any furnace arms, before the experimenting was finished.

There were four drop holes in the wall between the second and third floors, and six between the fourth and fifth floors. Four holes seem to be ample. The discharge from the furnace to the cooler, was an 8-in. pipe at the outer edge of the bottom hearth. A nearly dust-tight chute led

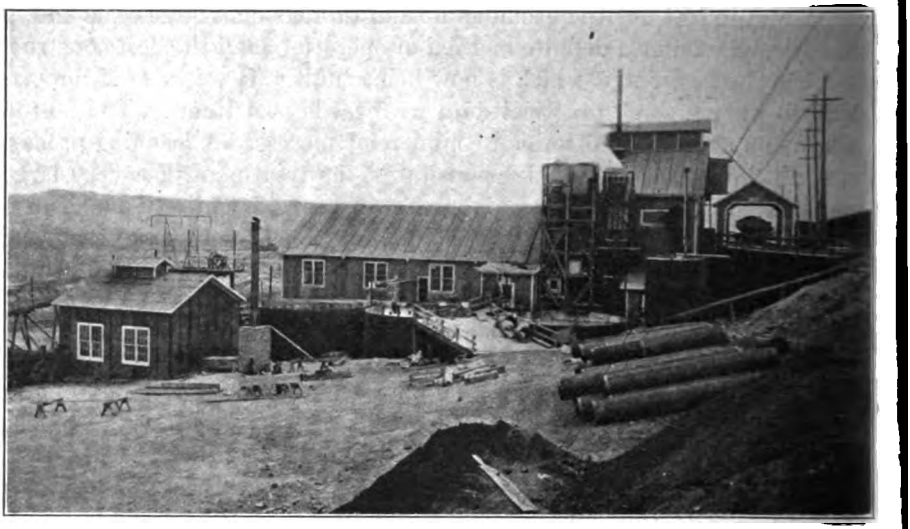

FIG. 1.—VIEW OF 80-TON LEACHING PLANT.

into the cooler. All dust made in the cooler was, therefore, drawn back into the furnace.

The cooler, shown in Fig. 2, was a rotating drum, built of ¼-in. boiler plate, 19 ft. long and 2 ft. in diameter. It had a slope of ¾-in. to the foot, and made 10 rev. per minute. At the lower end, cold water entered an annular chamber through a stuffing box, and from this chamber 26 iron pipes 1-in. in diameter carried the water to the upper end and discharged into a circular launder 18 in. from the end, and on the outside of the drum.

The calcine entered the cooler at 260° to 370° C., depending on the amount of sulphur in the feed, and was discharged at about 45° C. The rate of feed, of course, also affected the temperature of the dis-

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charged calcine. At 80 tons of an average feed, the cooling water required to cool the calcine to 45° C. was 25,710 gal. per 24 hr. To cool the same to 37.5° C. required 34,666 gal. per 24 hr. The cooler discharged into a 6-in. screw conveyor, into which a water spray, to settle the dust, was turned. The screw conveyor discharged to a series of belt conveyors, which carried the moistened calcine to the hoppers of the distributing launder above the tanks.

The distributing launder extended from the center to the circumference of the tank, where it was supported by a small fiber wheel. It

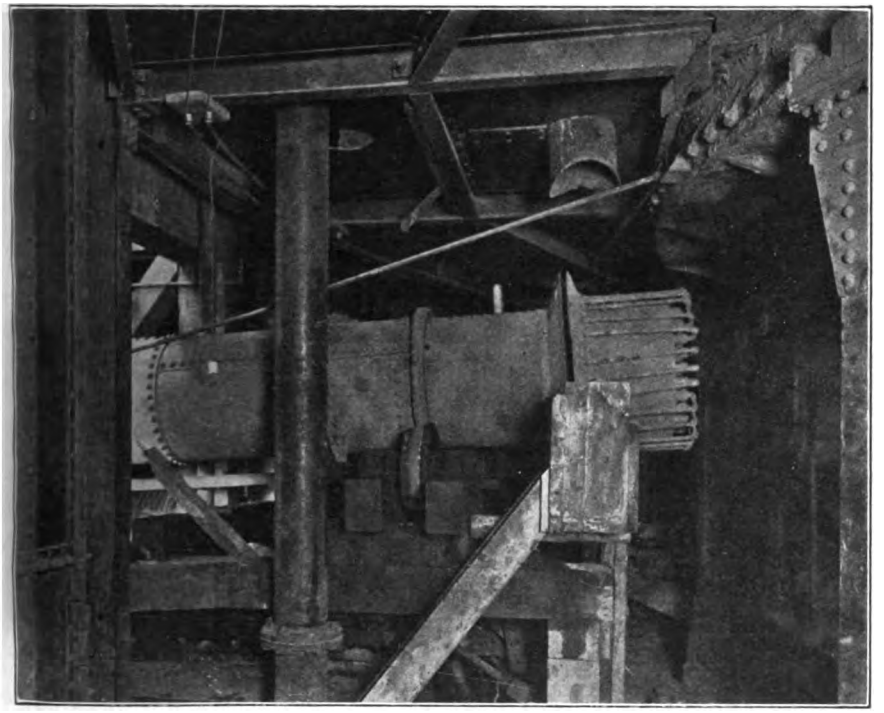

FIG. 2.—CALCINE COOLER.

was driven from the conveyor belt, through a series of bevel gears. The distributor was an iron launder 6 in. wide by 8 in. deep, with slots in the bottom spaced 18 in. apart. There was a slide on each slot, so that any or all of them could be closed. A 6-in. screw conveyor, resting down in the launder, moved the calcine from the center of the tank, outward.

Two leaching tanks, 32 ft. in diameter and 12 ft. deep, were placed in line with the furnace. A conveyor belt ran over each one, so that by moving the distributor from one tank to another, and by means of a switch gate, either tank could be filled. The tanks were of redwood. Each had five discharge gates, 12 in. in diameter, one in the center, and

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one in the center of each quarter of the tank. Suitable launders were provided underneath the tanks, for carrying away the leached tailings. Each launder had a stream of water entering through a 12-in. dirty-water main, from the concentrator. Valves were provided, so that all the water could be sent down one launder if necessary. Three-inch dirty-water pipes also led to the top of the tanks to furnish water for sluicing purposes. This was done with two 3-in. rubber fire-hose lines.

The filter bottoms were made with 2 by 4 in. pieces, spaced every 1 ft. across the tank, and 1-in. slats placed with 1-in. spaces across the tank in the other direction, and nailed to the 2 by 4 in. pieces with copper nails. Then the whole bottom was cut into sections convenient for handling, when taking the bottom up. A filter medium of two layers of coarse cocoa matting was spread over this. The bottom of the tank under the filter had a lead lining, which came up the sides 6 in. Suitable drains for the solutions were provided under the filter. Pipes for carrying concentrated acid were provided for each tank, and steam connections for heating the solutions, as they went on.

Two 10 by 10 ft. iron tanks were provided for holding the stock of concentrated acid. All connections and fittings for concentrated acid were of iron. As a safety, there was a lead plug which fitted into the discharge hole in the tank. This was operated from the top with an iron rod connected to the plug. There was a gauge on each tank, actuated by an air-tight iron float.

Two 28 by 10 ft. lead-lined redwood tanks held the stock solutions for leaching, and a third tank the copper solution. A heavy lead steam coil inside the tank was used to heat the solutions to prevent freezing in very cold weather. Drains from the tanks led to the air lifts, of which there were three, made of 5-in. lead pipe and lifting about 16 ft. The lower elbows were in a 16-ft. pit. The air lifts used 90-lb. air, but only because all of the available 15-lb. air was used for the furnace cooling. A very slight turn of the valve gave sufficient air. The discharge from the lifts was into a lead-lined box above the tops of all the tanks, leaching and solution. The launder system was such that it was possible to transfer solution from any tank to any other tank in the plant. A circulating air lift was provided for each leaching tank. It was not necessary to have a pit for these, however, as they only had to actually lift against a pressure of a few inches. They were connected to the bottom of the tank, under the filter, and discharged into the top, over the calcine. Each would handle 250 gal. or more per minute. Under the air-lift discharge box was a lead-lined heater box, into which live steam was injected to heat the solutions. Some of the launders were lead lined and others were not. In some places, the lead would creep and split at the seam. The wood launders with no lining were satisfactory for the time the plant was in operation, about six months, and are probably good for

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a long time yet. We did not try to protect them with paint in any way. A 28-ft. Oregon fir tank was put up as a spare acid tank. It was slow in taking up, but for an unlined tank, it did very well. It had been put up and taken down twice before this, so it was not in the best condition. The iron hoops for 4 ft. from the bottom were protected with lead strips.

A small 9 by 10 ft. wooden tank, with agitator equipment, was also put up for prospective sponge-iron precipitation. It was not lead lined and gave no trouble from leaks.

In two of the large lead-lined solution tanks we added salt directly to the solution. It settled to the bottom, and in a day or so the tanks commenced to leak. After repairing, we discontinued this method of adding salt and were troubled no more. It seems that the hydrochloric acid liberated attacked the lead very materially. If a leak does occur in a lead-lined tank the wood backing does no good whatever. It is so thoroughly dried, of course, that it has no holding power for the solution.

To each of the solution tanks, there was an acid pipe connection and a fresh-water connection. A 1½-in. lead pipe with a bend of 1 ft. of its length at the bottom of the tank furnished 90-lb. air for agitation and mixing. This did not work very well. The air came to the top of the solution at the point where it entered, and did not agitate a very large proportion of the tank. We could get no swirling effect at all.

The lead lining and piping showed no appreciable corrosion in the six months of operation, excepting, as was stated before, where a large amount of salt came in contact with the lead, and where live steam impinged on the lead in the presence of acid solution. A lead pipe carrying steam will not stand up in our solutions when they contain 3 to 4 per cent. acid, but in the solutions after precipitation, which carried 1 per cent. or less, no corrosion could be observed in a month's continuous running. The lead air lifts showed no appreciable wear at the elbows, where the most corrosion would be expected. The lead pans in the bottoms of the leaching tanks were of no benefit, when the scheme whereby the acid solution was sprinkled on, instead of the tank being saturated, was abandoned. The leaching tanks leaked pretty badly, but this was due to faulty construction, rather than to any fault of the wooden tanks. In several places it was necessary to protect the hoops with lead strips. The air lifts worked very satisfactorily, handling a large amount of solution very easily, except when they started "foaming." This was something which we could not account for definitely. At times, the solution coming up the air lift gave a great deal of trouble because of this foam. The solutions averaged about 1.22 sp. gr. and were more or less viscous. Cold solutions seemed to foam much worse than the warm ones, so it seems as though the viscosity was what caused the trouble.

The precipitation launders were two lead-lined troughs, side by side,

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each 75 ft. long, by 3 ft. wide by 2½ ft. deep. The copper solution entered the upper end of one launder and returned in the other to an air lift, which raised it into the copper-solution tank. It was circulated in this way until the copper content was low enough, say, 0.06 per cent., to be returned to the leaching system. The precipitating launders were equipped with false bottoms 4 in. above the floors to give room for the cement to settle and leave all of the iron free. A settling tank was provided and, when cleaning up, the end gates were removed and the cement washed or pushed out of the launders into it. With fresh, clean iron, this equipment will precipitate 8 to 10 gal. per minute from 2 per cent. copper to 0.1 per cent.

Owing to the entirely new and experimental nature of this work, it was found necessary to make a great many changes from the procedure which had been planned out in advance. This was true in the case of handling the material, the roasting, and the scheme of leaching.

As the plant was originally built, the feed to the furnace, as well as the calcine from the furnace to the leaching tanks, was to be handled in small bucket elevators. These gave a great deal of trouble. The nature of the material to be handled showed very soon that the elevators would not do the work. The feed has a good deal of fine material, though much coarser than slime, which tends to pack and build up as it drops to any surface. In the case of the feed elevator, the buckets would fill up and not discharge, and finally would have to be cleaned out. We endeavored to moisten the calcine as it came from the furnace, to settle the dust, and it acted much the same way as the feed. Unless the calcine was dampened as it came from the furnace, the elevator raised so much dust that it was almost impossible to work in the building. These elevators were then discarded and belt conveyors substituted. It was necessary to put in two belts to get the feed to the top of the furnace. A pug mill was installed between these two belts, to mix in a certain percentage of slime, which was to be roasted and leached with the sand.

As originally built the furnace was designed for oxychloride roasting and salt was added on the fourth floor. All the doors were machined and made air tight. There was a combustion chamber between the second and third floors. The flames entered this chamber from the fire boxes and then traveled down through six holes 8 in. in diameter, spaced around the floor of the combustion chamber, to the third floor, and up through the drop holes to the second and to the top and out. The third floor was sealed from the fourth. A 3-in. drop hole on each side of the shaft, near the center, formed the bottom of a small hopper, which was set down in the brick. There was an adjustable gate on this drop hole, which could be set so the hopper had sand in it all, or nearly all, of the time, thus preventing any gases from the fourth hearth being drawn above into the third hearth. A fan furnished the draft for the

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upper three floors, and another forced the gases from the lower four through a scrubbing tower. This was for the purpose of separating the chloride or copper- and silver-bearing gases from the sulphur or waste gases.

There was a seventh floor provided. This was nothing but a flat water jacket to cool the calcine. There were also copper rakes on the seventh floor, in case it was desired to use a water spray to assist in the cooling and also settle the dust.

The bottom floor discharged through a 6-in. hole into a cast-iron pipe which contained a screw conveyor. This closed conveyor carried the material from the center of the furnace to the calcine elevator. In this way no air could enter the lower, or chloridizing floors, except what was allowed to go in through an open door regulated to suit conditions. Theoretically, this looked very good, but under operating conditions a great many difficulties arose. The use of the combustion chamber required much more fuel than direct firing into the third hearth. At the time the combustion chamber was taken out, the sealed drop holes between the third and the fourth floors were also discarded, and a regular center drop hole was put in. Then the upper fan was the only one used and all the gases were combined. The fuel percentage immediately dropped down to what it should have been. This was due to the greater efficiency without the combustion chamber, and also very largely to the benefit derived, on the top two floors, from the hot gases rising from the fourth, fifth, and sixth floors, where a large part of the oxidation went on. When the furnace was run with the fourth floor at 930° F., which temperature gave the best results, there was no oxidation on the second floor from the top. The sand was heated up, of course, and just about hot enough to ignite the sulphur as it dropped to the third or fired floor. The most of the oxidation took place on the fourth floor.

When the hot calcine dropped to the water-jacket floor the jacket bulged and the rakes cut through, causing it to leak. This caused a great deal of trouble, as the water-soluble copper in the calcine went into solution and precipitated out on the iron, taking it into solution. The water jacket was also a very inefficient cooler. It was discarded, and a discharge hole cut in the outer edge of the sixth floor.

Then the problem of an efficient calcine cooler presented itself. A water jacket 20 ft. long and 4 ft. wide was built, and connected to a Wilfley table motion. This scheme worked poorly in a mechanical way, as the weight of water in the jacket was considerable, and the jerk of the jacket caused a sort of a hydraulic-ram action inside it, which caused many leaks. Also, the cooling efficiency was not very good. A layer of the finer calcine formed on the jacket and the coarser material merely slipped down over the top of this, getting very little cooling. A coil of cooling pipes, the whole length and width of the jacket, was laid on

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top of it, and calcine allowed to run down over these pipes. In this scheme the calcine acted the same way, most of the calcine never touching the jacket, but sliding down over the top of the lower calcine layer. The next method tried was the rotary cooler described in the preceding pages. It worked very satisfactorily, giving an efficient cooling and requiring very few repairs. The cooler discharged into a screw conveyor. About 1 per cent. water was sprayed on the calcine at this point to settle the dust. Mechanically the screw worked very well, but the water-soluble copper ate out the iron rapidly, and made it necessary to replace it often.

A push conveyor was built to replace this screw. After proper adjustment, it worked quite well, mixing in the water nearly as well as the screw. In comparison with this, a rotating wooden drum 2 ft. in diameter and 12 ft. long, was also tried. This gave the best results of any arrangement so far, for a short conveyor, particularly where it was necessary to mix in a small amount of water, for dust settling or cooling purposes. No iron could come in contact with the moistened calcine, and the repairs were very low. It rotated slowly and mixed the water in perfectly, about 1 per cent. water being required to prevent the calcine from dusting.

The scrubbing tower for the chloridizing gases did not work very efficiently and an electrical precipitation unit was installed. It was put in by Walter Bradley, of the Research Corporation, and was designed to handle all of the gases, chloride, sulphur, and coal gases. The treater was built after the design of Dr. F. G. Cottrell, of the Bureau of Mines, and worked satisfactorily, as 90 per cent. of the copper and a considerable amount of acid were recovered from the fumes.

After a 50-day successful run, it was decided to discontinue the feeding of salt into the furnace, for a time, and see what results could be obtained. This method offered many advantages. No valuable metals were volatilized, and no accretions formed in the furnace to cause trouble. The results from the straight oxidizing roast proved to be so good that it was continued for 70 days. The extraction on the copper was even better than on the chloridizing roast, but the silver recovery was not so good. However, the excess silver that would be saved by roasting with salt would not pay for the increased expense and bother entailed by a salt roast. The leaching solutions carry salt, and a great deal of chloridizing is done in the tanks.

The results from the oxidizing roast have this advantage over those of the chloridizing period: The oxidizing period was the later one, and naturally everything ran more smoothly and better results were more easily obtained. Also there was a leaner feed to work on during the chloridizing period.

An attempt was made to roast and leach a 10 per cent. slime tailings

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mixture. The slime, dewatered on an Oliver filter, was carried by a conveyor belt to a pug mill, where it was mixed with the coarse tailings. The mixture was fairly uniform, but some pieces of slime the size of a walnut entered the furnace, did not break up, and were roasted on the outside only. The slime, even after roasting, decreases the percolation rate so much as to make its treatment by mixing with the sand impossible.

The operation of the furnace, after everything was systematized, was very simple. The firing takes a certain amount of skill and experience, and the regulation of the furnace also takes some practice. A pyrometer was used on the fourth floor, but an experienced man could regulate the furnace and keep it within 30° of the required temperature, by the appearance of the lower three floors. To do this, of course, the same grade of feed must be maintained. A marked change in the amount of sulphur in the feed necessitated a readjustment of conditions in the furnace. One point is rather important; that is, not to let the temperature get too low, say 880° F., on the fourth floor, for if it once starts down it is very hard to get it back without lowering the amount of feed. The fire caused by the burning sulphur seems to go out, and then it has to be ignited again, using a disproportionate amount of coal.

The feed averaged about 3.00 per cent. sulphur and the calcine about 0.6 per cent. sulphur.

It was our experience that the higher temperatures, say 1,000° or more, caused a large amount of ferrite or insoluble copper; that is, insoluble in any acid but hydrofluoric; 900° to 930° F. seemed to give the best results.

The feed to the furnace averaged about 5 per cent. moisture, but very often went as high as 8 or 10 per cent. When this happened, the material on the top floor would bank up over the arms, instead of going through between the rakes. This always made it necessary to shut the feed off and let the rakes clear themselves. This trouble often limited the amount of feed it was possible to put through the furnace. With three or four arms on this floor, and with a feed carrying 0.68 per cent. copper and a proportionate amount of sulphur, the furnace handled very close to 100 tons per day, dry weight. The rakes on the first or top hearth were the only ones which showed any appreciable wear during the six months' operation. They were replaced about every four or five weeks. The blades and blade holders should be used there, as replacing these is much cheaper than replacing the whole rake.

Fire bricks were used in the fire boxes and in the construction of the second and third floors. All the rest of the lining and the floors were built of ordinary red brick. No special endeavor was made to increase the water-soluble copper. It seemed to affect the acid consumption very little and electrolytic precipitation is out of the question at present,

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since the solutions carry chlorides. About 40 per cent. of the copper was soluble in water.

To give a full idea of the experimenting and changing in the leaching schemes, it seems best to take them up separately and show what was done on each.

In the experimental work done in the summer and fall of 1912, in a small experimental plant, a leaching scheme was worked out which seemed to give excellent results. This method consisted of sprinkling the No. 1 or weak solution on the charge, then letting it stand a few hours and following with the No. 2 or strong solution in three portions. This was sprinkled on in the same way, much as a lawn would be sprinkled, except that a few hours were allowed to elapse before the addition of each portion. The solution was allowed to drain out as fast as it would and no hydrostatic pressure built up in the tank. The No. 1 solution was precipitated, made up in acid, and used as the No. 2 solution for the next leach. The No. 2 solution was returned and used for the No. 1 in the next leach.

Accordingly, an automatic sprinkler was designed for this plant, and a lead pan was put into the bottom of the leaching tank, to prevent leakage.

The first six leaches were all subjected to the sprinkling system, and were treated in much the same way, except variations in acid and salt strengths. These show in the table of tank leaches. As far as the "sprinkling system," goes, it seems to have done about as good work as the later "flooding system," considering the class of calcine used at that time. The furnace was not making very good calcine until Oct. 20, and the first four leaches had been completed at that time. However, the flooding system offers fewer difficulties and dispenses with all automatic sprinklers, and, if anything, does better work. When a tank is flooded with solution, all parts of the calcine are apparently more apt to be saturated with acid than by the sprinkler system. However, the average results of the first six leaches come as near to the results from the laboratory leaches as those of the later ones.

Leach No. 7, the first one in which the flooding system was used, gave rather poor results. This was due entirely to improper washing. The tailings sample after washing in the laboratory gave a result of 0.096 per cent. copper in comparison with 0.135 per cent. copper after the wash in the leaching tanks. Up until this leach, at times there was a good deal of fine material and a good deal of slime in the feed. This hindered the percolation, and resulted in poor extraction and imperfect washing. Classifiers were installed in the mill to give a more perfectly deslimed feed. Leach No. 8 was very similar to leach No. 7. However, it gave a considerably better extraction percentage on the copper, but

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this was probably due to the fact that the calcine was richer. This leach did not get a perfect wash.

In leach No. 9, a new scheme was tried. Air under 90 lb. pressure was connected under the filter bottom of the tank. The No. 1 solution was added, circulated down to 0.5 per cent. $H_2SO_4$, and replaced with the No. 2 solution. This was then drained for about 8 hr. In that time, most of the solution which would drain had come off. The air was then turned on for 8 hr. This was repeated three times, the air coming up through the warm, moist calcine being about 40° C. and it seemed as though its oxidizing action on the calcine was sure to benefit the extraction. This was tried in some of the later leaches, but seemed to help them very little, if at all. Leach No. 9 gave an exceptionally good extraction. However, it is doubtful, from later results, whether the aeration was responsible, as the calcine was very good and also the percolation. In this leach, the top and bottom of the tank were sampled separately. The bottom showed about 1 per cent. poorer extraction than the top.

Leach No. 10 was made on a 6-ft. depth, or 224 tons, of calcine. An attempt was made to determine if the shallower bed would equalize the results in the top and bottom of the tank. The extraction on the bottom was 2 per cent. poorer than on the top. On leach No. 11, a copper and silver balance was made, in which the calcine, tailings, and all solutions were measured, sampled, and assayed. This leach was the first of the oxidizing-roasted material. It had the same treatment as leach No. 10, except that one portion of the No. 2 solution was added under the filter bottom and forced up through the calcine. This method gave the bottom of the tank the benefit of the first application of the warm strong acid. The results show a much better extraction at the bottom than at the top of the tank. There is such a wide variation in the other direction, that it looks as though perhaps there was an error in sampling or analysis. On none of the succeeding leaches where this was tried was the extraction better at the bottom than at the top of the tank. This method is worth consideration, at any rate.

Leach No. 12 had two portions of acid added at the bottom and two portions at the top. In this case, the top gave an extraction nearly 3 per cent. better than the bottom, in spite of adding the solution at the bottom. Aeration also was used on this tank and the air heated before going in. Leach No. 13 had no aeration and all of the solution was added at the top. Leach No. 14 was started with an $H_2SO_4$ solution with no salt. It was planned to run this solution through, and determine whether it would dissolve enough of the copper so that it could be precipitated on the scrap iron and all of it be discarded. In this way, the chloride solutions would not have so much copper to take up, and, therefore, when



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precipitated, would not take so much iron in solution or become so foul. This scheme would cut down the amount of discard solution and salt loss. The first solution only dissolved about one-half of the copper and the cycle was prolonged to such an extent that it seemed advisable to abandon the idea. A balance was also made on this leach and 98.5 per cent. of the copper was accounted for. In this leach the silver extraction was poor. An attempt was made to put the wash water on at the bottom; it was then changed about and started on at the top. In this way, some silver which was held in solution by the salt was precipitated in the tailings, due to the dilution with the wash water. The silver results on this leach were thrown out when averaging the results.

Leach No. 15 gave the best results of all the leaches. It had rather a high calcine—0.713 per cent. Cu—and percolated very readily and had about a six-day treatment. In this case, the strong acid necessary to bring the solutions up to strength was added to the stream of solution as it went on to the charge; 55 lb. of acid per ton of calcine was added in this leach. The No. 2 solution was put on and drained, then another portion put on to re-saturate it, and to this portion enough acid was added so that a zone of 15 per cent. acid traveled down through the charge. The repeated draining and re-saturating seems to help considerably in the extraction. A new scheme was tried in the washing which seemed to give excellent results, and was continued through all the rest of the leaches. Salt, to the amount of 1 per cent. of the weight of the charge in the tank, was spread on top of the leached calcine just before the wash water was added. This strong zone of salt solution traveled down through the charge and dissolved or washed out any silver chloride remaining. The salt going down first, went into the solution tanks to take the place of the discard solutions; thus no, or very little, salt found its way into the wash-water tanks.

Leach No. 16 was a very successful one, the percolation being very good. About 50 lb. of acid was used per ton of calcine, and added to the No. 2 solution as it went on. Aeration was used. The No. 1 solution was added and circulated down to 0.6 per cent. $H_2SO_4$ from 4.2 per cent. Then it was replaced with one-half of the No. 2 solution. This first half was then drained, and followed down with the second half of No. 2 solution, this portion having part of the strong acid added to it. The tank was then drained to about 10 per cent. moisture and aerated for 8 hr. The air coming up through the calcine, moistened with, say, an 8 to 10 per cent. $H_2SO_4$ solution, was expected to give excellent conditions for oxidation and subsequent solution. The air may be, and probably is, of some advantage, but it is so small as to be almost inappreciable. Also, aeration tends to prolong the cycle considerably. The repeated draining and re-saturation are thought to be more beneficial than the aeration. When a charge is drained and then re-saturated, all particles are bound to get a fresh

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supply of acid, while circulation does not accomplish quite the same end. During circulation, the solution travels slowly down through the charge, taking the easiest course, and if there are any softer portions they get more than their share of the acid, and harder places get less. In this leach there was about 1 per cent. difference in extraction between the regular extraction and that using a laboratory washed tailings as a basis for calculation.

Leach No. 17 was treated the same as No. 16, with the exception that the whole amount of acid was added to the second half of the No. 2 solution as it went on. This brought the acid strength up to about 18 per cent. $H_2SO_4$ for this portion. This was allowed to travel down through the calcine, replacing the first half. The charge was then aerated and the first half of the No. 2 solution returned, and the total No. 2 solution circulated was down to the same acid value as the No. 1 solution had when it went on. There was rather a large difference between the extraction on the top and bottom, and the washing could have been improved upon also.

In leach No. 18, the same solutions which were used in No. 17 were used without precipitating out the copper. They were made up in acid strength, however. The idea was to see if it was possible to run the solutions higher in copper content, without interfering with efficient leaching. The results were not very satisfactory. Using the richer copper solutions made washing of the charge much more difficult and also the variation between extractions on the top and bottom was larger than it should have been.

Leaches 19 and 20 were run side by side and at the same time. Both were treated with No. 1 solution, and then one had No. 2 solution added to it, and its drainage of No. 2 solution was lifted to the top of the other charge. The two leaches were see-sawed back and forth in this way several times; that is, one was drained as fast as possible and the drainage put on top of the other tank, adding, of course, its proportion of strong acid. This was to accomplish the large number of drainings and re-saturations spoken of above, without the use of spare or sump tanks. The scheme worked very well as far as time and the handling of solutions went. It is possible to drain a 10-ft. depth of this material to about 10 per cent. moisture in 6 to 7 hr. Leach 19 had 49 lb. $H_2SO_4$ per ton added to it and 48 lb. was added to leach 20.

The sluicing of tailings is a very simple and cheap operation. Seven million gallons of water per 24 hr. will sluice and carry away all the tailings which two men can sluice out of five 12-in. gates in the bottom of a tank. This amount of water was available and used, but probably much less than this would be sufficient. The greatest difficulty is at the start, when a hole is first opened up. The sand caves in and it takes a very large amount of water to carry away this sudden influx of sand

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into the tail race. Two men could sluice 400 tons from one of the tanks used here in about 5 hr.

There is not much to be said with reference to the precipitating. The launders were filled with scrap iron and the solution circulated through them. At 30° C., a 2 per cent. Cu solution, carrying 0.7 per cent. $H_2SO_4$, was precipitated to 0.1 per cent. Cu when circulating at the rate of 8 to 10 gal. per minute.

This strong copper solution precipitated as a very good cement copper. It was flocculent, and, on wrought iron, merely touching the iron would break the copper off and leave a fresh iron surface. The cement ran from 75 to 85 per cent. copper.

The hard or cast scrap iron was also tried. At first, the copper seems to precipitate on it as fast as on the soft iron, but a layer of copper quickly plates on it, and the cleaning up is a pretty expensive operation. Every piece must be handled separately, brushed, and most often scraped, before the iron is fit for further precipitation. By using malleable iron scrap, sweeping is hardly necessary, if the iron can be moved a little occasionally. The copper drops off and to the bottom of the launder immediately. The iron consumption was about 1.15 lb. per pound of copper.

## General

A copper tube was used for all tailings sampling. Iron cannot be used because of the precipitation on it of any copper in solution. The copper tube was closed at the lower end and brought to a sharp point. A slot was cut its full length, and one side bent out a little to form a cutting edge. A piece of rubber hose was pushed into the sampler, just filling it, then it was driven down through the charge, the hose withdrawn and the tube turned several times. In this way, the cutting edge filled the tube with tailings representing that part of the tank. This sampler was checked several times, by opening up a hole to the bottom of the charge and sampling up the side of the hole with a wooden scoop. If there was any abrasion of the copper off the tube, it did not appreciably affect the sample.

It requires about 20 per cent. of the weight of the calcine in weight of solution to saturate the charge. This assumes the solutions at a specific gravity of 1.00, while in reality they have a specific gravity of about 1.25.

It requires about 40 per cent. of the weight of the charge in wash water for a reasonably good wash.

The acid consumption will not exceed 50 lb. $H_2SO_4$ per ton of charge and the salt lost in solution discard will not be more than 1 per cent. of the weight of calcine, probably much less. By figuring acid consumptions on each leach, and using acid content of each solution, before going on and after coming off the calcine, the acid consumption on the first

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14 leaches averages about 25 lb. per ton. On the last six, 50 lb. of acid per ton was added. This may have given sonewhat better results, but not very much so. It seems as though perhaps the extra acid of the 50-lb. consumption was used in taking up iron and alumina, and had very little effect on the copper remaining in the charge.

The silver has a tendency to precipitate on the lead linings to a small extent. This would not offer much difficulty, as the lead rapidly becomes coated and precipitation ceases. The solutions, on cooling and on the addition of water, drop some of the silver out as silver chloride. This forms in the bottom of the solution tank to more or less extent and has to be cleaned up periodically. All of the silver in solution precipitated in the scrap-iron launders with no trouble whatever.

## Summary

The accompanying Tables I, II, and III speak for themselves, so there is not much to explain about results.

The feed from the mill averages between 0.65 and 0.70 per cent. copper, about 5 per cent. moisture, and about 3 per cent. sulphur. Ninety tons per 24 hr. of this feed can be roasted in a 20-ft. MacDougall furnace of the type used here with a coal consumption of 2.75 per cent. of its dry weight. The calcine will contain about 0.6 per cent. sulphur. About 40 per cent. of the copper is water soluble.

*Screen Analysis on the Feed*

| Mesh | Per cent. | Cumulative, Per cent. |
|---|---|---|
| On 14 | 4.50 | 4.50 |
| On 40 | 50.31 | 54.81 |
| On 60 | 15.33 | 70.14 |
| On 90 | 11.45 | 81.59 |
| On 110 | 6.54 | 88.13 |
| On 160 | 5.52 | 93.65 |
| On 200 | 2.86 | 96.51 |
| Through 200 | 5.31 | 101.82 |

Under conditions which would apply in a large plant, it is expected that an 85 per cent. extraction can be made on the copper, and very probably 86 or 87 per cent. A tailing carrying between 2 and 2½ lb. of copper per ton should be made. In a plant as small as this 80-ton plant, and built for temporary use, a great many petty annoyances are encountered, as the running of the furnace, and therefore of the whole plant, depended on so many small pieces of machinery in the conveying and cooling equipment. Not 10 per cent. of the furnace shutdowns were caused by troubles with the furnace itself. Naturally, the furnace

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does much better roasting, and with a less coal consumption, when it is running continuously or nearly so.

No salt is necessary in the roast for the better extraction of the copper. It would increase the silver extraction from 75 to 90 per cent., but this extra 15 per cent. of the silver recovered would not pay for increased cost of salt, installation, and operation of an apparatus to catch the values in the gases. This recovery of fume values would probably not be much over 90 per cent. The chloridizing of the silver is done in the leaching tanks.

*Summary of Results from Laboratory Leaches During Oxide-Chloride Period*

The extraction percentages on both copper and silver assume complete recovery of volatilized values.

No silver assays were made on the feed, en average result of 0.55 oz. per ton being used in figuring extraction.

| | |
|---|---|
| Recovery of copper, per cent | 86.7 |
| Recovery of silver, per cent | 94.3 |
| Pounds copper per ton in feed | 12.48 |
| Pounds copper per ton in tailings | 1.64 |
| Pounds copper per ton recovered | 10.84 |
| Average feed per 24 hr, tons | 90.8 |
| Average salt consumption, per cent | 1.04 |
| Average coal consumption, per cent | 2.94 |

*Summary of Results from Tank Leaches*

Oxide-Chloride Period

The extraction percentages of silver and copper during the chloridizing period are figured by using the copper and silver values in the feed and assuming all volatilized values recovered.

| | |
|---|---|
| Pounds copper in feed per ton | 12.04 |
| Pounds copper in tails per ton | 2.54 |
| Pounds copper recovered per ton | 9.50 |
| Per cent. extraction on copper | 78.9 |
| Oz. silver per ton in feed | 0.55 |
| Oz. silver per ton in tails | 0.15 |
| Per cent. extraction on silver | 72.9 |

Oxide Period

| | |
|---|---|
| Pounds copper in calcine per ton | 13.58 |
| Pounds copper in tails per ton | 2.12 |
| Pounds copper recovered per ton | 11.46 |
| Per cent. extraction on copper—top ½ of tank | 86.2 |
| Per cent. extraction on copper—bottom ½ of tank | 82.8 |
| Per cent. extraction on copper—total tank | 84.4 |
| Per cent. extraction copper—tails washed | 86.0 |
| Oz. silver in calcine, per ton | 0.57 |
| Oz. silver in tails, per ton | 0.15 |
| Per cent. extraction on silver | 70.9 |

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## TABLE I.—*Oxide-Chloride Roast*

### Results from Laboratory Leaches

| Date | | Rate of Feed Dry Weight | Per cent. Moisture in Feed | Per cent. Coal | Per cent. Salt | Per cent. Copper in Feed | Per cent. Copper in Calcine | Per cent. Volatilised Copper | Average Temperature of Roast | Per cent. Copper as $Cu_2S$ in Tails | Per cent. Copper as Ferrite in Tails | Per cent. Total Copper in Tails | Oz. Silver per Ton in Tails | Extraction per cent. on Copper | Extraction per cent. on Silver |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oct. | 21 | 84.7 | 4.5 | 3.00 | 1.16 | 0.587 | 0.504 | 0.083 | 1,070 | 0.048 | 0.060 | 0.108 | 0.05 | 81.7 | 91.0 |
| | 22 | 88.1 | 7.0 | 3.29 | 0.70 | 0.580 | 0.539 | 0.041 | 1,050 | 0.044 | 0.061 | 0.105 | 0.03 | 88.0 | 94.6 |
| | 23 | 101.2 | 4.0 | 3.38 | 1.24 | 0.590 | 0.531 | 0.059 | 1,060 | 0.045 | 0.068 | 0.113 | 0.05 | 81.0 | 91.0 |
| | 24 | 103.1 | 6.5 | 2.75 | 1.16 | 0.555 | 0.478 | 0.077 | 1,000 | 0.039 | 0.044 | 0.083 | 0.03 | 85.0 | 94.6 |
| | 25 | 102.0 | 7.5 | 3.00 | 0.93 | 0.501 | 0.460 | 0.041 | 1,000 | 0.036 | 0.029 | 0.065 | 0.03 | 87.0 | 94.6 |
| | 26 | 104.7 | 5.0 | 2.81 | 0.86 | 0.478 | 0.448 | 0.030 | 930 | 0.040 | 0.022 | 0.062 | 0.02 | 87.1 | 96.5 |
| | 27 | 104.6 | 5.0 | 2.82 | 0.97 | 0.545 | 0.521 | 0.024 | 950 | 0.038 | 0.025 | 0.063 | 0.04 | 88.4 | 92.8 |
| | 28 | 103.5 | 6.0 | 2.95 | 0.94 | 0.549 | 0.519 | 0.030 | 1,000 | 0.043 | 0.036 | 0.079 | 0.03 | 85.9 | 94.6 |
| | 29 | 103.6 | 6.5 | 3.16 | 0.87 | 0.555 | 0.501 | 0.054 | 1,000 | 0.040 | 0.039 | 0.079 | 0.02 | 84.0 | 96.5 |
| | 30 | 104.6 | 5.0 | 2.68 | 0.86 | 0.555 | 0.466 | 0.089 | 1,000 | 0.044 | 0.046 | 0.090 | 0.03 | 83.8 | 94.6 |
| | 31 | 109.9 | 5.5 | 2.84 | 0.84 | 0.596 | 0.519 | 0.077 | 1,000 | 0.044 | 0.046 | 0.090 | 0.03 | 85.9 | 94.6 |
| Nov. | 1 | 108.3 | 5.5 | 2.90 | 0.86 | 0.596 | 0.549 | 0.047 | 980 | 0.044 | 0.044 | 0.088 | 0.03 | 85.4 | 94.6 |
| | 2 | 104.1 | 5.5 | 2.94 | 1.04 | 0.614 | 0.553 | 0.061 | 930 | 0.043 | 0.028 | 0.071 | 0.03 | 88.5 | 94.6 |
| | 3 | 104.7 | 5.0 | 2.92 | 1.03 | 0.550 | 0.516 | 0.034 | 900 | 0.042 | 0.024 | 0.066 | 0.03 | 82.8 | 94.6 |
| | 4 | 105.0 | 5.0 | 3.24 | 1.14 | 0.562 | 0.533 | 0.029 | 900 | 0.040 | 0.020 | 0.060 | 0.03 | 89.4 | 94.6 |
| | 5 | 104.7 | 5.0 | 2.96 | 1.06 | 0.550 | 0.528 | 0.022 | 900 | 0.050 | 0.029 | 0.079 | 0.05 | 85.6 | 91.0 |
| | 6 | 103.5 | 6.0 | 2.82 | 1.01 | 0.597 | 0.562 | 0.017 | 920 | 0.050 | 0.020 | 0.070 | 0.03 | 87.8 | 94.6 |
| | 10 | 100.3 | 8.5 | 3.03 | 1.16 | 0.699 | 0.643 | 0.056 | 930 | 0.046 | 0.026 | 0.072 | 0.03 | 89.8 | 94.6 |
| | 11 | 103.6 | 6.0 | 2.92 | 1.15 | 0.712 | 0.661 | 0.051 | 930 | 0.067 | 0.022 | 0.089 | 0.03 | 87.7 | 94.6 |
| | 12 | 91.9 | 10.5 | 3.21 | 1.26 | 0.612 | 0.556 | 0.056 | 960 | 0.038 | 0.038 | 0.076 | 0.02 | 87.7 | 96.5 |
| | 13 | 91.3 | 9.3 | 3.13 | 1.37 | 0.625 | 0.556 | 0.069 | 920 | 0.042 | 0.033 | 0.075 | 0.04 | 88.1 | 92.8 |
| | 14 | 91.5 | 6.0 | 2.72 | 1.25 | 0.587 | 0.562 | 0.025 | 930 | 0.043 | 0.032 | 0.075 | 0.04 | 87.2 | 92.8 |
| | 15 | 93.7 | 6.0 | 2.93 | 1.25 | 0.656 | 0.619 | 0.037 | 930 | 0.065 | 0.027 | 0.092 | 0.03 | 86.0 | 94.6 |
| | 16 | 96.0 | 4.0 | 3.03 | 1.10 | 0.675 | 0.662 | 0.013 | 1,000 | 0.076 | 0.029 | 0.105 | 0.05 | 84.5 | 91.0 |
| | 17 | 86.5 | 6.0 | 3.50 | 1.22 | 0.644 | 0.619 | 0.025 | 1,020 | 0.060 | 0.045 | 0.105 | 0.03 | 83.7 | 94.6 |
| | 18 | 83.4 | 4.0 | 2.74 | 1.25 | 0.669 | 0.637 | 0.032 | 950 | 0.045 | 0.037 | 0.082 | 0.03 | 87.8 | 94.6 |
| | 19 | 80.1 | 6.0 | 2.80 | 1.22 | 0.700 | 0.650 | 0.050 | 940 | 0.064 | 0.022 | 0.086 | 0.02 | 87.7 | 96.5 |
| | 20 | 80.8 | 5.0 | 2.79 | 1.19 | 0.710 | 0.704 | 0.006 | 930 | 0.078 | 0.024 | 0.102 | 0.04 | 85.7 | 92.8 |
| | 21 | 80.2 | 5.0 | 2.99 | 1.12 | 0.761 | 0.742 | 0.019 | 1,000 | 0.072 | 0.034 | 0.106 | 0.05 | 86.0 | 91.0 |
| | 22 | 78.9 | 5.5 | 2.97 | 1.18 | 0.692 | 0.616 | 0.076 | 960 | 0.054 | 0.033 | 0.087 | 0.04 | 87.6 | 92.8 |
| | 23 | 77.0 | 4.0 | 2.88 | 1.10 | 0.660 | 0.597 | 0.063 | 960 | 0.053 | 0.025 | 0.078 | 0.02 | 88.2 | 96.5 |
| | 24 | 76.2 | 5.0 | 3.08 | 1.19 | 0.692 | 0.660 | 0.032 | 960 | 0.086 | 0.015 | 0.101 | 0.03 | 85.6 | 94.6 |
| | 25 | 75.2 | 6.0 | 2.75 | 1.24 | 0.666 | 0.609 | 0.057 | 1,000 | 0.048 | 0.017 | 0.065 | 0.03 | 90.4 | 94.6 |
| | 26 | 75.2 | 6.0 | 3.04 | 1.10 | 0.622 | 0.560 | 0.062 | 1,000 | 0.042 | 0.023 | 0.065 | 0.02 | 89.6 | 96.5 |
| | 27 | 79.5 | 4.5 | 2.58 | 0.92 | 0.684 | 0.610 | 0.074 | 1,010 | 0.055 | 0.022 | 0.077 | 0.02 | 88.9 | 96.5 |
| | 28 | 78.7 | 6.0 | 2.91 | 0.97 | 0.912 | 0.874 | 0.038 | 1,020 | 0.076 | 0.031 | 0.107 | 0.03 | 88.4 | 94.6 |
| Dec. | 2 | 81.4 | 3.5 | 2.60 | 0.83 | 0.595 | 0.533 | 0.062 | 980 | 0.054 | 0.021 | 0.075 | 0.03 | 87.5 | 94.6 |
| | 3 | 82.1 | 3.0 | 2.62 | 0.88 | 0.651 | 0.620 | 0.031 | 1,020 | 0.048 | 0.018 | 0.066 | 0.04 | 89.8 | 92.8 |
| | 4 | 78.4 | 7.5 | 2.77 | 0.95 | 0.620 | 0.558 | 0.038 | 1,000 | 0.055 | 0.014 | 0.069 | 0.03 | 89.0 | 94.6 |
| | 5 | 82.6 | 3.5 | 2.62 | 0.77 | 0.614 | 0.571 | 0.057 | 1,000 | 0.038 | 0.027 | 0.065 | 0.04 | 89.3 | 92.8 |
| | 6 | 82.0 | 4.5 | 3.09 | 0.78 | 0.614 | 0.601 | 0.087 | 950 | 0.046 | 0.028 | 0.074 | 0.03 | 88.0 | 94.6 |
| | 7 | 82.2 | 4.0 | 3.02 | 0.84 | 0.614 | 0.539 | 0.025 | 950 | 0.047 | 0.029 | 0.076 | 0.03 | 87.8 | 94.6 |
| | 8 | 75.7 | 11.5 | 3.12 | 0.87 | 0.614 | 0.595 | 0.081 | 950 | 0.047 | 0.031 | 0.078 | 0.03 | 87.4 | 94.6 |
| | 9 | 73.6 | 14.0 | 2.75 | 0.90 | 0.645 | 0.558 | 0.013 | 950 | 0.049 | 0.048 | 0.097 | 0.02 | 85.0 | 96.5 |
| Average | | 90.8 | 5.9 | 2.94 | 1.04 | 0.624 | 0.578 | 0.046 | | 0.050 | 0.032 | 0.082 | 0.03 | 86.7 | 94.3 |

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TABLE II.—*Oxide Roast. Results From Laboratory Leaches*

| Date | | Rates of Feed Dry Weight | Per cent. Moisture in Feed | Per cent. Coal | Per cent. Copper in Feed | Per cent. Copper in Calcine | Average Temperature of Roast | Per cent. Copper as $Cu_2S$ in Tails | Per cent. Copper as Ferrite in Tails | Per cent. Total Copper in Tails | Oz. Silver per Ton in Calcine | Oz. Silver per Ton in Tails | Extraction per cent. on Copper | Extraction per cent. on Silver |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Dec. | 10 | 81.0 | 5.5 | 2.72 | 0.580 | 0.568 | 950 | 0.047 | 0.030 | 0.077 | 0.44 | 0.07 | 86.7 | 84.2 |
| | 11 | 81.8 | 4.5 | 3.11 | 0.600 | 0.605 | 950 | 0.044 | 0.019 | 0.063 | 0.45 | 0.07 | 89.4 | 84.5 |
| | 12 | 82.6 | 4.0 | 2.72 | 0.584 | 0.574 | 960 | 0.051 | 0.020 | 0.071 | 0.44 | 0.08 | 87.6 | 81.8 |
| | 13 | 82.7 | 4.0 | 2.80 | 0.642 | 0.660 | 950 | 0.055 | 0.017 | 0.072 | 0.48 | 0.08 | 89.2 | 83.3 |
| | 14 | 81.2 | 5.0 | 2.87 | 0.672 | 0.710 | 940 | 0.074 | 0.017 | 0.091 | 0.62 | 0.10 | 87.3 | 84.0 |
| | 15 | 77.2 | 5.5 | 3.54 | 0.735 | 0.722 | 950 | 0.078 | 0.015 | 0.093 | 0.59 | 0.10 | 87.0 | 83.2 |
| | 16 | 78.7 | 4.5 | 3.05 | 0.710 | 0.703 | 970 | 0.079 | 0.041 | 0.120 | 0.56 | 0.09 | 82.9 | 83.9 |
| | 17 | 78.6 | 5.5 | 2.69 | 0.641 | 0.641 | 940 | 0.050 | 0.032 | 0.082 | 0.52 | 0.09 | 88.1 | 82.7 |
| | 18 | 81.0 | 6.0 | 2.50 | 0.732 | 0.707 | 930 | 0.060 | 0.019 | 0.079 | 0.55 | 0.07 | 88.8 | 87.3 |
| | 19 | 78.7 | 7.0 | 2.73 | 0.700 | 0.713 | 930 | 0.073 | 0.018 | 0.091 | 0.53 | 0.08 | 87.0 | 84.9 |
| | 20 | 77.4 | 9.0 | 2.54 | 0.601 | 0.601 | 930 | 0.048 | 0.025 | 0.073 | 0.54 | 0.08 | 87.9 | 85.3 |
| | 21 | 79.0 | 4.0 | 2.31 | 0.583 | 0.583 | 930 | 0.051 | 0.017 | 0.068 | 0.48 | 0.08 | 88.2 | 83.4 |
| | 22 | 81.1 | 5.0 | 2.59 | 0.696 | 0.663 | 930 | 0.063 | 0.015 | 0.078 | 0.56 | 0.08 | 88.3 | 85.7 |
| | 23 | 80.9 | 4.5 | 2.82 | 0.626 | 0.632 | 930 | 0.050 | 0.033 | 0.083 | 0.54 | 0.09 | 86.9 | 83.3 |
| | 24 | 79.4 | 7.0 | 2.57 | 0.645 | 0.676 | 930 | 0.048 | 0.027 | 0.075 | 0.62 | 0.12 | 89.0 | 80.7 |
| | 26 | 68.7 | 5.5 | 3.74 | 0.626 | 0.632 | 940 | 0.062 | 0.012 | 0.074 | 0.59 | 0.09 | 88.4 | 84.8 |
| | 27 | 80.7 | 5.5 | 2.90 | 0.654 | 0.666 | 930 | 0.047 | 0.024 | 0.071 | 0.60 | 0.08 | 90.2 | 86.6 |
| | 28 | 80.9 | 5.0 | 2.87 | 0.616 | 0.635 | 930 | 0.063 | 0.013 | 0.076 | 0.51 | 0.08 | 88.1 | 84.4 |
| | 29 | 79.2 | 7.5 | 2.95 | 0.585 | 0.610 | 920 | 0.054 | 0.023 | 0.077 | 0.44 | 0.07 | 87.5 | 84.2 |
| | 30 | 80.4 | 6.5 | 2.54 | 0.660 | 0.672 | 920 | 0.054 | 0.020 | 0.074 | 0.47 | 0.08 | 89.1 | 83.0 |
| | 31 | 81.1 | 5.0 | 2.03 | 0.710 | 0.697 | 920 | 0.058 | 0.022 | 0.080 | 0.58 | 0.08 | 88.6 | 86.3 |
| Jan. | 1 | 81.1 | 5.0 | 2.69 | 0.753 | 0.741 | 930 | 0.079 | 0.017 | 0.096 | 0.63 | 0.09 | 87.04 | 85.7 |
| | 2 | 81.3 | 3.5 | 2.84 | 0.716 | 0.691 | 920 | 0.064 | 0.015 | 0.079 | 0.46 | 0.06 | 88.56 | 87.0 |
| | 3 | 83.5 | 4.0 | 2.49 | 0.691 | 0.685 | 910 | 0.061 | 0.023 | 0.084 | 0.41 | 0.05 | 87.74 | 87.8 |
| | 4 | 81.0 | 5.0 | 2.76 | 0.740 | 0.672 | 920 | 0.065 | 0.023 | 0.088 | 0.49 | 0.08 | 86.90 | 83.7 |
| | 5 | 80.1 | 5.0 | 2.88 | 0.722 | 0.729 | 920 | 0.078 | 0.022 | 0.100 | 0.56 | 0.09 | 86.28 | 83.9 |
| | 6 | 81.8 | 4.0 | 2.75 | 0.735 | 0.729 | 920 | 0.054 | 0.016 | 0.070 | 0.42 | 0.06 | 90.40 | 85.7 |
| | 7 | 81.1 | 5.0 | 2.59 | 0.645 | 0.676 | 920 | 0.061 | 0.022 | 0.083 | 0.46 | 0.08 | 85.08 | 82.6 |
| | 8 | 82.3 | 3.5 | 2.47 | 0.657 | 0.608 | 920 | 0.053 | 0.016 | 0.069 | 0.48 | 0.06 | 88.65 | 87.5 |
| | 9 | 82.1 | 4.0 | 2.50 | 0.707 | 0.701 | 920 | 0.065 | 0.019 | 0.084 | 0.56 | 0.10 | 88.02 | 82.2 |
| | 10 | 79.2 | 4.5 | 2.72 | 0.707 | 0.688 | 920 | 0.061 | 0.019 | 0.080 | 0.51 | 0.07 | 88.68 | 86.3 |
| | 11 | 80.8 | 4.5 | 2.62 | 0.732 | 0.756 | 920 | 0.095 | 0.016 | 0.111 | 0.56 | 0.07 | 85.32 | 87.5 |
| | 12 | 80.7 | 4.0 | 2.59 | 0.756 | 0.750 | 920 | 0.081 | 0.027 | 0.108 | 0.58 | 0.09 | 85.60 | 84.5 |
| | 13 | 78.0 | 3.5 | 2.94 | 0.750 | 0.775 | 930 | 0.085 | 0.012 | 0.097 | 0.55 | 0.10 | 87.48 | 81.8 |
| | 14 | 83.0 | 3.0 | 2.66 | 0.856 | 0.818 | 950 | 0.109 | 0.022 | 0.131 | 0.70 | 0.12 | 83.99 | 82.9 |
| | 15 | 75.8 | 4.5 | 3.29 | 0.781 | 0.832 | 950 | 0.107 | 0.035 | 0.142 | 0.70 | 0.15 | 82.93 | 78.5 |
| | 16 | 76.5 | 3.0 | 2.69 | 0.732 | 0.725 | 940 | 0.074 | 0.024 | 0.098 | 0.62 | 0.13 | 86.48 | 79.0 |
| | 19 | 84.1 | 5.5 | 2.61 | 0.756 | 0.738 | 950 | 0.084 | 0.023 | 0.107 | 0.75 | 0.12 | 85.50 | 84.0 |
| | 20 | 73.2 | 4.0 | 2.92 | 0.787 | 0.795 | 960 | 0.083 | 0.023 | 0.106 | 0.67 | 0.12 | 86.32 | 82.1 |
| | 21 | 72.2 | 4.5 | 3.00 | 0.769 | 0.756 | 960 | 0.107 | 0.011 | 0.118 | 0.71 | 0.13 | 84.39 | 81.7 |
| | 22 | 72.3 | 4.5 | 2.76 | 0.756 | 0.775 | 980 | 0.089 | 0.024 | 0.113 | 0.52 | 0.10 | 85.42 | 80.8 |

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TABLE II.—*Oxide Roast. Results From Laboratory Leaches. (Continued)*

| Date | | Rate of Feed Dry Weight | Per cent. Moisture in Feed | Per cent. Coal | Per cent. Copper in Feed | Per cent. Copper in Calcine | Average Temperature of Roast | Per cent. Copper as $CuS$ in Tails | Per cent. Copper as Ferrite in Tails | Per cent. Total Copper in Tails | Oz. Silver per Ton in Calcine | Oz. Silver per Ton in Tails | Extraction per cent. on Copper | Extraction per cent. on Silver |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Jan. | 23 | 71.7 | 4.5 | 2.53 | 0.763 | 0.781 | 1,000 | 0.063 | 0.026 | 0.099 | 0.65 | 0.17 | 87.32 | 73.8 |
| | 24 | 72.3 | 4.0 | 2.94 | 0.738 | 0.732 | 1,000 | 0.063 | 0.029 | 0.092 | 0.58 | 0.14 | 87.43 | 75.8 |
| | 25 | 71.5 | 5.0 | 2.83 | 0.692 | 0.717 | 1,000 | 0.048 | 0.033 | 0.081 | 0.70 | 0.18 | 88.70 | 74.3 |
| | 26 | 76.3 | 5.5 | 2.62 | 0.625 | 0.630 | 960 | 0.040 | 0.030 | 0.070 | 0.55 | 0.14 | 88.88 | 74.5 |
| | 27 | 83.8 | 5.0 | 2.70 | 0.655 | 0.643 | 950 | 0.043 | 0.021 | 0.064 | 0.48 | 0.08 | 90.05 | 83.3 |
| | 28 | 84.8 | 6.5 | 2.74 | 0.631 | 0.655 | 940 | 0.041 | 0.024 | 0.065 | 0.54 | 0.10 | 90.08 | 81.5 |
| | 29 | 83.4 | 5.5 | 3.28 | 0.576 | 0.643 | 950 | 0.046 | 0.018 | 0.064 | 0.58 | 0.12 | 90.05 | 79.3 |
| | 30 | 84.0 | 6.0 | 2.71 | 0.613 | 0.619 | 930 | 0.042 | 0.019 | 0.061 | 0.48 | 0.09 | 90.15 | 81.2 |
| | 31 | 80.6 | 6.0 | 2.82 | 0.668 | 0.625 | 930 | 0.053 | 0.012 | 0.065 | 0.57 | 0.10 | 89.60 | 82.5 |
| Feb. | 1 | 82.7 | 4.5 | 2.62 | 0.686 | 0.704 | 950 | 0.046 | 0.025 | 0.071 | 0.59 | 0.11 | 89.91 | 81.3 |
| | 2 | 80.0 | 6.0 | 2.33 | 0.695 | 0.775 | 970 | 0.057 | 0.025 | 0.082 | 0.66 | 0.12 | 89.42 | 81.8 |
| | 3 | 81.5 | 5.0 | 2.27 | 0.702 | 0.695 | 960 | 0.046 | 0.027 | 0.073 | 0.66 | 0.15 | 89.50 | 77.3 |
| | 4 | 78.5 | 7.0 | 2.50 | 0.691 | 0.683 | 970 | 0.046 | 0.027 | 0.073 | 0.78 | 0.19 | 89.31 | 75.7 |
| | 5 | 74.8 | 4.5 | 2.35 | 0.641 | 0.634 | 980 | 0.035 | 0.032 | 0.067 | 0.49 | 0.15 | 89.43 | 68.1 |
| | 9 | 81.5 | 4.5 | 2.56 | 0.653 | 0.671 | 980 | 0.038 | 0.021 | 0.059 | 0.48 | 0.08 | 91.21 | 83.3 |
| | 10 | 74.4 | 12.0 | 3.15 | 0.641 | 0.659 | 960 | 0.041 | 0.025 | 0.066 | 0.47 | 0.08 | 89.98 | 83.0 |
| | 11 | 81.0 | 5.0 | 2.44 | 0.666 | 0.691 | 920 | 0.046 | 0.020 | 0.066 | 0.67 | 0.12 | 90.45 | 82.1 |
| | 12 | 80.8 | 5.0 | 2.49 | 0.703 | 0.716 | 930 | 0.080 | 0.009 | 0.089 | 0.83 | 0.15 | 87.57 | 81.9 |
| | 13 | 81.7 | 4.0 | 2.59 | 0.635 | 0.660 | 930 | 0.053 | 0.019 | 0.072 | 0.68 | 0.11 | 89.09 | 83.8 |
| | 14 | 81.5 | 4.0 | 2.89 | 0.594 | 0.600 | 950 | 0.038 | 0.019 | 0.057 | 0.56 | 0.11 | 90.50 | 80.3 |
| | 15 | 81.2 | 4.5 | 2.92 | 0.643 | 0.643 | 960 | 0.041 | 0.023 | 0.064 | 0.67 | 0.14 | 90.05 | 79.1 |
| | 16 | 81.3 | 4.5 | 2.82 | 0.637 | 0.655 | 960 | 0.035 | 0.034 | 0.069 | 0.61 | 0.11 | 89.47 | 82.0 |
| | 17 | 79.9 | 5.5 | 2.78 | 0.625 | 0.637 | 930 | 0.043 | 0.030 | 0.073 | 0.54 | 0.09 | 88.54 | 79.6 |
| | 18 | 81.3 | 5.5 | 2.61 | 0.613 | 0.619 | 930 | 0.038 | 0.024 | 0.062 | 0.61 | 0.11 | 89.98 | 82.0 |
| | 19 | 80.0 | 6.0 | 2.73 | 0.588 | 0.606 | 920 | 0.040 | 0.023 | 0.063 | 0.53 | 0.09 | 89.60 | 79.2 |
| | 22 | 80.6 | 4.5 | 2.78 | 0.588 | 0.600 | 920 | 0.033 | 0.031 | 0.064 | 0.56 | 0.09 | 89.33 | 84.0 |
| | 23 | 80.6 | 4.5 | 2.87 | 0.662 | 0.643 | 920 | 0.044 | 0.024 | 0.073 | 0.58 | 0.09 | 88.65 | 84.4 |
| | 24 | 80.6 | 6.0 | 2.61 | 0.649 | 0.686 | 930 | 0.042 | 0.027 | 0.069 | 0.47 | 0.08 | 89.94 | 83.1 |
| | 25 | 89.6 | 5.5 | 2.77 | 0.637 | 0.662 | 930 | 0.042 | 0.023 | 0.065 | 0.38 | 0.04 | 90.18 | 89.5 |
| Ave. | | 79.8 | 5.1 | 2.75 | 0.675 | 0.680 | | 0.058 | 0.023 | 0.081 | 0.57 | 0.10 | 88.09 | 82.5 |

| | |
|---|---|
| Recovery of copper, per cent. | 88.09 |
| Recovery of silver, per cent. | 82.5 |
| Pounds copper per ton in calcine. | 13.6 |
| Pounds copper per ton in tailings. | 1.62 |
| Pounds copper per ton recovered. | 11.98 |
| Average feed per 24 hr. (tons dry wt.). | 79.8 |
| Average coal consumption, per cent. | 2.75 |

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TABLE III.—*Results From Tank Leaches.*—(*Oxide-Chloride Roast*)

| Leach Number | Tons Calcine | Per cent. Copper in Feed | Per cent. Copper in Calcine | Top Half | | Bottom Half | | Total Tank | | Washed Clean | | Pounds Copper Recovered per Ton | Oz. Silver per Ton in Feed | Oz. Silver per Ton in Tails | Per cent. Extraction on Silver | Temperature Solution C° | Per cent. $H_2SO_4$ in No. 1 Solution | Per cent. NaCl in No. 1 Solution | Per cent. $H_2SO_4$ in No. 2 Solution | Per cent. NaCl in No. 2 Solution |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Per cent. Copper in Tails | Per cent. Extraction Copper | Per cent. Copper in Tails | Per cent. Extraction Copper | Per cent. Copper in Tails | Per cent. Extraction Copper | Per cent. Copper in Tails | Per cent. Extraction Copper | | | | | | | | | |
| 1 | 309 | 0.580 | 0.540 | ..... | ..... | ..... | ..... | 0.202 | 65.2 | ..... | ..... | 7.56 | 0.55 | 0.23 | 5.82 | 20 | 3.03 | 4.70 | 5.10 | 4.90 |
| 2 | 391 | 0.569 | 0.500 | ..... | ..... | ..... | ..... | 0.151 | 73.5 | ..... | ..... | 8.36 | 0.55 | 0.17 | 69.1 | 20 | 3.72 | 5.16 | 6.15 | 7.17 |
| 3 | 390 | 0.560 | 0.506 | ..... | ..... | ..... | ..... | 0.140 | 75.0 | ..... | ..... | 8.40 | 0.55 | 0.15 | 72.7 | 20 | 4.70 | 7.05 | 6.33 | 8.50 |
| 4 | 380 | 0.592 | 0.526 | ..... | ..... | ..... | ..... | 0.113 | 80.9 | ..... | ..... | 9.58 | 0.55 | 0.20 | 63.6 | 65 | 5.20 | 8.26 | 7.40 | 9.09 |
| 5 | 385 | 0.541 | 0.491 | ..... | ..... | ..... | ..... | 0.097 | 82.1 | ..... | ..... | 8.88 | 0.55 | 0.08 | 85.5 | 65 | 5.30 | 5.00 | 8.90 | 7.80 |
| 6 | 360 | 0.536 | 0.491 | ..... | ..... | ..... | ..... | 0.111 | 79.3 | ..... | ..... | 8.50 | 0.55 | 0.12 | 78.2 | 65 | 5.95 | 6.93 | 8.60 | 7.96 |
| 7 | 457 | 0.575 | 0.540 | ..... | ..... | ..... | ..... | 0.135 | 76.5 | ..... | ..... | 8.80 | 0.55 | 0.19 | 65.5 | 65 | 4.30 | 7.50 | 7.50 | 8.40 |
| 8 | 370 | 0.680 | 0.654 | ..... | ..... | ..... | ..... | 0.136 | 80.0 | ..... | ..... | 10.88 | 0.55 | 0.17 | 69.1 | 65 | 4.30 | 7.90 | 9.10 | 7.00 |
| 9 | 368 | 0.683 | 0.631 | 0.077 | 88.7 | 0.085 | 87.6 | 0.081 | 88.1 | ..... | ..... | 12.04 | 0.55 | 0.08 | 85.5 | 65 | 2.90 | 7.00 | 6.00 | 6.90 |
| 10 | 224 | 0.701 | 0.662 | 0.092 | 86.8 | 0.106 | 84.9 | 0.099 | 85.9 | ..... | ..... | 12.04 | 0.55 | 0.10 | 81.8 | 65 | 3.40 | 8.20 | 5.00 | 7.50 |
| Ave | ... | 0.602 | 0.554 | 0.085 | 87.8 | 0.096 | 86.8 | 0.127 | 78.9 | ..... | ..... | 9.50 | 0.55 | 0.15 | 72.9 | .... | ..... | ..... | ..... | ..... |
| | | | | | | | | *Oxide Roast* | | | | | | | | | | | | |
| **11** | **370** | **0.616** | **0.629** | **0.097** | **84.8** | **0.072** | **88.5** | **0.085** | **86.5** | **0.081** | **87.1** | **10.88** | **0.53** | **0.15** | **71.7** | **65** | **2.13** | **8.91** | **4.68** | **7.53** |
| **12** | **410** | **0.686** | **0.681** | **0.104** | **84.7** | **0.124** | **81.8** | **0.114** | **83.3** | **0.101** | **85.2** | **11.34** | **0.53** | **0.26** | **50.9** | **65** | **2.90** | **8.16** | **7.13** | **7.59** |
| **13** | **430** | **0.636** | **0.648** | **0.091** | **85.9** | **0.157** | **75.8** | **0.124** | **80.9** | **0.105** | **83.8** | **10.48** | **0.56** | **0.15** | **73.2** | **65** | **2.18** | **11.06** | **6.50** | **9.62** |
| **14** | **400** | **0.713** | **0.710** | **0.129** | **81.8** | **0.146** | **79.4** | **0.138** | **80.6** | **0.122** | **82.8** | **11.44** | **0.54** | **0.42** | **22.2** | **65** | **2.58** | **10.30** | **5.33** | **7.17** |
| **15** | **375** | **0.718** | **0.713** | **0.082** | **88.5** | **0.087** | **87.8** | **0.085** | **88.1** | **0.077** | **89.2** | **12.56** | **0.51** | **0.08** | **84.3** | **65** | **5.02** | **11.61** | **6.60** | **9.18** |
| **16** | **390** | **0.761** | **0.759** | **0.096** | **87.4** | **0.105** | **86.2** | **0.100** | **86.8** | **0.095** | **87.5** | **13.18** | **0.66** | **0.18** | **72.7** | **65** | **4.18** | **7.11** | **9.20** | **9.84** |
| **17** | **410** | **0.656** | **0.660** | **0.071** | **89.2** | **0.100** | **84.9** | **0.086** | **87.1** | **0.075** | **88.6** | **11.48** | **0.56** | **0.17** | **69.6** | **65** | **2.82** | **8.44** | **18.00** | **8.72** |
| **18** | **385** | **0.677** | **0.686** | **0.097** | **85.9** | **0.139** | **79.7** | **0.118** | **82.8** | **0.101** | **85.3** | **11.36** | **0.60** | **0.21** | **65.0** | **65** | **2.85** | **7.72** | **6.50** | **8.80** |
| **19** | **395** | **0.654** | **0.670** | **0.086** | **87.2** | **0.122** | **81.8** | **0.104** | **84.5** | **0.097** | **85.5** | **11.32** | **0.60** | **0.15** | **75.0** | **65** | **3.78** | **7.29** | **7.05** | **8.59** |
| **20** | **395** | **0.616** | **0.629** | **0.086** | **86.3** | **0.115** | **81.7** | **0.101** | **84.0** | **0.096** | **84.7** | **10.54** | **0.58** | **0.15** | **74.1** | **65** | **4.65** | **10.15** | **7.05** | **9.50** |
| **Ave.** | ... | **0.673** | **0.679** | **0.094** | **86.2** | **0.117** | **82.8** | **0.106** | **84.4** | **0.095** | **86.0** | **11.46** | **0.57** | **0.17** | **70.9** | .... | ..... | ..... | ..... | ..... |

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Curves for the Sensible-Heat Capacity of Furnace Gases

BY C. R. KUZELL AND G. H. WIGTON, GREAT FALLS, MONT.

(Salt Lake Meeting, August, 1914)

## INTRODUCTION

KNOWLEDGE of the thermal capacity of gases is of great importance in making metallurgical calculations.

The metallurgist is frequently called upon to investigate and determine furnace efficiencies in which the heat carried into or out of the furnace by gases is a large item in the heat balance. Not only do such problems present themselves in the determination of furnace efficiency, but also in the study of the application of heat is accessory apparatus, such as stoves, regenerators, waste-heat boilers, driers, etc. The thermal effect of the use of excess air in the combustion of fuel, the theoretical temperatures of combustion, the quantity of heat in hot blast at various temperatures, the effect of hot blast on furnace temperatures, are a few more examples of frequently occurring calorific problems involving gases. So many are the applications of the data on the heat capacity of gases that the subject merits careful study.

The heat in a gas may be due to its heat of combustion, if it is a combustible gas, and to its temperature. The latter is called its sensible heat and is the heat absorbed or evolved by a gas as its temperature is raised or lowered. The heat of combustion of gases is well established and is commonly known. The values can be found in almost any modern treatise on metallurgy. The sensible heat of gases has not been so well established because it is in most cases a function of the temperature, and the values of specific heats of gases over a wide range of temperatures have only recently been determined. It is the purpose of this paper to deal with the sensible heat of gases.

The calculation of sensible-heat capacities[1] from specific-heat equations is a comparatively long and tedious operation. This is especially

[1] Dr. Fulton uses the term "thermal capacity" (*Principles of Metallurgy*, p. 410). However, in the case of a combustible gas, the term "thermal capacity" might be erroneously supposed by some to include heat of combustion.

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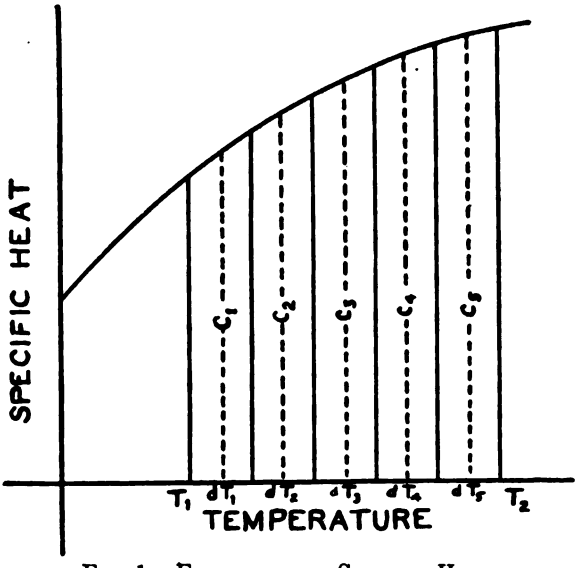


FIG. 1.—EQUATION FOR SPECIFIC HEAT.

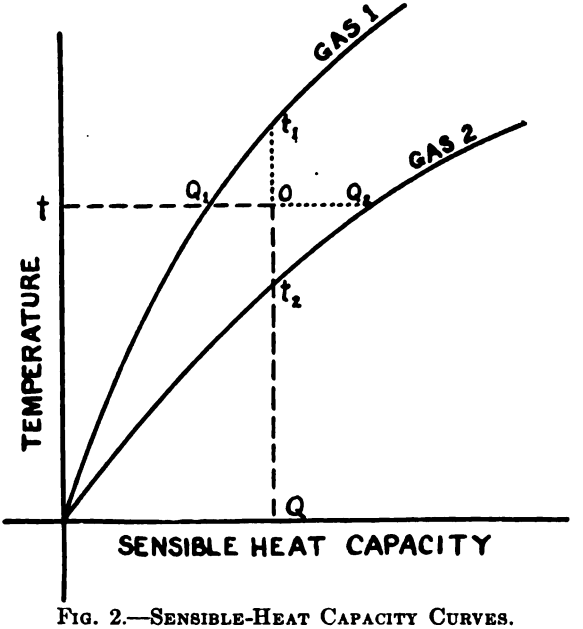


FIG. 2.—SENSIBLE-HEAT CAPACITY CURVES.

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true in the case of polyatomic gases the specific-heat equations of which are quadratics or are higher in degree.

To simplify this calculation the authors have selected what have appeared to be the most reliable and recent data and have constructed a series of curves by means of which the sensible heat in a gas at any temperature or between any temperatures up to 2,000° C., or 3,600° F., may be easily determined. The use of these curves has not only effected a great saving in time, but has encouraged the making of many calculations which would not have been attempted if the more laborious method was used.

## Experimental Data on the Specific Heat of Gases

Dr. Joseph W. Richards has, in his *Metallurgical Calculations*, Part I, given formulæ for the mean specific heats of all the common gases in terms of both the imperial and metric systems of measurement. These formulæ are derived from the results of experimental determination of the specific heats at various temperatures. The data were obtained from the researches of Regnault, Mallard, and LeChatelier. Since the time of publication of that book, however, other investigators have been at work on the subject. Among these are: Pier, Holborn, Austin, Henning, Njerrum, Furstenau, and Swan. A summary of the results of their researches has been very ably made by Messrs. Lewis and Randall, who presented their conclusions in a paper entitled A Summary of the Specific Heats of Gases.[2] They have selected formulæ for molal heats[3] of gases to agree as closely as possible with all the experimental work, and where a difference in data existed the values obtained by the most reliable method were chosen. Since their formulæ agree with the most probable values of all the experimental data, they are especially valuable, and have been used in constructing the curves for heat capacity that are the subject of this paper.

## Specific-Heat Equations and Their Derivation

As noted above, Lewis and Randall give formulæ for the molal heats of the gases. It has been necessary to convert these formulæ into more convenient form for use with either volumetric or gravimetric units in either the Imperial or Metric system.

To facilitate the use of formulæ, the following symbols have been adopted:

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[2] See *Journal of the American Chemical Society*, vol. xxxiv, No. 9.

[3] The molal heat is the product of the specific heat and the molecular weight.

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$C_p$ = molal heat = heat required to raise the temperature of a molecular weight of the gas 1° under constant pressure.[4]
$c_p$ = specific heat under constant pressure.
$c_m(t_1 \text{ to } t)$ = mean specific heat between the temperatures $t_1$ and t.
T = absolute temperature, degrees centigrade.
t = temperature, degrees centigrade.
*t* = temperature, degrees Fahrenheit.
$Q(t_1 \text{ to } t)$ = quantity of sensible heat (heat capacity) between the temperatures $t_1$ and t.

The following formulæ were selected by Lewis and Randall:

| | |
|---|---|
| Nitrogen, oxygen, carbon monoxide, | $C_p = 6.50 - 0.0010T$ |
| Hydrogen, | $C_p = 6.50 - 0.0009T$ |
| Water vapor, hydrogen sulphide, | $C_p = 8.81 - 0.0019T + 0.00000222T^2$ |
| Carbon dioxide, sulphur dioxide, | $C_p = 7.0 + 0.0071T - 0.00000186T^2$ |

Table I gives the formulæ for mean specific heats that we have calculated from the above molal heats for 1 kg., 1 cu. m., 1 lb., and 1 cu. ft. of each gas.

The values for methane are calculated from a formula given by Fulton (*Principles of Metallurgy*, p. 408) on the authority of J. V. Ehrenwerth (*Metallurgie*, vol. vi, p. 306).

For an explanation of the method of converting specific-heat formulæ to mean specific-heat formulæ see Appendix A.

## Heat-Capacity Curves; Construction and General Application

The amount of sensible heat that a gas will absorb between any two temperatures $t_1$ and t is equal to the product of the mean specific heat between those two temperatures times the difference in temperature. Therefore, if one temperature is zero,

$$Q(0 \text{ to } t) = c_m\, t$$

By substituting in this equation the various values of $c_m$ given in the preceding tabulation and then substituting various values of t, any desired number of values of Q may be obtained. From these values a curve can easily be plotted. Such a curve will give for any temperature up to 2,000° C., or 3,600° F., the sensible-heat capacity of the gas.

On Plate I we have drawn curves for each gas giving the sensible-heat capacity in British thermal units per pound and also per cubic foot (measured under standard conditions). On Plate II are corresponding

[4] Gases in metallurgical furnaces usually expand under constant pressure. If the gas is confined to a constant volume, external work is done to maintain the constancy of volume. In such a case the amount of the outer work may be calculated (see Richards's *Metallurgical Calculations*, pt. i, p. 60) by subtracting from the specific heat at constant pressure 2 calories per degree for a molecular weight, or 0.09 calorie per degree for 1 cu. m., or 0.09 ÷ weight of 1 cu. m. for a kilogram of gas.

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curves giving kilogram-calories per kilogram and per cubic meter (also measured under standard conditions).

It should be noted that the values given apply to the substances only in the gaseous state.

Dissociation is not great within the range of temperatures for which the curves are drawn. (See Fulton, *Principles of Metallurgy*, p. 414.)

The general application of the curves is simple. The temperatures are laid off along the axis of ordinates and the sensible-heat capacities are laid off along the axis of abscissæ. If the temperature of a gas is known and it is desired to find the sensible-heat capacity between zero and this temperature, one reading gives it directly. If two temperatures are given and the sensible-heat capacity between these two temperatures is desired, two readings must be taken and subtracted one from the other.

In addition to Plate I and Plate II we have drawn two more series of curves in Plates III and IV, which give the actual specific heats at various temperatures, the former for units of weight and the latter for units of volume. These are for use in finding the theoretical temperature of combustion, as explained in Appendix B and as illustrated in Example 4.

## Application of Heat-Capacity Curves in Metallurgical Calculations

The applications of the curves to metallurgical problems are many. On account of lack of space only four typical examples, just enough to illustrate the use of the curves, will be worked out. Special attention is directed to Example 4, which illustrates a method of finding the theoretical temperature of combustion much more easily than by any other method we know of.

### *Example* 1

A reverberatory furnace discharges 10,000 cu. ft. of waste gases per minute (measured under standard conditions) at 2,000° F. The volumetric analysis of the gas is as follows:

| $H_2O$ | $CO_2 + SO_2$ | $O_2$ | $N_2$ |
|---|---|---|---|
| 8 | 15 | 2 | 74 |

Find the sensible heat in the waste gases above 0°F.

Referring to Plate I, we obtain the following values for each of the constituent gases at 2,000° F.

Sensible-heat capacity, B.t.u. per cubic foot.

| $H_2O$ | $CO_2 + SO_2$ | $O_2 + N_2$ |
|---|---|---|
| 49.6 | 63.6 | 40.8 |

Combining these in the proper proportion we get:

$49.6 \times 0.08 + 63.6 \times 0.15 + 40.8 \times 0.77 = 44.92$ B.t.u. per cubic foot,

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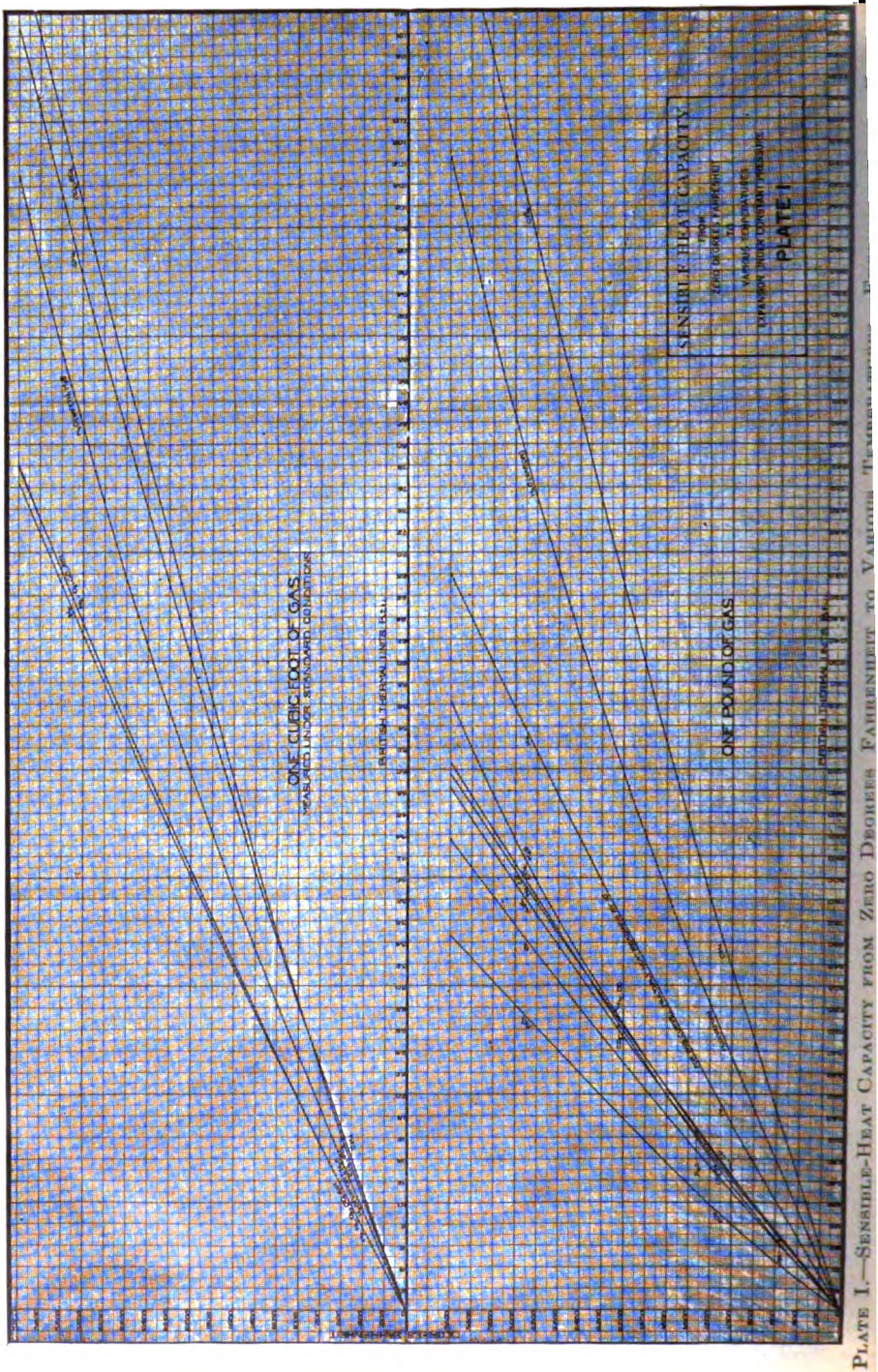


PLATE I.—SENSIBLE-HEAT CAPACITY FROM ZERO DEGREES FAHRENHEIT TO VARIOUS TEMPERATURES

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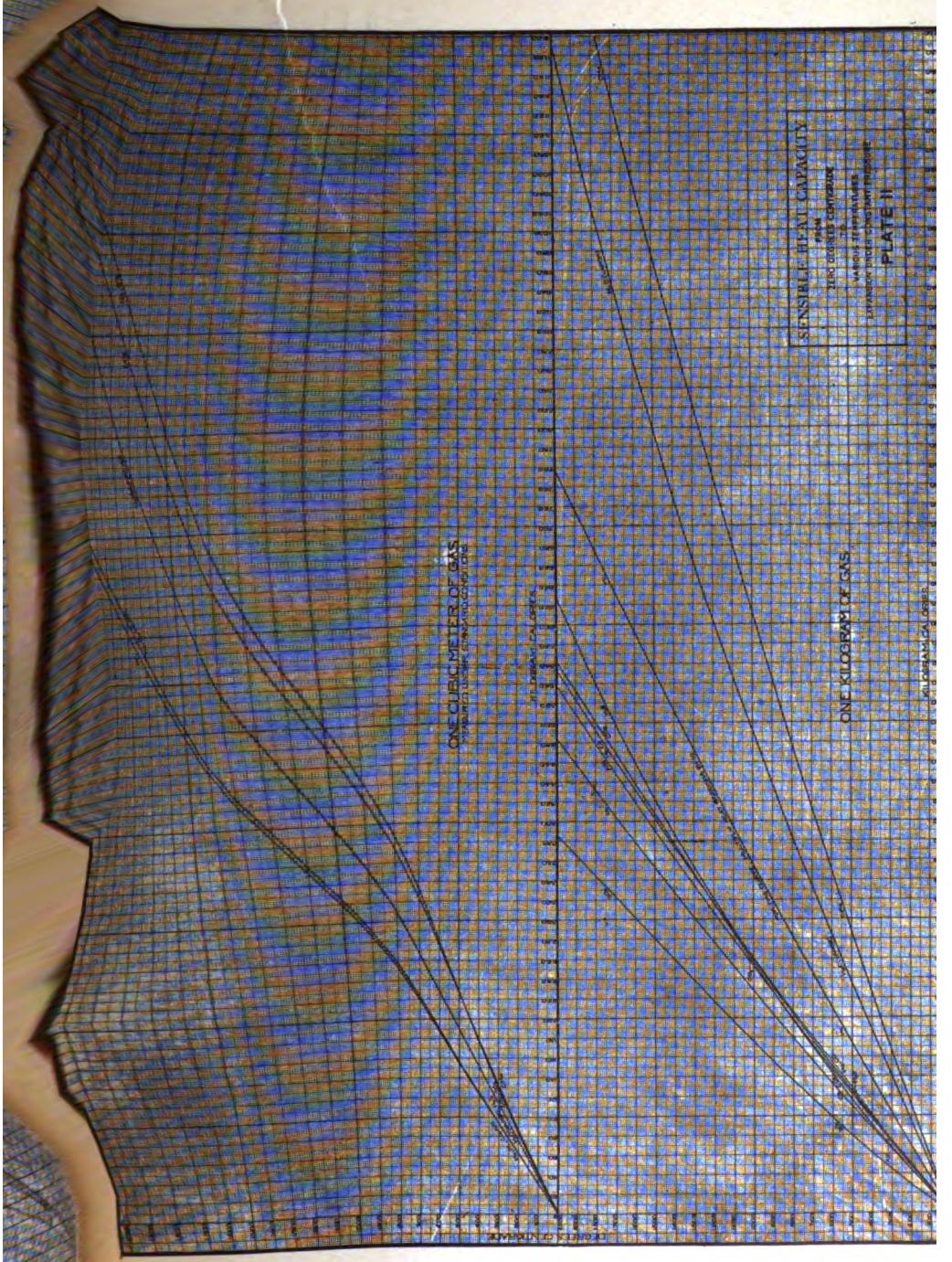




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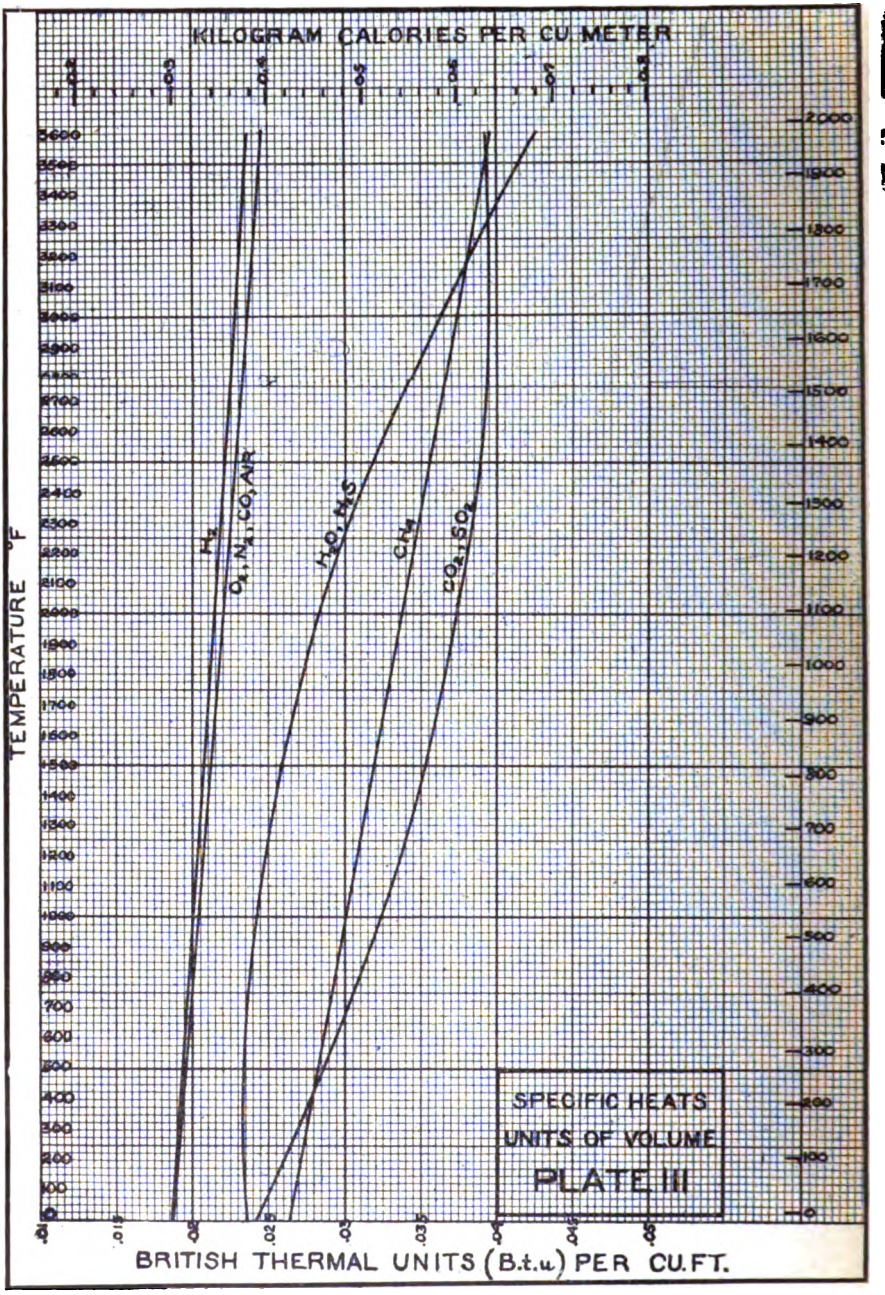


PLATE III.—SPECIFIC HEATS. UNITS OF VOLUME.

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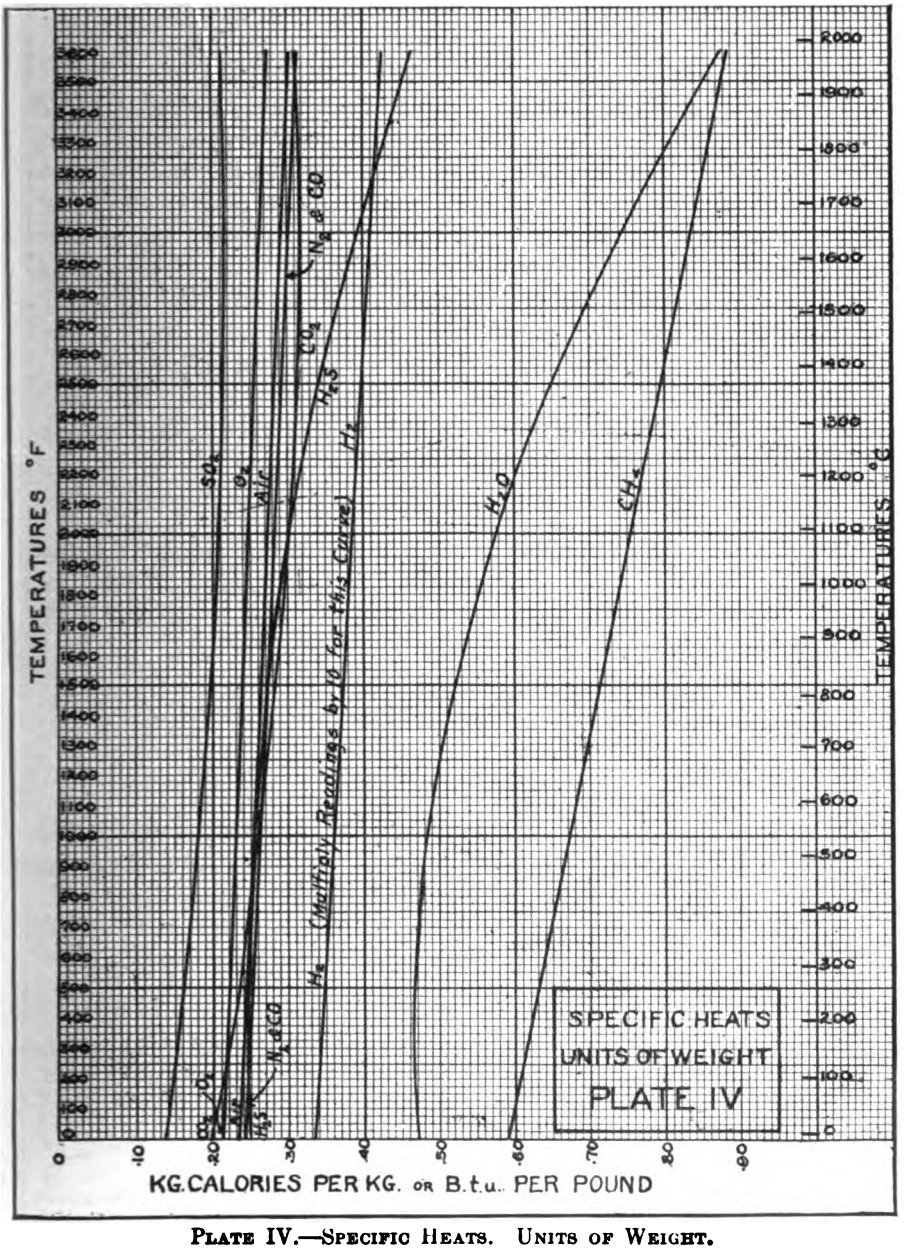


PLATE IV.—SPECIFIC HEATS. UNITS OF WEIGHT.

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which is the sensible heat in 1 cu. ft. of the waste gases. Therefore the total sensible heat passing out of the furnace in the waste gases each minute is 439,500 B.t.u.

*Example* 2

Suppose the gases of Example 1 enter a brick regenerator at 2,000° F. and leave it at 600° F., and that the regenerator subsequently heats 8,000 cu. ft. of air from 70° F. to 1,200° F. for every 10,000 cu. ft. of waste gases (both air and gases being measured under standard conditions). What is the calorific efficiency of the regenerator? From Example 1 we know that the sensible heat in the gases at 2,000° F. is 439,500 B.t.u. Proceeding in the same manner (using Plate I), we find that at 600° F. the sensible heat per cubic foot is:

$$14.1 \times 0.08 + 16.2 \times 0.15 + 11.6 \times 0.77 = 12.49 \text{ B.t.u.}$$

The sensible heat in 10,000 cu. ft. at 600° is therefore 124,900 B.t.u. The heat input of the regenerator is therefore

$$439{,}500 - 124{,}900 = 314{,}600 \text{ B.t.u.}$$

The heat in 1 cu. ft. of air at 1,200° F. is 23.7 B.t.u. and at 70° F. it is 1.3 B.t.u. Therefore, the net heat absorbed by 1 cu. ft. of air is 22.4 B.t.u. and the heat output of the regenerator is 8,000 × 22.4, or 179,200 B.t.u.
∴ Efficiency of regenerator =

$$\frac{\text{Heat output}}{\text{Heat input}} \times 100 = \frac{179{,}200 \times 100}{314{,}600} = 57.0 \text{ per cent.}$$

*Example* 3

At an iron blast-furnace plant a stove receives 35,000 cu. ft. of blast-furnace gas per minute. The heating value of this gas is 98 B.t.u. per cubic foot and its volumetric composition is:

| $CO_2$ | CO | $H_2$ | $CH_4$ | $H_2O$ | $O_2$ | $N_2$ |
|---|---|---|---|---|---|---|
| 14.0 | 24.0 | 3.5 | 0.5 | 3.0 | 0.0 | 55.0 |

The gas is at a temperature of 600° F.

The stove discharges 61,000 cu. ft. of gases at 550° F. and of the following volumetric composition.

| $CO_2$ | $H_2O$ | $O_2$ | $N_2$ |
|---|---|---|---|
| 22.1 | 4.3 | 2.0 | 71.6 |

The temperature of the air that is used for burning the gas in the stove is 50° F.

The stove (while on air) heats 100,000 cu. ft. of air from 50° F. to 1,100° F. for every 35,000 cu. ft. of the original blast-furnace gas.

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All volumes are measured under standard conditions of temperature and pressure.

What is the efficiency of the stove?

The per cent. efficiency is $\frac{\text{Heat output}}{\text{Heat input (net)}} \times 100$.

| | B.t.u. |
|---|---|
| ***Heat input:*** | |
| Heat of combustion of 35,000 cu. ft. at 98 B.t.u. = | 3,330,000 |
| (From Plate I) Sensible heat in 35,000 cu. ft. gas at 600° F. 35,000 [16.2 × 0.14 + 11.6 × (0.24 + 0.035 + 0.55) + 14.1 × 0.03 + 17.0 × 0.005] = | 632,250 |
| (From Plate I) Since the air required for combustion is 30,700 cu. ft. (by calculation), the sensible heat in it at 50° F. is 30,700 × 0.9 = | 27,630 |
| Gross heat input of stove, | 3,989,880 |
| Deduct sensible heat in 61,000 cu. ft. gases at 550° F. 61,000 [14.7 × 0.221 + 12.9 × 0.043 + 10.5 × 0.736] = | 703,330 |
| Net heat input of stove, | 3,286,550 |
| ***Heat output:*** | |
| (From Plate I) Sensible heat in 100,000 cu. ft. air at 1,100° F. 100,000 × 21.6 = | 2,160,000 |
| (From Plate I) Sensible heat in 100,000 cu. ft. air at 50° F. 100,000 × 0.9 = | 90,000 |
| Heat output of stove = | 2,070,000 |

***Efficiency of stove:***

$$\text{Per cent. efficiency} = \frac{2{,}070{,}000}{3{,}286{,}550} = 63 \text{ per cent.}$$

*Example* 4

Find the theoretical temperature of combustion, or "pyrometric effect," of the blast-furnace gas in *Example* 3.

The total sensible-heat capacity of the 61,000 cu. ft. of the products of combustion is 3,989,880 B.t.u., or 65.4 B.t.u. per cubic foot

The composition of the products of combustion is:

| $CO_2$ | $H_2O$ | $O_2$ | $N_2$ |
|---|---|---|---|
| $22.x$ | 4.3 | 2.0 | 71.6 |

The theoretical temperature of combustion t is found by the formula:

$$t = \frac{t_1p_1c_1 + t_2p_2c_2 + t_3p_3c_3 + \text{etc.}}{p_1c_1 + p_2c_2 + p_3c_3 + \text{etc.}}$$

in which, $t_1$, $t_2$, $t_3$, etc., are the theoretical temperatures of combustion of each of the constituent gases, if we consider for a moment that the gases contain only that constituent and no other. In other words $t_1$, $t_2$, $t_3$, etc., are the temperatures corresponding to the points of intersection of the curves on Plate I with an ordinate drawn through 65.4 B.t.u.

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$p_1$, $p_2$, $p_3$, etc., are equal to the volumetric percentage of each constituent gas.

$c_1$, $c_2$, $c_3$, etc., are equal to the mean specific heat of 1 cu. ft. of each constituent gas between t and $t_1$, t and $t_2$, etc., and are obtained by reading the specific heats from the curves of Plate III just above or below the temperatures $t_1$, $t_2$, $t_3$, etc., so that the readings will be very closely the mean specific heats between t and $t_1$, t and $t_2$, etc.

For a full explanation of the method of obtaining the formula for the theoretical temperature see Appendix B.

In this particular example we shall let the subscripts 1, 2, 3, and 4 represent respectively, $CO_2$, $H_2O$, $O_2$, and $N_2$.

From Plate I, we obtain by erecting an ordinate through 65.4 B.t.u.

$$t_1 = 2{,}045^\circ \text{ F.}, \quad t_2 = 2{,}520^\circ \text{ F.}, \quad t_3 \text{ and } t_4 = 3{,}075^\circ \text{ F.}$$

From Plate III, we obtain:

$$c_1 = 0.0385, \quad c_2 = 0.0335, \quad c_3 \text{ and } c_4 = 0.0235$$

From the analysis:

$$p_1 = 22.1, \quad p_2 = 4.3, \quad p_3 = 2.0, \quad p_4 = 71.6$$

Substituting: $t =$

$$\frac{(2{,}045 \times 22.1 \times 0.0385) + (2{,}520 \times 4.3 \times 0.0335) + (2.0 + 71.6)\ (3{,}075 \times 0.0235)}{(22.1 \times 0.0335) + (4.3 \times 0.0335) + (73.6 \times 0.0235)}$$

$$= \frac{7{,}423}{2.725} = 2{,}724^\circ\text{F.}$$

## APPENDIX A

### *Method of Transforming the Equation for Specific Heat into the Equation for Mean Specific Heat*

Let the curve in Fig. 1 represent the general equation for specific heat,

$$c_p = A + BT + CT^2 + DT^3 + \text{etc.}$$

In which A, B, C, D, etc., are constants and T is the temperature.

Let $T_1$ and $T_2$ be any two temperatures on the curve.

The mean specific heat $c_m$ between $T_1$ and $T_2$ will therefore be,

$$c_m(T_1 \text{ to } T_2) = \frac{c_1 dT_1 + c_2 dT_2 + c_3 dT_3 + \ldots + c_n dT_n}{dT_1 + dT_2 + dT_3 + \ldots + dT_n}$$

As dT decreases without limit,

$$c_m(T_1 \text{ to } T_2) = \frac{\int_{T_1}^{T_2} c\,dT}{\int_{T_1}^{T_2} dT} = \frac{\int_{T_1}^{T_2} c\,dT}{T_2 - T_1}$$

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Substituting the general equation for c,

$$c_m(T_1 \text{ to } T_2) = \frac{\int_{T_1}^{T_2} (A + BT + CT^2 + DT^3 + \text{etc.})dT}{T_2 - T_1}$$

$$= \frac{A(T_2 - T_1) + \frac{B}{2}(T_2^2 - T_1^2) + \frac{C}{3}T_2^3 - T_1^3) + \text{etc.}}{T_2 - T_1}$$

$$= A + \frac{B}{2}(T_2 + T_1) + \frac{C}{3}(T_2^2 + T_1T_2 + T_1^2) + \text{etc.}$$

When $T_1 = 0$,

$$c_m(0 \text{ to } T) = A + \frac{B}{2}T + \frac{C}{3}T^2 + \text{etc.}$$

## Appendix B

### *Derivation of Formula for Finding the Theoretical Temperature of Combustion*

Consider the products of combustion of a combustible gas as made up of several constituent gases, which we shall call Gas 1, Gas 2, Gas 3, etc.

In Fig. 2, the curves represent the sensible-heat capacity curves for the series of constituent gases, Gas 1, Gas 2, Gas 3, etc.

$Q$ = sensible-heat capacity of the combined gases of combustion from 0° to t°.

$Q_1$, $Q_2$, $Q_3$, etc. = ditto for Gases 1, 2, 3, etc., from 0° to $t_1$, $t_2$, $t_3$, etc.

$t$ = theoretical temperature of combustion of the gas $t_1$, $t_2$, $t_3$, etc. = theoretical temperature of combustion if the total product of combustion were only one gas, Gas 1, Gas 2, etc.

$p_1$, $p_2$, $p_3$, etc. = portions of Gas 1, Gas 2, Gas 3, etc., in the products of combustion.

It is evident that $Q = p_1Q_1 + p_2Q_2 + p_3Q_3 + \text{etc.}$

From the figure $Q_1 = Q\,\overline{OQ_1}$, and $Q_2 = Q + \overline{OQ_2}$, etc.

$$\therefore Q = p_1(Q + \overline{OQ_1}) + p_2(Q + \overline{OQ_2}) + \text{etc.}$$

whence,

$$Q = Q(p_1 + p_2 + \text{etc.}) + (p_1\overline{OQ_1} + p_2\overline{OQ_2} + \text{etc.})$$

Since

$$p_1 + p_2 + p_3 + \text{etc.} = 1$$

$$p_1\overline{OQ_1} + p_2\overline{OQ_2} + \text{etc.} = 0 \qquad (1)$$

It is evident that

$$t = t_1 + \overline{t_10} = t_2 + \overline{t_20} = \text{etc.} \qquad (2)$$

Let $m_1$, $m_2$, $m_3$ = average slopes of curves connecting the points $Q_1$ and $t_1$, $Q_2$ and $t_2$, etc.

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Then

$$OQ_1 = \frac{\bar{t}_1\bar{0}}{m_1}, \quad \bar{O}\bar{Q}_2 = \frac{\bar{t}_2\bar{0}}{m_2}, \text{ etc.}$$

Substituting these values in equation (1), we get

$$\frac{p_1}{m_1}\bar{t}_1\bar{0} + \frac{p_2}{m_2}\bar{t}_2\bar{0} + \text{etc.} = 0 \qquad (3)$$

From equations (2) and (3) we get the following simultaneous equations:

$$\frac{p_1}{m_1}\bar{t}_1\bar{0} + \frac{p_2}{m_2}\bar{t}_2\bar{0} + \text{etc.} = 0$$

$$t_1 - t_2 = \bar{t}_2\bar{0} - \bar{t}_1\bar{0}$$

$$t_1 - t_3 = \bar{t}_3\bar{0} - \bar{t}_1\bar{0}$$

Solving for $\bar{t}_1\bar{0}$ and substituting in $t = \bar{t}_1\bar{0} + t_1$

$$t = \frac{\frac{p_1t_1}{m_1} + \frac{p_2t_2}{m_2} + \text{etc.}}{\frac{p_1}{m_1} + \frac{p_2}{m_2} + \text{etc.}}$$

Since the reciprocal of the slope is the specific heat and the reciprocal of the average slope is the mean specific heat, we have:

$$\frac{1}{m} = c$$

$$t = \frac{t_1p_1c_1 + t_2p_2c_2 + t_3p_3c_3 + \text{etc.}}{p_1c_1 + p_2c_2 + p_3c_3 + \text{etc.}}$$

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TABLE I.—*Mean Specific Heats of Gases Under Constant Pressure up to 2,000° C., or 3,600° F.*
($t$ = °C., $t$ = °F.)

| | $N_2$ | $O_2$ |
|---|---|---|
| Cm(0 to t) for 1 kg. in kg-cal | $0.2417 + 0.0000178t$ | $0.2117 + 0.0000156t$ |
| Cm(0 to t) for 1 cu. m. in kg-cal | $0.3025 + 0.0000223t$ | $0.3025 + 0.0000223t$ |
| Cm(0 to t) for 1 lb. in B.t.u. | $0.2411 + 0.0000099t$ | $0.2111 + 0.0000087t$ |
| Cm(0 to t) for 1 cu. ft. in B.t.u. | $0.0188 + 0.0000008t$ | $0.0188 + 0.0000008t$ |
| | $H_2O$ | $H_2S$ |
| Cm(0 to t) for 1 kg. in kg-cal | $0.4694 - 0.0000191t + 0.00000000411t^2$ | $0.2481 - 0.0000101t + 0.00000000217t^2$ |
| Cm(0 to t) for 1 cu. m. in kg-cal | $0.3777 - 0.0000154t + 0.00000000330t^2$ | $0.3777 - 0.0000154t + 0.00000000330t^2$ |
| Cm(0 to t) for 1 lb. in B.t.u. | $0.4701 - 0.0000118t + 0.00000000127t^2$ | $0.2485 - 0.0000062t + 0.00000000067t^2$ |
| Cm(0 to t) for 1 cu. ft. in B.t.u. | $0.0236 - 0.0000006t + 0.00000000006t^2$ | $0.0236 - 0.0000006t + 0.00000000006t^2$ |

| | Air | CO | $H_2$ |
|---|---|---|---|
| Cm(0 to t) for 1 kg. in kg-cal | $0.2348 + 0.0000173t$ | $0.2419 + 0.0000179t$ | $3.3461 + 0.0002232t$ |
| Cm(0 to t) for 1 cu. m. in kg-cal | $0.3025 + 0.0000223t$ | $0.3025 + 0.0000223t$ | $0.2995 + 0.0000200t$ |
| Cm(0 to t) for 1 lb. in B.t.u. | $0.2342 + 0.0000096t$ | $0.2413 + 0.0000099t$ | $3.3381 + 0.0001240t$ |
| Cm(0 to t) for 1 cu. ft. in B.t.u. | $0.0188 + 0.0000008t$ | $0.0188 + 0.0000008t$ | $0.0187 + 0.0000007t$ |
| | $CO_2$ | $SO_2$ | $CH_4$ |
| Cm(0 to t) for 1 kg. in kg-cal | $0.2000 + 0.0000691t - 0.00000001411t^2$ | $0.1373 + 0.0000475t - 0.00000000997t^2$ | $0.593 + 0.0000741t$ |
| Cm(0 to t) for 1 cu. m. in kg-cal | $0.3930 + 0.0001359t - 0.00000002777t^2$ | $0.3930 + 0.0001359t - 0.00000002777t^2$ | $0.425 + 0.0000530t$ |
| Cm(0 to t) for 1 lb. in B.t.u. | $0.1975 + 0.0000388t - 0.00000000437t^2$ | $0.1356 + 0.0000267t - 0.00000000307t^2$ | $0.5904 + 0.0000411t$ |
| Cm(0 to t) for 1 cu. ft. in B.t.u. | $0.0242 + 0.0000048t - 0.00000000057t^2$ | $0.0242 + 0.0000048t - 0.00000000057t^2$ | $0.0264 + 0.00000184t$ |

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[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Salt Lake meeting, August, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Slime-Concentrating Plant at Anaconda

BY FREDERICK LAIST AND ALBERT F. WIGGIN, ANACONDA, MONT.

(Salt Lake Meeting, August, 1914)

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## I. INTRODUCTION

The new slime-concentrating plant at the Washoe Reduction Works, Anaconda, was put into operation during March, 1914. This plant, which has a capacity of 26,000,000 gal. of slime pulp carrying 2,500 tons of solid matter per day, consists of Dorr thickeners and Anaconda multiple-deck slime concentrators. This plant is the culmination of a great deal of experimental work extending over a considerable period of time, which work will be described briefly in the following pages.

## II. THE SOURCES AND AMOUNT OF SLIME

Up to the beginning of this year, the ore from the Anaconda company's mines in Butte, Mont., has been concentrated both at the Great Falls

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plant and at the Anaconda plant, the former plant treating about 3,000 and the latter about 9,500 tons per day. This year it was decided to do all of the concentrating at the Anaconda plant, which decision of course confines our slime problem to this plant alone. At present larger rolls and additional tables and trommels are being installed in the Anaconda mill to handle the additional tonnage. When milling at the rate of 12,500 tons per day there will be produced 2,500 tons, 20 per cent. of the ore fed to the mill, of material which cannot be concentrated in the mill on account of its extreme fineness. This material is known locally as "slime" and 95 per cent. of it will pass a 200-mesh (0.08 mm.) screen. The flow sheet of the Anaconda mill showing the sources of this slime was given in a paper presented at the Butte meeting, August, 1913, by A. E. Wiggin, entitled The Great Falls System of Concentration installed in Section No. 1 of the Washoe Concentrator at Anaconda.[1]

The 2,500 tons of solid matter is contained in about 26,000,000 gal. of pulp. Heretofore this material has been sent to six large slime ponds 300 by 600 ft. by about 15 ft. deep, which were operated in cycles, three being used in parallel when possible. The settlement was excavated and piled at the end of the ponds by Lidgerwood bucket excavators. This material was then sent to the briquetting plant at the blast-furnace department, where it was used as a binder in briquettes consisting of fine concentrate and fine first-class ore screenings. The smelter could handle not over 25 per cent. of the production of slime and the smelting of this material was expensive on account of its high insoluble content.

## III. The Composition of the Slime

Physically the slime may be said to consist of granular and colloidal material. Elaborate settling tests have shown that the colloidal matter amounts to about 35 per cent. of the total material and carries about 21 per cent. of the total copper.

Chemically the slime has the following average analysis:

| Cu, | $SiO_2$, | $Al_2O_3$, | Sulphur, | CaO, | FeO, | Oz. Per Ton | |
|---|---|---|---|---|---|---|---|
| Per cent. | Per cent. | Per cent. | Per cent. | Per cent. | Per cent. | Ag | Au |
| 2.10 | 61.0 | 19.0 | 4.4 | 0.6 | 4.1 | 1.8 | 0.005 |

## IV. The Experimental Development of the Anaconda Slime Plant

### 1. *The Great Falls Slime Plant and Experimental Work*

During 1904, a slime-concentrating plant was erected at Great Falls of sufficient size to treat about one-sixth of the total slime produced in the mill, or about 100 tons of slime per day. This plant consisted of 32

[1] *Bulletin* No. 80, August, 1913, pp. 1818, 1819.

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8-ft. Callow cones for the thickening of the slime pulp and 16 17-ft. round-table decks for the concentration of the thickened pulp. The round tables were second-hand wooden frame and deck tables which had been displaced in the sand mill by Wilfley tables. The slope of the deck surfaces varied considerably, largely due to the warping of the wooden decks and frames. Later some of the wooden decks were covered with linoleum, rubber, cement, and canvas and these various surfaces tested. It was found that canvas gave the highest recovery but this was at the expense of the grade of the concentrate, while linoleum, rubber, and wood gave a cleaner concentrate, but a lower recovery. Cement seemed to fall between the two groups, although the work of this form of deck is largely dependent upon the character of the finish given the cement. (The "cement" was in reality a concrete containing about one part cement to two parts sand.) It was concluded that the cement surface was the best adapted to the work, particularly because of its durability, and that the framework should be steel, as cement laid over a wooden deck and framing was not at all satisfactory. It was decided then to build a four-deck concrete-and-steel round table, the decks being supported on a central shaft and spaced about 4 ft. apart. Before building this table, however, it was necessary to determine the proper slope of deck surface, as the decks of the tables in the slime plant at that time had slopes varying from 1 to 1½ in. per foot. An extensive series of experiments was carried out during 1909, covering a period of about six months, to determine not only the proper slope of deck surface but also the proper density of pulp, the proper speed of revolution of the table, and the relation between the recovery and the grade of concentrate. A sector of a round table the slope of which could be varied was used in making these experiments. Slopes ranging from ½ in. per foot to 2 in. per foot (4 per cent. to 16 per cent.) were tried, the intervals being ⅛ in. per foot. This work proved conclusively that the best slope of deck surface for the concentration of this slime is 1¼ in. per foot (10 per cent.). At slopes less than 1⅛ in. per foot it was found that a considerable portion of the gangue was deposited beneath the mineral and was not displaced by the mineral, the mineral seemed to form a veritable blanket over the gangue material. This condition, of course, resulted in a very low-grade concentrate as it was impossible to wash out the gangue without a heavy loss of the mineral. At slopes greater than 1⅜ in. per foot the loss of the valuable minerals became excessive, due to the more rapid pulp flow.

It was shown that a density of pulp of from 8 to 12 per cent. solids was best adapted to round-table concentration. Too great a density makes the pulp too viscous and does not permit of a proper separation of the mineral from the gangue material. Too low a density cuts down the capacity of the table unnecessarily. The water contained in the feed pulp plays an important part in the dressing of the concentrate by remov-

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ing a great deal of the gangue material. It had been the practice in the Great Falls slime plant to feed the deck over one-half of its surface, dressing with clean water over about seven-sixteenths of the surface and removing the concentrate by strong jets of water on the remaining one-sixteenth of the surface. Our experimental work showed that the deck could be fed to advantage over about three-fourths of its surface leaving about one-fourth for dressing and removing the concentrate. We also determined that the best speed of revolution was from 4 to 6 min. per revolution. This work also showed that it is possible to make a net recovery of 54 per cent. of the copper and silver in a concentrate assaying 6.5 per cent. copper, or better, from a slime feed containing about 2.25 per cent. copper. At the time this work was carried out the slime produced from the Great Falls mill was quite similar in character to that produced in the Anaconda mill to-day. It did settle somewhat more readily, the capacity of settling tanks on the Great Falls slime being about 40 per cent. greater, when settling 95 per cent. of the solid matter, than the same settling tanks when treating the present Anaconda slime.

The Great Falls slime plant as a unit never made a very satisfactory net recovery, due to the fact that the settling capacity was inadequate and to the poor condition of the round tables, which were for the most part second-hand wooden tables. As there was not slime-plant capacity for the treatment of all of the slime produced in the mill it was the policy of the management to overfeed the slime plant and in that way produce more copper, although at a sacrifice in recovery.

A great deal of credit should be given to Dr. R. H. Richards for valuable suggestions made during this work at the Great Falls plant.

### 2. *The Experimental Work Done at Anaconda*

During 1912 the Great Falls system of concentration was installed in the No. 1 section of the Anaconda mill. This system had been developed at Great Falls for the treatment of Butte ore. The round-table concentration of the slime was a part of this system, and in connection with the installation of the system in section No. 1 of the Anaconda mill a slime section consisting of 24 8-ft. Callow settling tanks and two four-deck steel-and-concrete round tables was installed to treat the slime from the remodeled section. The slime section was built to confirm our Great Falls test data and had a capacity of about 425,000 gal. of slime pulp per day, or approximately 20 per cent. of the slime produced in section No. 1 of the mill. We anticipated from our Great Falls data that the 24 Callow tanks would have a capacity of 600,000 gal. when settling at an efficiency of 95 per cent., but, owing to the fact that the Anaconda slime has a lower rate of settling than the Great Falls slime, we found our capacity to be only 425,000 gal. of pulp, or 12.3 gal. per tank

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per minute, which is equivalent to 0.286 gal. per square foot of settling area per minute. This amount of pulp carried about 35 tons of solid matter, so that our eight round-table decks were somewhat underloaded.

This experimental slime plant was operated for a period of 14 months during which time a great deal of test work was carried out.

(*a*) *The Slime-Thickening Devices.*—The slime pulp leaves the mill at a density of 2 per cent. solids, and as the mechanical concentration requires a density of about 10 per cent. the first step in the treatment of the slime is to remove the excess water.

Considerable experimental work has been done by C. D. Demond, head of the Washoe Reduction Works testing department, on the use of baffles in tanks for the settlement of slime. We also carried on a series of experiments using the Kuchs-Laist centrifugal colloid separator to eliminate the colloidal material from the slime, treating only the granular portion on the concentrating machines. The work of this machine was very satisfactory and the net recovery from the slime was slightly greater than that made when concentrating the entire slime with the colloid included. It would, however, have taken considerable time to perfect a full-sized machine and for this reason the development of the machine was dropped. A great deal of experimental work was done also upon the Garred filter as a slime-thickening device but it was found that a filter was not practical for the thickening of such a low-density slime pulp (2 per cent. solids) to a relatively low-density pulp of about 10 per cent. solids. The details of the above experiments are given in a paper by Ralph Hayden presented at the Butte meeting of this society (August, 1913).

During the early part of 1913, a Dorr continuous thickener, 28 ft. in diameter by 10 ft. deep, was installed at the experimental slime plant. This thickener had a capacity of 195,000 gal. per day when operating at about 95 per cent. efficiency, or 135 gal. per minute, which is equivalent to 0.236 gal. per square foot of settling area per minute. Contrary to our expectations the 28-ft. Dorr thickener showed a slightly less capacity per square foot of settling surface than the 8-ft. Callow cones. The comparative results showed, however, that the settling area is a far more important factor than the length of the overflow weir, as the 24 Callow cones had a total weir length of 560 ft. while the Dorr thickener weir was but 85 ft. long. However, the effective weir length of the Callow tanks was probably not over 280 ft. on account of the very delicate adjustment of the overflow bands that was required when the head on the weir was so extremely small. Realizing the importance of a large settling area we next conceived the idea of a slime-thickening device composed of a number of superimposed shallow trays. Our idea was to feed these trays at one end with the slime pulp, allowing the clear water to overflow at the other end and the thickened pulp to collect in the tray.

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Periodically the trays would be mechanically tilted at a steep angle and the thickened pulp would be discharged. The trays would then resume a horizontal position and the cycle would repeat itself. We constructed a shallow tray for testing purposes but did not make the tests because this arrangement of trays had suggested to us the possibility of superimposing the Dorr thickener tanks. The Dorr thickeners had the advantage over the tray thickener of continuous operation and we decided, therefore, to drop the tray thickener for the time being. We found that the use of 28-ft. diameter tanks 3 ft. deep permitted of the superimposing of the tanks without getting into too heavy steel work for supports. We therefore made a test to determine the capacity of the 28-ft. diameter Dorr thickener when operated at a depth of 3 ft. This we found to be 163,000 gal. per day, 0.200 gal. per square foot of settling area per minute. This is 85 per cent. of the capacity of the 10 ft. deep tank. At the suggestion of H. W. Spicer of the Dorr Cyanide Machinery Co., we also tested the 28-ft. tank at a depth of 2 ft., and found that at this depth it had about 85 per cent. of the capacity of the 3 ft. deep tank. We finally decided to adopt the Dorr continuous thickener, using tanks 28 ft. in diameter by 3 ft. deep, as being the most economical and efficient form of slime-thickening device. The power required to operate the rakes which scrape the thickened slime to the central discharge is less than $\frac{1}{4}$ h.p. per tank for the 28 by 3 ft. unit. The arms carrying the rakes are set horizontal, or parallel to the bottom, in the 3 ft. deep tank.

(*b*) *Slime Concentrators.*—During the first eight months of operation of the experimental plant the four-deck round table was thoroughly tested. These tests included the production of a low-grade concentrate on the round table with subsequent treatment on one of the standard reciprocating-deck slime concentrators. The latter machine made a high-grade concentrate and a rich middling which was returned to the round-table system. It was found that this flow sheet did not give as good commercial results as the production of a clean concentrate directly from the round table requiring no secondary treatment. Among other arrangements we tried treating the round-table middling on a reciprocating table making a concentrate, middling (returned to the round table), and tailing. However, neither this nor the treatment of the round-table middling on a separate round-table deck gave as good results as simply mixing all of the round-table middling with the original table feed for treatment.

A great deal of experimental work was done on the Peck centrifugal slime concentrator, although this work was not carried on at the experimental slime plant.

Several of the standard slime concentrators, which are among the latest types on the market to-day, were tested quite thoroughly. The results of these tests, compared with the work of the round table under operating conditions, showed that the round table is the most efficient

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and economical slime concentrator. It was therefore decided to adopt the round table in the new slime-concentrating plant.

As noted previously, the round tables in the experimental slime plant were four-deck machines with the central shaft support and the decks spaced about 4 ft. apart. To make the machine more compact, thereby cutting down the first cost of the plant, we decided to build a 20-deck round table, the decks to be spaced 1 ft. between centers. The last six months of operation of the slime plant was devoted to the development and testing of a 20-deck machine. (See Figs. 3 and 4.)

The entire table was made of steel and concrete. The central shaft support was eliminated, the framework carrying the 20 decks being supported at the periphery by wheels which ran on a circular track. The individual decks consisted of sheet steel, supported by uprights at the periphery but self-supporting at the center, upon which was laid a layer of concrete (1 part cement to 2 parts of sand). This concrete formed the concentrating surface and was given a finish similar to that of a medium-weight canvas. Suitable steel launders were arranged around the outer edge of the decks to receive the products of concentration, two decks discharging into one launder. At the center and extending through each deck was an opening 4 ft. in diameter in which was placed a stationary steel tower. This tower carried the water and feed pulp pipes and a ladderway making each deck easily accessible. The feed pulp and wash water were supplied from a circular launder at the head of each deck. The entire structure carrying the decks was revolved at the rate of 15 times an hour, the power being supplied from a 5-h.p. motor placed beneath the table and engaging through a train of gears a circular rack placed near the periphery of the framework beneath the bottom deck. The actual power required to drive the table under working conditions was 3 h.p. The decks sloped 1¼ in. per foot and were 17 ft. in diameter.

The table was operated for about six months and proved to be entirely successful. It was therefore decided to adopt this type of concentrator in the new slime plant. Only a few minor changes were made in the design of the 20-deck table, such as increasing the diameter of the decks to 19 ft. and substituting manganese-steel wheels and rail in place of chilled cast iron. The table occupies a floor area equivalent to a circle about 21 ft. in diameter and the 20-deck table has a height over all, including the automatic feed pulp distributor, of about 35 ft. The 20-deck machine will handle 140 tons of Anaconda slime (dry) per day including the treatment of the middling. The net concentrating area per ton of total feed (including the returned middling) is 37 sq. ft.

The results shown in the following tabulation are typical of the round-table work and are obtained from an eight-day test made during April, 1913. As we did not have sufficient settling capacity at this time to supply feed to the 20 decks only eight decks were used.



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*Test on 20-Deck Round Table. Middling Returned to Table for Treatment*

| Product | Rate per 24 Hr. | | Assay | | Per Cent. of Total | |
|---|---|---|---|---|---|---|
| | Gallons Pulp | Pounds Solids | Cu | Insol. | Solids | Copper |
| Feed | 87,290 | 93,730 | 1.95 | ..... | 104.6 | 102.8 |
| Concentrate | 69,000 | 13,050 | 7.02 | 58.8 | 14.6 | 51.6 |
| Tailing | 134,580 | 76,540 | 1.13 | ..... | 85.4 | 48.4 |
| Total Produced | 203,580 | 89,590 | 1.98 | ..... | 100.0 | 100.0 |

Per cent. of copper in original slime recovered..... 51.6
Tons treated per deck, excluding middling..... 5.6

The round table requires about 3 gal. of dressing water per deck per minute and about 6 gal. of water to remove the concentrate per deck per minute. Fully 80 per cent. of the latter water is recovered in the dewatering of the concentrate and can be re-used.

It is our practice to feed the round-table deck over about 70 per cent. of its surface, supply dressing water over about 18 per cent. of the surface, and to use the remaining 12 per cent. of the surface for the removal of the concentrate. The amount of deck surface required for the dressing water is directly proportional to the grade of concentrate desired, the higher the grade the greater the dressing surface required. Approximately one-half of the material removed by the dressing water enters the middling and is returned to the table for treatment with the original feed. The middling amounts to from 8 to 10 per cent. of the original feed tonnage and carries about 5 to 6 per cent. of the total copper.

(*c*) *Dewatering of the Slime Concentrate.*—The slime-plant concentrate leaves the tables at a density of about 3 per cent. solids. It is of course necessary to dewater this material for treatment in the smelter. We decided to try a Dorr thickener followed by an Oliver filter as a dewatering system. In our experimental work we used a Dorr thickener 28 ft. in diameter by 10 ft. deep, and a 6-ft. continuous Oliver filter. The pulp was thickened to a density of approximately 50 per cent. solids in the Dorr thickener and then fed to the Oliver filter. Our tests showed that the Dorr thickener would treat 250,000 gal. of concentrate pulp containing about 3 per cent. solids and thicken it to a density of from 50 to 60 per cent. solids at practically 100 per cent. efficiency. The Oliver filter had a capacity of about 125 tons of solids contained in a pulp having a density of 55 per cent., when producing a cake carrying not over 15 per cent. moisture. The filter was operated under the "wet" system; that is, one Roots rotary pump handled both air and water in producing the vacuum. The average vacuum maintained was about 16 in.

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(*d*) *The Use of Chemicals to Accelerate the Settlement of Slime.*—Ferrous Sulphate: Two 8-ft. Callow cones were used in a test to determine the efficiency of ferrous sulphate as an accelerator in slime settlement. The two cones were operated under the same conditions of feed, overflow, and spigot discharge, the latter product being maintained at a density of about 10 per cent. solids. To the feed to one cone was added ferrous sulphate in the proportion of 1 lb., 2 lb., and 5 lb. per ton of solid slime. It was found that the addition of 1 lb. per ton gave an increase in capacity of 30 per cent., 2 lb. an increase of 40 per cent., and 5 lb. an increase of 50 per cent.

Salt: Some laboratory experiments were made to determine the effect of salt upon the rate of settlement of slime. Using ¼, ½ and 1 lb. of salt per ton of dry solids caused the slime to settle in only from 3 to 4 per cent. less time than the untreated slime. The salt had no effect upon the settlement of the round-table concentrate.

Glue and Ferrous Sulphate: Tests to determine the efficiency of glue and ferrous sulphate for the settlement of slime were made in a 28 by 10 ft. Dorr thickener. The addition of a mixture of ¼ lb. of each chemical per ton of dry slime increased the capacity of the Dorr thickener from 195,000 to 311,000 gal. of pulp per day, or an increase of 60 per cent. The use of glue and soluble sulphates to increase the rate of settlement of slime is covered by U. S. patent 1065878, held by A. G. French of British Columbia. Some laboratory experiments were made on the use of a mixture of glue and ferrous sulphate to accelerate the settlement of round-table concentrate. It was found that the addition of a mixture of the two chemicals in the proportion of ¼ lb. of each per ton of solids caused the same degree of settlement to take place in 1¼ min. as in 11 min. with the untreated concentrate.

It was decided, however, to install sufficient tank capacity in the new slime plant to thicken the slime at 95 per cent. or better efficiency without the use of any chemical. This of course increased the cost of installation but reduced the subsequent operating expense. However, the use of chemicals offers a cheap and simple means for increasing the settling capacity of a given installation.

(*e*) *The Effect of Temperature upon the Settlement of Slime.*—The range in temperature of the average pulp from our mill is from a minimum of about 38° F. during the winter months to a maximum of 50° F. during the summer months. There is an average daily range of about 6 per cent.

C. D. Demond, head of the testing department of the Washoe Reduction Works, made an interesting series of experiments to determine the effect of temperature upon the rate of settlement of our slime. Portions of slime pulp taken from the main flume during the winter months were used in these experiments. The rates of settlement at the flume

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temperature and at other higher temperatures were determined, the pulp being treated by immersing the container in warm water. A depth of pulp of 17½ in. was used and the slime was considered to have settled when the thickened pulp occupied a volume equal to one-fifth of the original. Thus the density had been increased from 2 to 10 per cent. solids. The following table shows the length of time required to settle the Anaconda concentator slime from a depth of 17½ in. to 3½ in.:

| Temperature, Degrees F. | Time in Minutes | | |
|---|---|---|---|
| | Average | Maximum | Minimum |
| 35 | 41.7 | 57.0 | 29.5 |
| 40 | 38.0 | 50.7 | 27.3 |
| 45 | 35.0 | 45.2 | 25.0 |
| 50 | 31.7 | 40.0 | 23.0 |
| 55 | 28.5 | 36.0 | 20.5 |
| 60 | 26.0 | 33.0 | 18.3 |
| 65 | 23.3 | 29.8 | 16.0 |
| 70 | 21.2 | 26.7 | 13.8 |
| 75 | 19.5 | 25.0 | 12.0 |

The above tabulation shows that the *average* slime settles 24 per cent. faster at the maximum summer temperature of 50° than at the minimum winter temperature of 38°. However, we find that the deviations in time from that of the average slime are so great, due to physical differences in the pulp other than temperature, that we may have a slime pulp from our mill which at a temperature of 40° will settle in 27.3 min. while pulp from the same source at another time may require 33 min. to settle at a temperature of 60°, or 20° higher.

Thus it would seem that from a practical standpoint and in designing a slime-settling plant the temperature of the pulp does not play as important a part as other physical characteristics. Undoubtedly the varying ratio of sand to colloid is the most important factor to be considered.

As far as the effect of temperatures met with in ordinary practice upon the recovery of valuable minerals from the thickened pulp on any form of mechanical concentrator is concerned, the operation of our experimental plant covering both summer and winter months showed this to be negligible.

## V. Description of the Anaconda Slime Plant

Following is given a brief description of the new Anaconda slime plant which was put into operation on Mar. 13, 1914. The plant consists of three divisions, the slime-thickening division, the concentrating division, and the concentrate-dewatering division. It is designed to treat daily

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26,000,000 gal. of slime pulp carrying 2,500 tons of solid matter, this being the amount of slime pulp resulting from the treatment of 12,500 tons of ore in the mill.

### 1. *Diagrammatic Flow Sheet of Plant*

In Fig. 1 is given a diagrammatic flow sheet of the plant showing number of machines, tonnages of solids per 24 hr., and gallons of pulp per 24 hr.

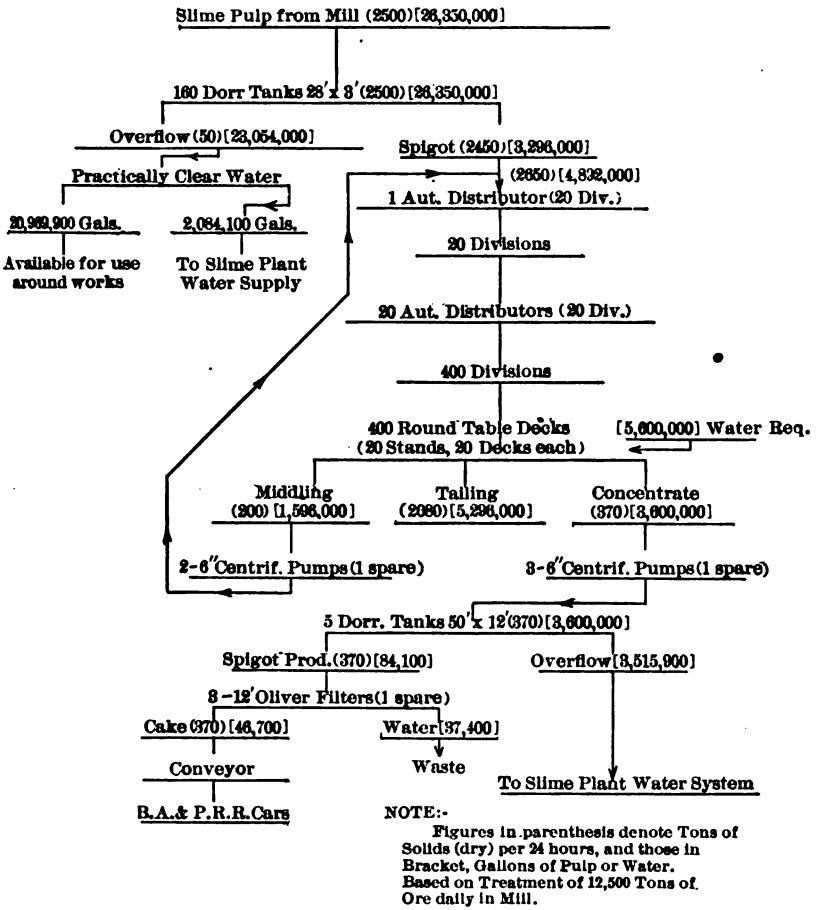
Slime Pulp from Mill (2500) [26,350,000]
160 Dorr Tanks 28' x 3' (2500) [26,350,000]
Overflow (50) [23,054,000]
Spigot (2450) [3,296,000]
Practically Clear Water
(2850) [4,832,000]
1 Aut. Distributor (20 Div.)
20,969,900 Gals.
2,084,100 Gals.
20 Divisions
Available for use around works
To Slime Plant Water Supply
20 Aut. Distributors (20 Div.)
400 Divisions
400 Round Table Decks (20 Stands, 20 Decks each)
[5,600,000] Water Req.
Middling (200) [1,596,000]
Tailing (2080) [5,296,000]
Concentrate (370) [3,600,000]
2-6" Centrif. Pumps (1 spare)
3-6" Centrif. Pumps (1 spare)
5 Dorr. Tanks 50' x 12' (370) [3,600,000]
Spigot Prod. (370) [84,100]
Overflow [3,515,900]
8-12' Oliver Filters (1 spare)
Cake (370) [46,700]
Water [37,400]
Conveyor
Waste
To Slime Plant Water System
B.A.& P.R.R. Cars
NOTE:-
Figures in parenthesis denote Tons of Solids (dry) per 24 hours, and those in Bracket, Gallons of Pulp or Water.
Based on Treatment of 12,500 Tons of Ore daily in Mill.

FIG. 1.—FLOW SHEET OF ANACONDA SLIME PLANT.

### 2. *The Slime-Thickening Division*

The feed to this division consists of the original slime pulp from the mill at a density of about 2 per cent. solids and amounting to about 26,000,000 gal. per day carrying 2,500 tons of solids. This division contains 160 Dorr thickeners 28 ft. in diameter and 3 ft. deep, with central

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feed and peripheral overflow, arranged in batteries four tanks high (Fig. 2). Each tank is provided with two rake arms set horizontal and making 1 rev. in 12 min. About ¼ h.p. is required per tank, or 50 h.p. for the entire division, including transmission. The tanks are operated at an efficiency of about 98 per cent. The overflow water, amounting to about 23,000,000 gal. per day, is available for use in the concentrating division and around the reduction works. The concentrating division requires only about 2,000,000 gal. of this water. The spigot discharge of the tanks,

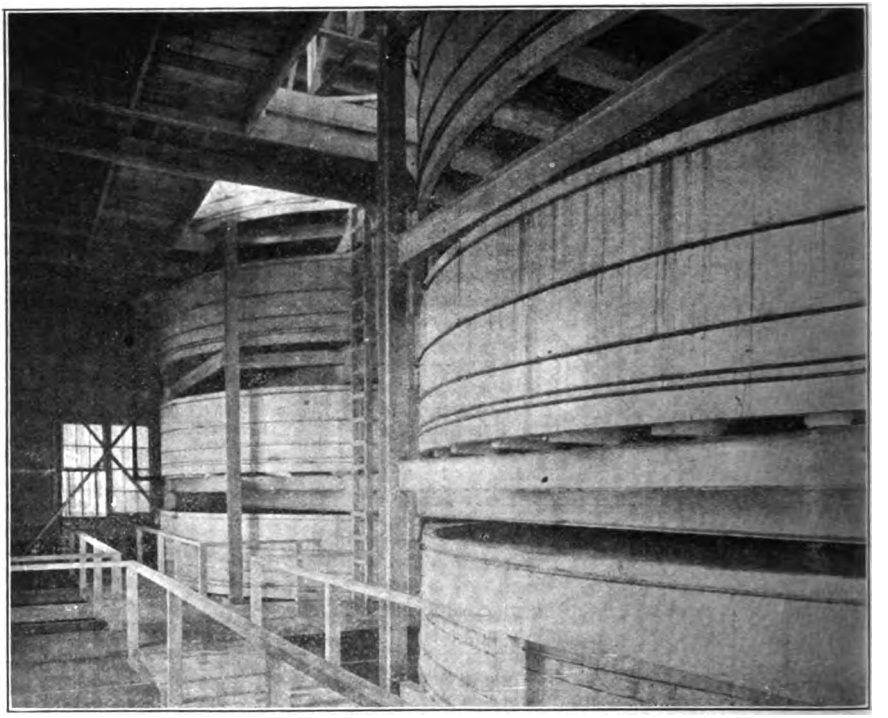

FIG. 2.—28 BY 3 FT. DORR THICKENERS IN SLIME-PULP THICKENER DIVISION.

which is the feed to the concentrating division, averages about 15 per cent. density. Four men per shift are employed in this division.

### 3. *The Concentrating Division*

This division receives as feed the thickened pulp from the thickening division, amounting to about 3,333,000 gal. daily carrying 2,450 tons of solids. The pulp passes to a central automatic distributor which divides it into as many portions as there are concentrators in operation. The division contains 20 20-deck Anaconda multiple concentrators (round tables) constructed entirely of concrete and steel. This table is described

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in detail earlier in this paper under the heading "Slime Concentrators." Each concentrator is provided with an automatic pulp distributor which gives to each deck its proportionate amount of feed. Under full feed conditions each deck will receive 8,200 gal. of pulp daily carrying 6.1 tons of solids. The middling, amounting to about 1,600,000 gal. daily carrying about 200 tons of solids, or 8 per cent. of the original solids, is returned to the original feed by means of a Traylor 6-in. sand pump. The lift is approximately 45 ft., exclusive of friction. In addition to a middling product the tables make a finished concentrate and a tailing. The concentrate, amounting to about 3,600,000 gal. per day carrying

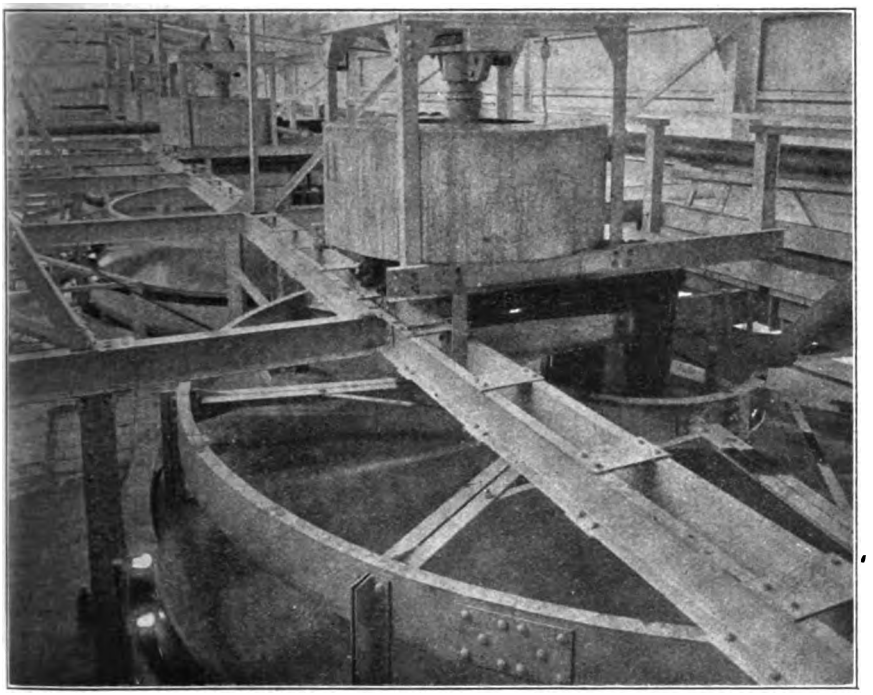

FIG. 3.—TOP DECK OF ANACONDA MULTIPLE-DECK CONCENTRATOR AND FEED DISTRIBUTOR.

about 370 tons of solids, or 15 per cent. of the original slime, is pumped by two 6-in. Traylor two-stage pumps to the concentrate-dewatering division. The lift, exclusive of friction, is about 105 ft. The tailing is being settled and recovered in the old slime ponds for future treatment. The total concentrating area is approximately 108,000 sq. ft., or 44 sq. ft. per ton of original slime. This enormous concentrating area is contained in a building 275 by 60 ft. Each table requires not over 3 h.p., or about 70 h.p. for the plant, including the driving of the distributors. There are five tablemen, each man having 80 decks under his care, and one pump-

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man, employed in this division per shift. The middling and concentrate pumps require about 260 h.p., of which the two concentrate pumps use about 210 h.p.

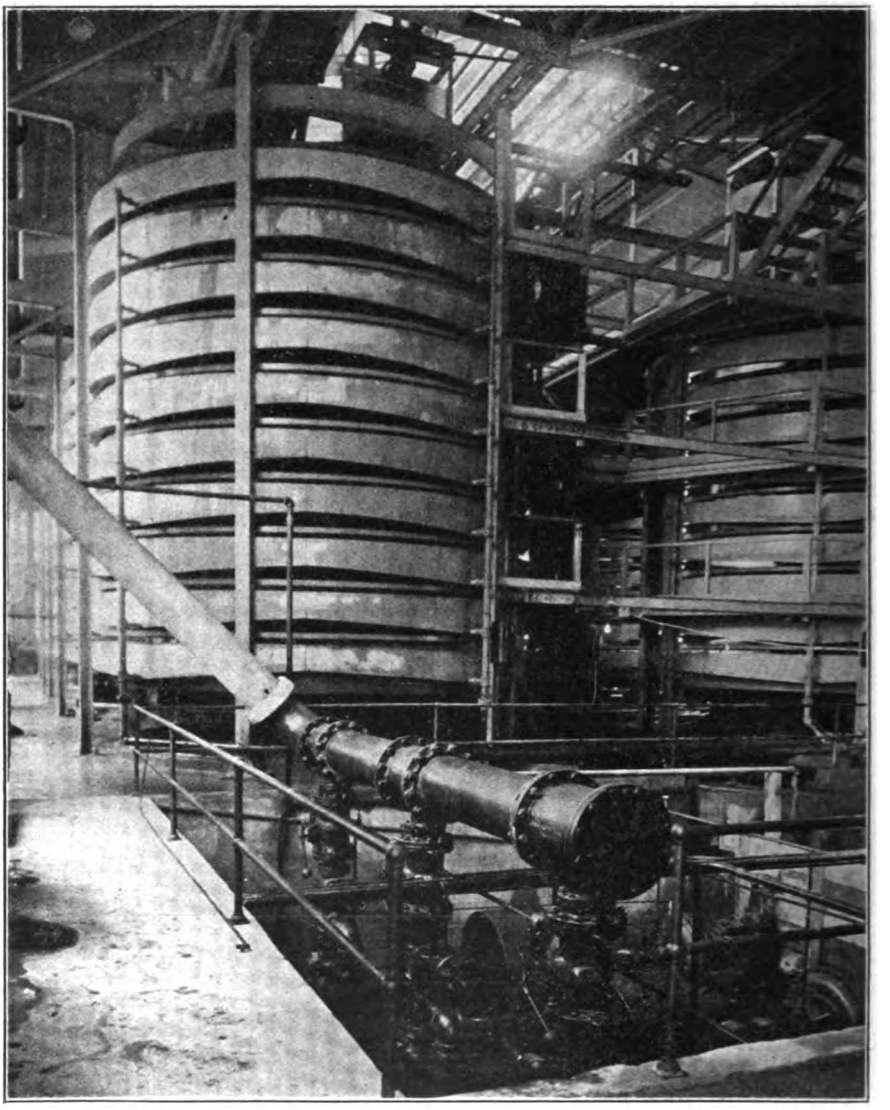

FIG. 4.—ANACONDA MULTIPLE-DECK SLIME CONCENTRATOR.

### 4. *The Concentrate-Dewatering Division*

The feed to this division—concentrate pulp from the concentrating division—passes first into five 50 by 12 ft. Dorr thickeners. Each thick-

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ener is equipped with four rake arms and requires about 1 h.p. The pulp is here thickened from a density of about 2.5 per cent. to one of about 50 per cent. solids. The overflow is clear water and is returned to the water-supply tank for the concentrating division. The thickened pulp passes to two 12 by 12 ft. Oliver continuous filters which produce a cake carrying about 15 per cent. moisture. This is comparatively a dry cake, the material being dry enough to fall apart after pressing in the hand. The cake is discharged on to a conveyor-belt system which dumps directly into 50-ton steel cars for shipment to the smelter. Three men per shift are employed in this division. The Oliver filters require about 30 h.p. each.

### 5. *General*

In addition to the labor enumerated in the preceding pages there are employed one superintendent on day shift only, one foreman on each shift, one sampler on each shift, and a repairman and helper on day shift only. This makes the total labor requirement 48 men per day. The total power requirement for the plant is about 450 h.p., of which 210 h.p. is required for pumping the concentrate. The total cost of treatment per ton of slime will be about 12c.

The results of the first few weeks of operation have come entirely up to our expectations as regards recovery and grade of concentrate. The plant has operated very smoothly from the start, not having been shut down since it was first put into operation. The entire plant is as nearly automatic in its operation as it is possible to make a concentrating plant.

It is an interesting fact from the economic standpoint that the slime plant is returning fully 200 per cent. on the investment.

The mechanical construction of the plant was carried on in a most satisfactory and efficient manner by the Washoe Reduction Works engineering department, under the direction of U. A. Garred, chief engineer, who was later succeeded by W. N. Tanner. The plant was completed and in operation fully a month and a half before the time set at the beginning of the construction work. The work was under the immediate supervision of W. C. Capron, assistant chief engineer, and George E. Tryon, construction engineer.

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# Electrostatic Separation at Midvale

BY H. A. WENTWORTH, BOSTON, MASS.

(Salt Lake Meeting, August, 1914.)

THE Huff electrostatic plant of the United States Smelting Company operated in conjunction with its wet concentrator at Midvale, Utah, was the second plant of substantial size installed using the Huff process, and the first plant to be put in operation on the so-called Western complex ores.

In spite of the machinery being of early design and consequently embracing many of the mechanical weaknesses incident to a pioneer plant the mill has operated steadily and uniformly since the summer of 1909, about five years now, without any material change other than some re-arrangement of the machinery for better handling of the sizes.

The ore, at present coming almost entirely from the company's mines in Bingham, is of practically the same composition as that upon which the plant was first put in operation.

The crude ore, consisting of the sulphides of copper, lead, zinc, and iron, and containing small amounts of gold and silver, is brought by train and delivered to the hoppers of the wet concentrator, where by jig and table treatment, there are produced a shipping lead concentrate, a tailing and a middling product, the latter being conveyed in push cars to the adjoining electrostatic mill.

The crude ore delivered to the wet concentrator contains from 6 to 9.5 per cent. zinc. At times the plant is used also for the treatment of custom ores from the Bingham district and elsewhere, and at such times the results of the plant vary according to the material in use, but as by far the larger portion of the product is that from the company's own mines, the results given below are fairly indicative of the general work obtained.

The accompanying diagram illustrates graphically the flow of the ore through the electrostatic mill. The middling coming from the wet mill containing from 15 to 18 per cent. moisture is hoisted while on the cars by an elevator, and delivered from the top of the elevator to a hopper over the drier. The drier first installed was of too low capacity to take care of the moisture present in the tonnage to which the mill was later

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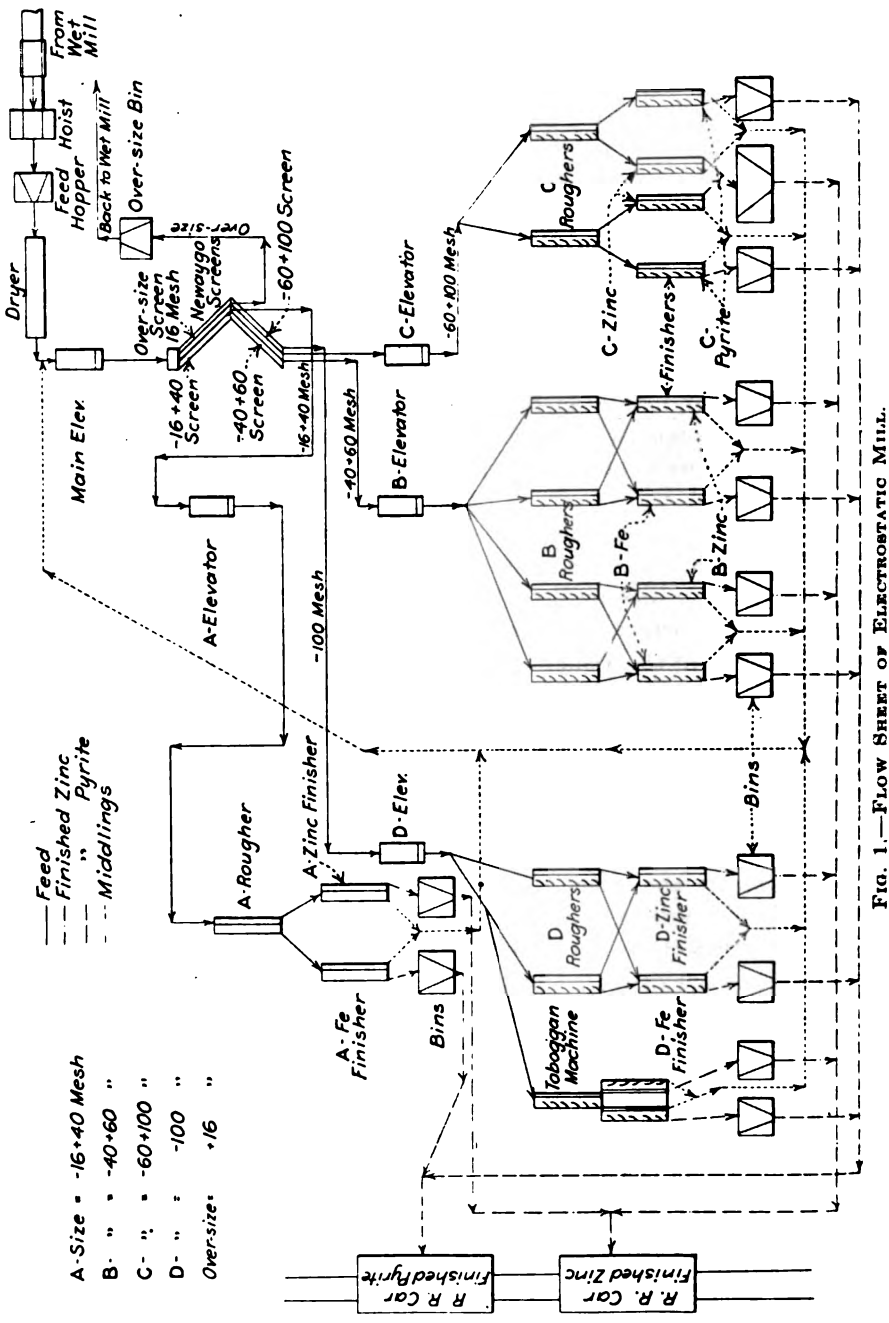


FIG. 1.—FLOW SHEET OF ELECTROSTATIC MILL.

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raised, and a second drier was placed in series with it. Drying material of this nature requires in the neighborhood of 1 lb. of coal for 5 to 7 lb. of water to be evaporated, the higher the moisture the somewhat more efficient the use of the fuel, due to the heat wasted in raising the temperature of the ore as a whole, being a constant.

The drier delivers the bone-dry material to the boot of the main elevator, where it is raised, and delivered at the top of the mill to two Newaygo screens set in series, each having a double vibrating surface. The feed to these screens is practically all through 16 mesh, and mostly through 30 mesh. The top vibrating screen of the first Newaygo is a 16-mesh scalper, removing the oversize material, chips, etc., which are delivered to a bin and returned again to the wet mill. The Newaygos produce four sizes, through 16 on 40 mesh, through 40 on 60 mesh, through 60 on 100 mesh, and through 100 mesh. Of the total feed, 16 per cent. passes a 200-mesh screen; and of the material in the through 100-mesh size, which is later treated by the so-called fines machines, 46 per cent. is through a 200-mesh screen.

From the Newaygos, which deliver on to the second machine floor of the mill, each size of material passes to its separate elevator and is delivered on the third floor to small hoppers which in turn empty into the separators below.

The separators are for the most part arranged in units of three, one being above the other two. Other arrangements are indicated on the flow sheet, the idea being to make a rough split on the first machine of the minerals to be separated, and the final cleanup on the separators below. The arrangement of the units is largely dependent on the percentage of the minerals in the given material to be separated, and the machines are arranged as nearly as possible to keep a uniform load.

Each unit produces three products, a finished "iron," a finished "zinc," and a middling product. This middling product is returned to the boot of the main elevator and again passed into the Newaygo screens. This middling is due partly to mechanical entanglement, and partly to the fact that abrasion in passing through the mill has broken up some of the minerals; so by passing all the material again over the screens, it unites with the warm material coming from the drier and is therefore again thoroughly dried and screened into its proper sizes.

The finished products fall into bins under the main floor of the mill, and are thence conveyed by hand cars to the railroad cars outside the mill building.

The separators are electrically energized by a small self-contained electric unit maintained in a dust-free room at the side of the mill. This unit consists of a 3-h.p. motor, belted to a special generator and exciter. The potential is raised by transformer and rectifying device to 18,000 to 22,000 volts which is the potential used on the machines (at

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this plant). The potential generated in this way is practically uniform, and varies only with such slight variations as are present in the speed of the motor. One terminal of the potential line is grounded direct and the machines are likewise grounded. From the other terminal, a highly insulated wire runs through the mill, being connected to the so-called "attracting" electrodes of the separators, of which there is one to each roll or separating element. The electrical equipment in this plant is considerably more massive than the equipments now being put on the market.

The principle of the process and the general apparatus in use have been described in some detail during the past four years in several publications. It was given by the writer at the New York meeting of this Institute in February, 1912. Numerous improvements have been made since that time, one of the most important from an operating standpoint being the substitution of bare-metal "electrodes," which has been made possible by the institution of a regulating device in the circuit to the machines. This substitution of the metal for the wooden electrodes has at Midvale reduced the cost of maintenance of the electrodes 75 per cent. Further improvement along these lines has been made, and it is believed that shortly any cost for electrode maintenance will be entirely eliminated. Since the construction of the Midvale plant, a separator has been designed called the "Toboggan" type. The only moving part in this is the feed roll; belts and bearings being eliminated. One of these separators is at present in operation at Midvale and accomplishes about the same result as a unit of three of the roll type. Elsewhere this type is in operation handling all the material through 20 mesh.

The quality of the work in the Midvale plant is largely dependent upon the character of the blende which varies from time to time. The zinc sulphide crystals in the ores from the United States company's mines contain chemically combined from 3 to 5.5 per cent. iron. When the iron content of the zinc sulphide crystals is under 4 per cent., it is not difficult to make a 48 to 49 per cent. blende and at the same time keep the zinc content of the pyrite product around 11 per cent. or under. When, however, the zinc sulphide crystals contain much over 4 per cent. iron, it is impossible to keep the grade of the blende up above 48 per cent. unless considerable zinc is run into the pyrite product. The zinc present in the pyrite product consists partly of attached particles and partly of an intimate mixture of zinc-lead mineral, and partly the blende high in iron. The impurities left in the zinc product are largely gangue, left in the middling in the wet mill, and small amounts of attached minerals. That the percentage of iron chemically combined with the blende is not a criterion in judging the conductivity of the blende, is illustrated by the fact that some blendes, containing as high as 14 per cent. iron in

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the crystals, are separated electrostatically. It is not known definitely what determines this conductivity.

Typical analyses of the wet-mill feed, and the head and resulting products of the static mill are as follows (the impurities of minor metals in each of the products being largely as minute attached particles):

| | Au | Ag | Pb | Cu | Fe | Zn | $SiO_2$ |
|---|---|---|---|---|---|---|---|
| Head, wet mill | 0.08 | 3.8 | 8.4 | 0.41 | 14.3 | 9.0 | 28.8 |
| Middling to static | 0.07 | 3.4 | 2.2 | 0.96 | 21.6 | 26.8 | 4.6 |
| Blende | 0.02 | 2.0 | 1.1 | 0.71 | 5.4 | 47.8 | 8.3 |
| Pyrite | 0.10 | 3.8 | 2.6 | 1.63 | 32.0 | 12.0 | 2.4 |

The capacity of the present mill when running full is approximately 65 tons, and on the various sizes the units have the following capacity: on 40 mesh, 12 tons; on 60 mesh, 10 tons; on 100 mesh, 8 tons; through 100 mesh, 5 tons. The Midvale ore is a comparatively difficult one with which to obtain the best results, and the tonnage per unit is therefore lower than with a more simple ore. The separators for crude ore are of much larger capacity.

Six mills of this general type are in operation in the Western zinc field, some modifications being required to meet local conditions in each case; as well as plants in operation on other classes of work as described in the articles above mentioned.

The writer is much indebted to Messrs. Heintz, Anderson, and Lemke for their kind assistance in furnishing recent data on the plant, and for preparing the flow sheets of the mill.

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# The Occurrence of Bournonite, Jamesonite, and Calamine at Park City, Utah

BY FRANK ROBERTSON VAN HORN, CLEVELAND, OHIO

(Salt Lake Meeting, August, 1914)

## Introduction

In June, 1911, the writer spent a few days in studying the economic geology of the vicinity of Park City. During this rather hurried visit a number of specimens of ore were collected, which have since been examined. The results of these investigations show that bournonite ($PbCuSbS_3$) is present in the ores in greater or less quantity. This fact has never before been reported, as far as can be learned. The failure to recognize this mineral is due to its similarity to tetrahedrite, with which it has probably been confused during a long period of years.

Jamesonite ($Pb_2Sb_2S_5$) has been reported from the region in a rather doubtful manner, but the writer has seen it in notable amounts from several mines, and can offer the first analysis of this mineral from the district.

Calamine ($H_2Zn_2SiO_5$) was found at one mine and has never been recognized and reported from the district.

## Geography and Geology

Park City, Utah, is located at an elevation of about 7,000 ft. above sea level on the eastern slope of the Wasatch mountains. It is situated approximately 25 miles southeast of Salt Lake City. The geology and ore deposits have been described in detail by Boutwell and Woolsey.[1]

The mines visited are situated south-southeast from the town. The Silver King Coalition mine is located in Woodside gulch about 1 mile southeast of Park City, at an altitude of about 8,100 ft. The other mines visited were the Daly West and Quincy, which are in Empire canyon about 2 miles southeast of the town, at elevations of 8,300 to 8,500 ft. respectively. The Daly Judge tunnel goes into the mountain from Empire canyon about 1 mile south of the city, at an altitude of 7,700 ft.

[1] Geology and Ore Deposits of the Park City District, Utah, *Professional Paper No. 77, U. S. Geological Survey* (1912).



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About two-thirds of the area of the district is covered by sedimentary rocks, and the remainder are of igneous origin. According to Boutwell,[2] the sedimentaries are divided as follows:

| | Feet | |
|---|---|---|
| Ankareh | 1,150+ | Triassic |
| Thaynes formation (limestone, sandstone, shale) | 1,190 | Triassic |
| Woodside shale | 1,180 | Triassic |
| Park City formation (limestone, quartzite, sandstone, shale) | 590 | Permian (?) |
| Weber quartzite | 1,350+ | Pennsylvanian |

The sediments were intruded from Cretaceous to Eocene by diorite, diorite porphyry, and finally andesite. It was these igneous rocks, especially the earlier ones, which are thought to have caused the formation of the ores of the district. Both previously and subsequently to the deposition of the ore, more or less faulting has taken place.

## Occurrence of Ores and the Common Ore Minerals

The ores of the region are found both in fissure veins, and as replacement deposits in limestones. The two types are often associated, as in the Silver King Coalition and Daly West mines. The fissures have a general northeast-southwest trend. Most of the replacement or bedded deposits are found in the limestones of the Park City formation, although a few occur in the limestones of the Thaynes formation.

The commonest sulphide minerals, according to previous writers, are galena, tetrahedrite, sphalerite, and pyrite. To this rather short list bournonite can be added. At one time the common occurrence of tetrahedrite in the region was doubted by the writer but now it is known that both tetrahedrite and bournonite are fairly common ore minerals. It is certain that in the past more or less bournonite has been called tetrahedrite, but which of the two minerals predominates is still a matter of conjecture. The chief oxidation products are cerussite and anglesite, with small amounts of malachite and azurite. According to Boutwell,[3] considerable amounts of bindheimite ($Pb_3Sb_2O_8+aq$) are found. This would seem to indicate that bournonite might be a more prominent constituent of the ores than tetrahedrite, because the former is a lead-antimony sulphide, whereas the latter is a copper-antimony compound, and would not oxidize directly to bindheimite without interaction of other substances.

One interesting feature which the writer has never before seen under natural conditions was observed on "coarse cleavable" galena from the Silver King Coalition mine. It was a polysynthetic twinning in two

[2] *Op. cit.*, p. 44. [3] *Op. cit.*, p. 114.

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directions perpendicular to each other, although one set of striations was much better developed than the other (see Fig. 1). The twinning lines make angles of 45° with the cleavage cracks after the cube. In other words, the twinning plane is parallel to the diagonals through a cube face or to the secondary planes of symmetry, and consequently must be the face of a rhombic dodecahedron. For galena, this face is a gliding plane, and the phenomenon may be produced artificially by placing cubes of galena in a vise and applying pressure. A similar behavior of calcite is perhaps more widely known, where a proper application of pressure will produce artificial twin crystals parallel to the face of the negative rhombohedron $\frac{1}{2}R$. To the writer the presence of such striations on galena, indicates pressure after the deposition of the mineral, and the lines were undoubtedly formed by the pressures caused by the postmineral faulting

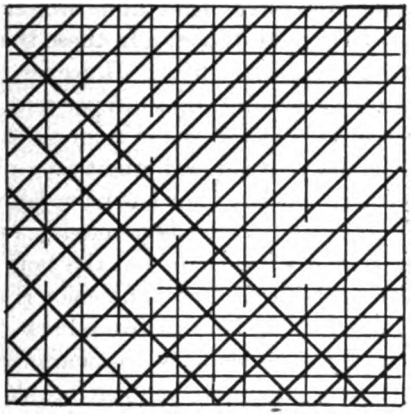

FIG. 1.—GALENA WITH CUBICAL CLEAVAGE PARALLEL TO THE EXTERIOR, SHOWING POLYSYNTHETIC TWINNING DUE TO PRESSURE ALONG THE CUBE DIAGONALS.

of the district. Faults of this character have been recognized in such mines as the Silver King and Daly West.

## OCCURRENCE OF THE BOURNONITE

Bournonite is a very rare mineral, and has been reported previously from but three or four localities in the United States. The writer first saw a crystal of the mineral from Park City in the magnificent collection of the late Albert F. Holden, of Cleveland. A. T. Dalley, of the Silver King Coalition Mines Co., gave the writer a crystal which is now in the mineral collection of Case School of Applied Science at Cleveland. At the time, Mr. Dalley had in his possession another bournonite crystal, which he later donated to the Geological Department of the University of Utah, at Salt Lake City. It was also learned that the crystal in the

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Holden collection was originally obtained from Mr. Dalley. The three crystals came from the 1,300-ft. level of the Silver King Coalition Mine and as far as the writer knows there are no others like them from the Park City district in existence. Mr. Dalley did not see the crystals in the mine and therefore does not know the exact conditions under which they were found. He has since seen other "smaller crystals that are in quartzite associated with other sulphides such as pyrite, galenite, sphalerite, and jamesonite."[4] Because of the rarity and size, the dimensions and weights of these crystals are given, as follows:

| | Grams |
|---|---|
| No. 1. Holden Collection, now at Harvard University, $3\frac{1}{2} \times 3\frac{1}{2} \times 6$ cm..... | 185.0 |
| No. 2. Case School of Applied Science, $2 \times 2\frac{1}{2} \times 4$ cm..... | 61.7 |
| No. 3. University of Utah, $1\frac{1}{2} \times 1\frac{1}{2} \times 3\frac{1}{2}$ cm..... | 44.4 |

Other specimens from the Silver King mine were given to the writer by the Superintendent, George D. Blood. In these the mineral was massive and was supposed to be tetrahedrite. However, most of the specimens contain lead in varying amounts up to 15 per cent., and are probably bournonite, as they resemble that mineral in color and luster. It is nevertheless possible that the massive mineral might be a plumbiferous tetrahedrite, which is a very rare variety, but has been found at several other places. Bournonite and tetrahedrite, when massive, resemble each other very closely. As far as the writer can see, the bournonite appears to possess a more brilliant metallic luster. The color is blackish lead-gray and the streak is black. In tetrahedrite from Park City, the luster is duller; the color has none of the lead-gray appearance, but is more like iron-black; and the streak is distinctly reddish brown instead of black. Possibly more than one variety of tetrahedrite occurs in the district. Chemically it is also difficult to distinguish the two minerals, especially if the tetrahedrite is plumbiferous, which the writer believes to be the case at Park City, because both minerals might contain lead, copper, antimony, and sulphur. In general, however, lead should predominate in the bournonite, and copper in the tetrahedrite. A quantitative analysis is necessary to determine the two minerals from a chemical point of view.

Mr. Talbot, the Superintendent of the Daly West mine, gave the writer several specimens of so-called "tetrahedrite" from the lower levels (1,000–1,200 ft.). One of these showed crystals on which we could not recognize any of the forms of tetrahedrite, but which seemed to possess an orthorhombic habit. These crystals were quite rough and much distorted, and seemed to disappear into a massive mineral, which had the same color and luster as the crystals, and which was in our estimation certainly the same mineral as the crystals. Toward the center of the

[4] A. T. Dalley: Personal communication, Oct. 9, 1913.

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specimen pyrite in pentagonal dodecahedrons was found, and this was followed by coarse cleavable galena. Portions of some crystals and the adjacent massive mineral were broken off as carefully as possible, and analyzed by Dr. W. R. Veazey, of the Chemical Department of Case School of Applied Science, with the following results:

| | Theoretical Composition of Bournonite Based on Formula $PbCuSbS_3$ | Analysis of Daly West Mineral | Combining Weights | Ratio |
|---|---|---|---|---|
| Pb | 42.54 | 43.18 | 0.209 | 1.009 |
| Cu | 13.04 | 13.14 | 0.207 | 1.000 |
| Sb | 24.64 | 25.03 | 0.208 | 1.004 |
| S | 19.77 | 19.59 | 0.611 | 2.951 |
| | 100.00 | 100.94 | | |
| | | (Sp. gr., 5.829) | | |

An inspection of the results obtained shows conclusively that the mineral is bournonite, and that the ratio obtained from our analysis is very close to the theoretical composition required by the formula $PbCuSbS_3$. The specific gravity, 5.829, also conforms well to the average density obtained from specimens of the mineral from other localities. In all specimens seen by the writer the mineral was associated with pyrite and coarse cleavable galena, and was evidently formed later than either.

## Occurrence of the Jamesonite

Jamesonite is another rare mineral, and has been found at but two or three localities in the United States. In the very able monograph on the Park City district, Boutwell[5] says: "Three specimens of a crystalline gray metallic mineral were found on the California dump. . . . . The material was too meager to permit thorough chemical determinations, but Dr. Hillebrand obtained qualitative tests for sulphur, antimony and a salt of lead, and accordingly is inclined to regard the mineral as jamesonite or warrenite." Under "Bournonite" the present writer has mentioned a personal communication from A. T. Dalley which shows at the Silver King Coalition mine an association of jamesonite with bournonite, galena, pyrite, and sphalerite. In the same letter Mr. Dalley says: "The best jamesonite (fibrous) occurs at the Daly Judge mine with (most commonly) pyrite, occasionally pyrite and galenite." In the Case collection are notable amounts of a mineral from the Silver King Coalition mine. It occurs in capillary crystals which are woven together into a felt-like mass. The mineral has a dark lead-gray color, a brilliant metallic luster, and a black streak. In one specimen the mineral rests upon a

[5] *Op. cit.*, p. 109.

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perfect striated cube of pyrite. In three other specimens the jamesonite rests upon coarse cleavable galena, which in turn incloses pyrite. It would therefore seem that the jamesonite was the latest mineral to form. The felt-like mass was found to contain many small crystals of pyrite, which were carefully isolated so that an analysis of the capillary crystals could be made. The results obtained by Dr. W. R. Veazey were as follows:

| | Theoretical Composition of Jamesonite Based on the Formula $Pb_2Sb_2S_5$ | Silver King Mineral |
|---|---|---|
| Pb..... | 50.84 | 55.75 |
| Sb..... | 29.46 | 21.34 |
| As..... | 00.00 | 4.14 |
| S..... | 19.70 | 18.43 |
| | 100.00 | 99.66 |

Our analysis shows that the mineral investigated conforms in general to the theoretical composition of jamesonite. It was found on comparing 38 different analyses[6] that there is a wide variation in the constituents between certain limits. The lead in our analysis is higher than the average, although one analysis of material from Russia contained 63.61 per cent. One feature of the Park City mineral, which is not common in most jamesonite, is that part of the antimony has been replaced by arsenic. It would therefore seem that a small amount of dufrenoysite ($Pb_2As_2S_5$), which is generally regarded as isomorphous with jamesonite, must be present in the Silver King mineral. In order to compare this analysis with most jamesonite analyses, the arsenic was converted into antimony (4.14 As = 6.63 Sb), after which the combining weights and molecular ratios were obtained as follows:

| | | Combining Weights | Ratio |
|---|---|---|---|
| Pb..... | 55.75 | 0.270 | $1.16 \times 2 = 2.32$ |
| (Sb, As)..... | 27.97 | 0.233 | $1.00 \times 2 = 2.00$ |
| S..... | 18.43 | 0.575 | $2.47 \times 2 = 4.94$ |

It will be seen that the ratio of the recalculated analysis conforms quite closely to the theoretical formula $Pb_2(Sb,As)_2S_5$. If the original analysis be compared with the theoretical composition of dufrenoysite on one side, and that of jamesonite on the other, it will be found that, in general, the Park City jamesonite stands between the two, and is therefore probably an isomorphous growth of the two minerals.

[6] Hintze: *Handbuch der Mineralogie*, pp. 1031, 1032.

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| | Dufrenoysite, $Pb_2As_2S_5$ | Park City Jamesonite, $Pb_2(Sb, As)_2S_5$ | Jamesonite, $Pb_2Sb_2S_5$ |
|---|---|---|---|
| Pb | 57.1 | 55.75 | 50.84 |
| As | 20.7 | 4.14 | 0.00 |
| Sb | 00.0 | 21.34 | 29.46 |
| S | 22.2 | 18.43 | 19.70 |
| | 100.0 | 99.66 | 100.00 |

The only discrepancy is in the sulphur, which is a trifle low. It is also to be remembered that our analysis reached a total of 99.66 per cent. and was not calculated on the basis of 100 per cent.

## Occurrence of the Calamine

Calamine ($H_2Zn_2SiO_5$) has never been reported before from the Park City region, as far as can be ascertained. The mineral was found on the dump of the Quincy mine as a drusy coating on cavities and pores. Several specimens were collected. The calamine was always associated with a brownish manganese oxide resembling wad. It was evidently entirely manganous oxide, as the usual manganese reactions were not observed until the mineral was oxidized with niter. The calamine crystals were grouped in the characteristic sheaf-like masses, which were in some places assembled into botryoidal and mamillary imitative shapes. The crystals were all thin tabular after the brachypinacoid $b(010, \infty P\bar{\infty})$, which was striated vertically. The prism $m(110, \infty P)$ was present as small planes, as were also a macro- and brachy-dome [evidently $t(301, 3P\bar{\infty})$, and $i(031, 3P\infty)$]. The basal pinacoid $c(001, 0P)$ was not observed, and if present was very small. Although the writer was perfectly convinced as to the identity of the mineral, a qualitative chemical test was made for verification. The mineral was dissolved in hydrochloric acid and evaporated to dryness. It was taken up in hydrochloric acid and water, and boiled, after which the silica was filtered off. Nitric acid was added to oxidize any iron, and ammonium chloride was added to keep up any zinc. Ammonia was added to precipitate iron and manganese, and the precipitate was filtered. Hydrogen sulphide was passed through the filtrate, and a white precipitate of zinc sulphide was obtained, which fully corroborated the mineralogical determination.

The writer is unable to say whether the calamine exists in large amounts, but the discovery and associations here remind one very decidedly of the occurrence at Leadville, Colo., where large amounts of calamine and smithsonite have been recently found after having escaped recognition for many years. It is, of course, an oxidation product of sphalerite, and was probably associated with the usual minerals found in the upper

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levels of the Quincy mine, such as cerussite, anglesite, malachite, and azurite.

## Paragenesis and Conclusion

The common associated minerals, as well as the probable order of deposition, have been mentioned in the detailed discussion of the occurrence of the various minerals. The writer is, moreover, impressed with the fact that there is evidently constantly present in the ores of the Park City district, a lead-copper-sulphur-antimony series of minerals represented by galena, bournonite, tetrahedrite, and jamesonite. One gets the general impression that Park City is a camp mainly of argentiferous galena. The fact that an antimonial lead series of minerals is present is shown by the constant oxidation to bindheimite ($Pb_3Sb_2O_8+aq$), which is repeatedly mentioned by Boutwell. Study of the ores leads to the further conclusion that the sulph-antimonides bournonite, tetrahedrite, and jamesonite were all deposited later than the galena, and therefore that the lead-antimony solutions entered at a subsequent period. There are certain questions relative to depth which the writer cannot answer. The copper content of the Park City veins is evidently increasing with depth. Is this also true of the zinc? What relation have tetrahedrite and bournonite to each other? I have never found both minerals on the same specimen. What is the critical level of bournonite compared with tetrahedrite? Was bournonite more common on the upper sulphide levels, and has it simply escaped recognition during a long period of years? Is it being replaced by tetrahedrite in the lower levels as the copper content of the veins increases with depth? Or on the other hand is the bournonite now being found for the first time in the lower levels? These questions cannot be answered with the material and data at command. The suggestions are given in the hope that some of the questions may be answered by members who are more familiar with the district.

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## Curves for the Sensible-Heat Capacity of Furnace Gases

BY C. R. KUZELL AND G. H. WIGTON

The Editor desires to call attention to the charts in the paper of the above title which were published in the August *Bulletin* on pages 2190 and 2191. It was impracticable to include in the *Bulletin* charts of sufficient size to be used as working diagrams for the direct determination of operating values, but the Institute is prepared to furnish copies of these charts on a large-size scale, at a moderate cost, to those who desire to have them for this purpose.

### Corrections

Page 2185, line 3. For "is accessory apparatus" read "in accessory apparatus."

Page 2188. Substitute for lines 11 and 12:

Nitrogen, oxygen, carbon monoxide, $C_p = 6.50 + 0.0010T$
Hydrogen, $C_p = 6.50 + 0.0009T$

Page 2189. Substitute for lines 4 and 5 under Example 1:

| $H_2O$ | $CO_2 + SO_2$ | $O_2$ | $N_2$ |
|---|---|---|---|
| 8 | 15 | 2 | 75 |

Page 2192, Plate III. Scale on right-hand side is temperature in degrees centigrade.

Page 2194, lines 3 and 9. For "439,500 B.t.u." read "449,200 B.t.u." For line 15, substitute:

$$449{,}200 - 124{,}900 = 324{,}300 \text{ B.t.u.}$$

For line 20, substitute

$$\frac{\text{Heat output}}{\text{Heat input}} \times 100 = \frac{179{,}200 \times 100}{324{,}300} = 55.3 \text{ per cent.}$$

Page 2195, line 6 from bottom. For "$t$," substitute "t."
Line 8 from bottom. For "$22.x$" read "22.1."

Page 2196, center. Substitute:

$$\frac{(2{,}045 \times 22.1 \times 0.0385) + (2{,}520 \times 4.3 \times 0.0335) + (2.0 + 71.6)\,(3{,}075 \times 0.0235)}{(22.1 \times 0.0385) + (4.3 \times 0.0335) + (73.6 \times 0.0235)} = \frac{7{,}423}{2.725} = 2{,}724^\circ \text{F.}$$

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Page 2197. For line 2 *et seq.*, substitute

$$c_m(T_1 \text{ to } T_2) = \frac{\int_{T_1}^{T_2}(A + BT + CT^2 + DT^3 + \text{etc.})dT}{T_2 - T_1}$$

$$= \frac{A(T_2 - T_1) + \frac{B}{2}(T_2^2 - T_1^2) + \frac{C}{3}(T_2^3 - T_1^3) + \text{etc}}{T_2 - T_1}$$

For line 11 from bottom, substitute:

From the figure $Q_1 = Q + \bar{O}Q_1$, and $Q_2 = Q + \bar{O}Q_2$, etc.

Page 2198. For line 10 *et seq.*, substitute:

Solving for $t_1 0$ and substituting in $t = \bar{t}_1 0 + t_1$

$$t = \frac{\frac{p_1 t_1}{m_1} + \frac{p_2 t_2}{m_2} + \text{etc.}}{\frac{p_1}{m_1} + \frac{p_2}{m_2} + \text{etc.}}$$

Page 2199, Table I. For "C"$_m$ read "$c_m$."

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Surface Decarbonization of Tool Steel

BY J. V. EMMONS, CLEVELAND, OHIO

(Pittsburg Meeting, October, 1914)

IT has long been known that the outside skin of tool steel frequently exhibits properties widely different from the interior of the mass. Within the last few years it has been generally recognized that many of these properties were due to surface decarbonization. The origin and effects of this altering of the composition have been but little studied.

C. M. Johnson[1] arrives at the conclusion that, in annealing, "the most active agent in surface decarbonization is the rust or scale (present on the surface), which is actually reduced to metal at the expense of the carbon in the steel to which it adheres." W. H. Hatfield in his admirable paper, The Chemical Physics Involved in the Decarburization of Iron-Carbon Alloys,[2] has given us the most tenable theory of the removal of carbon from iron.

While Mr. Hatfield's work has been principally done in connection with the annealing of white cast iron to produce malleable castings, it is frequently applicable to the decarbonization of tool steel. The following equations which he proposed have been fully borne out by the author's experiments:

$$\left.\begin{aligned} 2Fe_3C + 2CO_2 &= 3Fe_2 + 4CO \\ 2CO + O_2 &= 2CO_2 \\ 2CO_2 + 2Fe_3C &= 3Fe_2 + 4CO \end{aligned}\right\} \text{Continuous}$$

It is the purpose of the author to show, in a quantitative manner, the effect of various treatments in eliminating the carbon in tool steel. Some of the phenomena of the decarbonized surface will also be described, as well as the practical problems they present to the manufacturer and user of tool-steel products.

A ½-in. round bar of crucible steel having the following composition was selected: C, 1.15 per cent.; Mn, 0.31; P, 0.022; Si, 0.16; and S, 0.018 per cent. A microscopic examination having revealed a slight decarbonization already present, the bar was turned down to ⅜-in. diameter to remove all traces of decarbonization. The bar was in

[1] *Journal of Industrial and Engineering Chemistry*, vol. i, No. 7, p. 459 (July, 1909).
[2] *Journal of the Iron and Steel Institute*, vol. lxxix, p. 242 (No. 1, 1909).

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the annealed condition, the structure consisting of finely divided lamellar pearlite, with many small grains of globular cementite. The bar was cut into test pieces of about 2 in. in length. The heating operations were all conducted in an electric resistance furnace of the tube type. The temperatures were regulated by means of a Le Chatelier thermo-couple.

Johnson having stated that he found a slight decarbonization upon annealing tool steel in hydrogen, this was first investigated: A test piece was placed in the furnace cold and gradually raised to a temperature of 1,800° F. in a current of carefully purified and dried hydrogen. After holding at this temperature for 3 hr. the current was shut off and the

FIG. 1.—100 DIAMETERS.

piece was allowed to cool slowly in the furnace. On removal from the furnace, the test piece had a dull-gray crystalline appearance. Microscopic examination revealed no sign of decarbonization. A second test piece was placed in the furnace in the same manner as before, held at 2,130° F. for 6 hr., and then allowed to cool slowly in the furnace. This piece exhibited the gray crystalline surface in a still more marked degree. A microscopic examination revealed a coarse network of cementite extending to the extreme edge of the cross-section, giving conclusive evidence that no decarbonization had taken place. (See Fig. 1.) This proves that hydrogen is not a factor in the elimination of carbon from steel.

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The part taken in the decarbonization of cast iron by iron oxide has been investigated by Wüst.[3] His conclusions are that ferric oxide, at the higher temperatures, gives off a part of its oxygen which combines with the carbon in the iron to form CO. The CO still further reduces the ferric oxide to ferrous oxide, being itself oxidized to $CO_2$. The $CO_2$ again reacts with the carbon in the iron according to the equation $CO_2 + C = 2CO$, the process being continuous.

There is no doubt that these reactions are very closely duplicated in the annealing of tool steel. The bars are always covered with more or less hammer scale and rust, and are packed in a closed pipe or box. The principal difference is that, as there is no free carbon normally present in tool steel, the elimination of the carbon is probably accomplished according to the equation:

$$CO_2 + Fe_3C = 3Fe + 2CO.$$

In order to confirm these reactions, and also to determine the rate at which they proceed at various temperatures, the following experiment was carried out:

Pieces from the bar above described were annealed in a strong current of pure, dry $CO_2$, at various temperatures. The furnace was brought to the required heat, the pieces placed in it, held at the heat for 3 hr., and allowed to cool slowly in the furnace. Microscopic examination was then made of the cross-section, and the depth of decarbonization measured by means of a micrometer eye piece. For greater ease and accuracy, the decarbonization was measured only from the surface to the point of supersaturation, or to the point where free cementite first made its appearance. This point is much more easily determined than the point where the interior is first affected by the elimination of carbon. Each figure of the depth of decarbonization given below is the average of four measurements made at different points of the cross-section.

| Piece No. | Temperature, ° F. | Depth of Decarbonization, inches, |
|---|---|---|
| 1 | 1,350 | 0.000 |
| 2 | 1,400 | 0.007 |
| 3 | 1,500 | 0.009 |
| 4 | 1,600 | 0.018 |
| 5 | 1,700 | 0.025 |
| 6 | 1,800 | 0.034 |
| 7 | 1,900 | 0.053 |
| 8 | 2,000 | 0.072 |

[3] *Metallurgie*, vol. v, No. 1, p. 11 (Jan. 8, 1908); *Stahl und Eisen*, vol. xxiii, 2, No. 20, p. 1136 (Oct. 15, 1903).

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A photomicrograph of the decarbonization of piece No. 3 is shown i
Fig. 2.

A micrograph of piece No. 6 is shown in Fig. 3. The structure
piece No. 8, showing the outer portion of the decarbonization only,
illustrated in Fig. 4.

These results indicate that $CO_2$ gives up to the carbon very rea
one of its atoms of oxygen. When the results are plotted in the form
a curve, with the depth of decarbonization as abscissæ and the temp

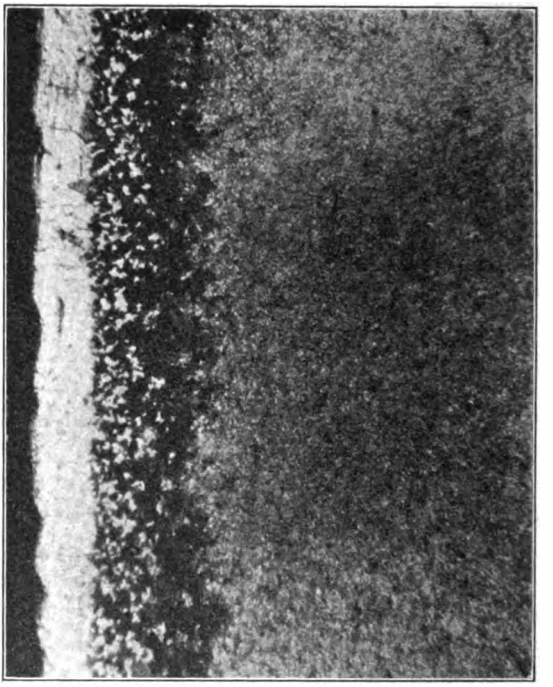

FIG. 2.—100 DIAMETERS.

ture as ordinates, it is also observed that the decarbonization incre
at a definite rate as the temperature is increased. (See Fig. 5.)

A similar experiment was then performed by annealing the test p
for 3 hr. in a tube containing air and to which air had slight ac
through a small opening in one end. As shown in the following tab
the results are very similar to those obtained in an atmosphere of $CO_2$,
with the exception that the rate of decarbonization is now slightly less
at all temperatures:

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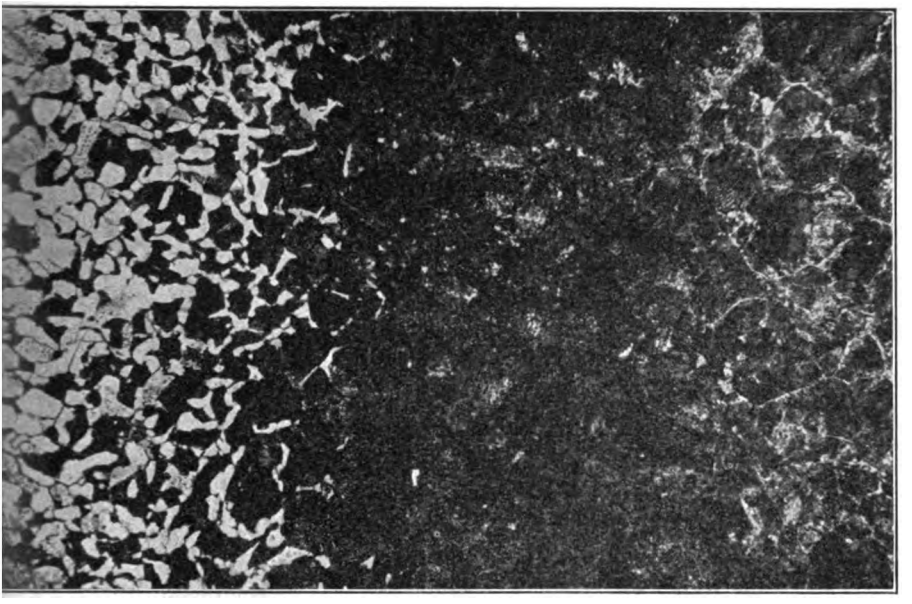

FIG. 3.—100 DIAMETERS.

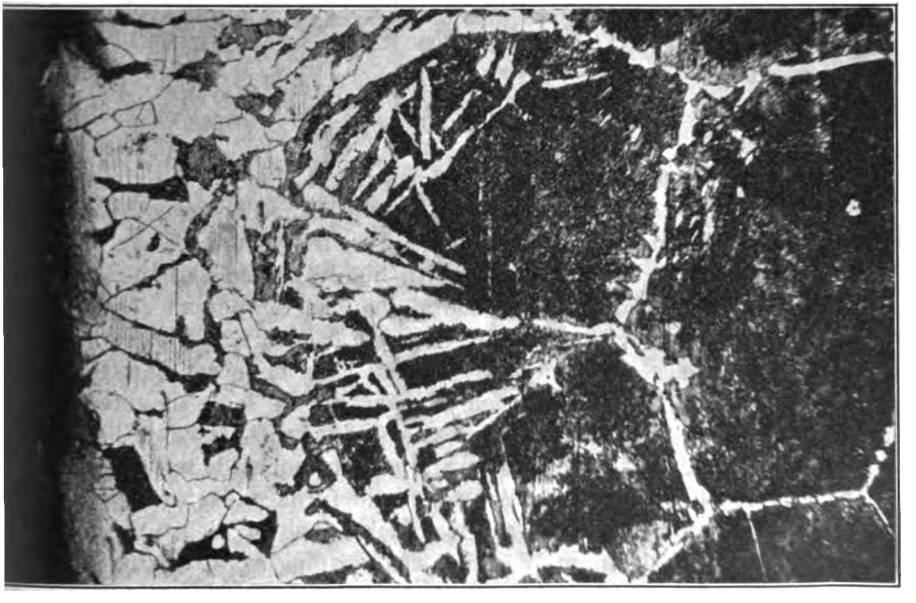

FIG. 4.—100 DIAMETERS.

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| Piece No. | Temperature, ° F. | Depth of Decarbonization, Inches |
|---|---|---|
| 1 | 1,200 | 0.000 |
| 2 | 1,300 | 0.000 |
| 3 | 1,400 | 0.004 |
| 4 | 1,500 | 0.005 |
| 5 | 1,600 | 0.014 |
| 6 | 1,700 | 0.016 |
| 7 | 1,800 | 0.025 |
| 8 | 1,900 | 0.039 |
| 9 | 2,000 | 0.064 |

The curve plotted from these results is shown in Fig. 5.

A similar experiment was then performed by annealing the pieces for 3 hr. in a current of pure, dry oxygen.

The results are very similar to those obtained in air and $CO_2$.

The curve plotted from the following table is shown in Fig. 5.

| Piece No. | Temperature, ° F. | Depth of Decarbonization, Inches |
|---|---|---|
| 1 | 1,300 | 0.000 |
| 2 | 1,400 | 0.002 |
| 3 | 1,500 | 0.006 |
| 4 | 1,600 | 0.012 |
| 5 | 1,700 | 0.017 |
| 6 | 1,800 | 0.027 |
| 7 | 1,900 | 0.040 |
| 8 | 2,000 | 0.068 |

A test piece was then annealed for 3 hr. at 2,000° F. in a current of pure, dry carbon monoxide. Microscopic examination revealed no trace of decarbonization. The CO used was first freed from oxygen and $CO_2$ by passing through a tube filled with highly heated carbon, and then through several wash bottles containing caustic potash solution. A flask of lime water was placed in the train at both the entrance and the exit of the furnace, to detect any traces of $CO_2$ either entering or escaping.

A second test piece was then annealed in the same manner, except that a clay boat filled with hammer scale ($Fe_3O_4$) was placed in the furnace in such a position that the current of carbon monoxide would pass over it before reaching the steel. As soon as the furnace reached its temperature it was observed that a gas was being evolved within the furnace. A noticeably larger quantity of gas was escaping from the furnace than was entering it. This would indicate that the $Fe_3O_4$ was probably giving up part of its oxygen. This evolution of gas continued through the entire period of annealing. The gases escaping from the furnace were

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ound to contain considerable quantities of $CO_2$. The $CO_2$ was probably ormed by the following reactions:

$$Fe_3O_4 = 3FeO + O.$$
$$CO + O = CO_2.$$
$$FeO + CO = Fe + CO_2.$$

Microscopic examination of the annealed specimen revealed that the decarbonization had penetrated to a depth of 0.096 in.

To test the effect of water upon decarbonization, a test piece was annealed at 1,700° F. in a current of steam, the water from which the

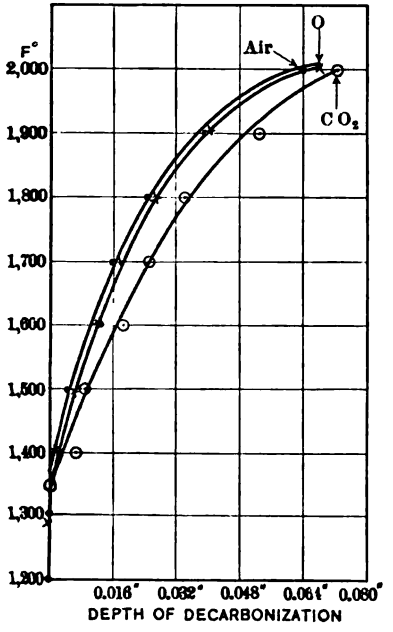


FIG. 5.—DECARBONIZATION OF STEEL ANNEALED IN AIR, OXYGEN AND CARBON DIOXIDE.

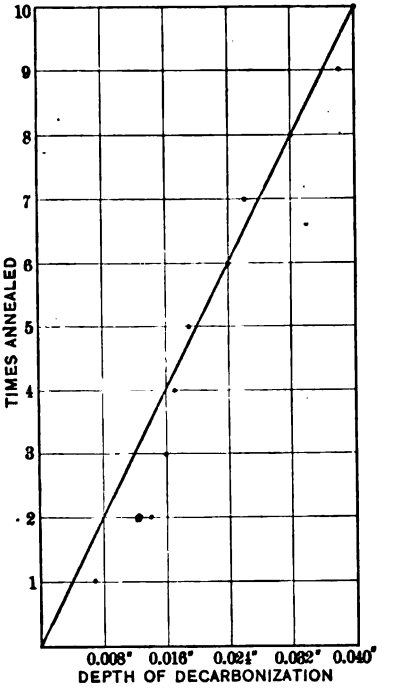


FIG. 6.—EFFECT OF REPEATED ANNEALING ON DECARBONIZATION OF STEEL.

steam was derived being first freed from $CO_2$ and air by prolonged boiling. Examination of the annealed specimen revealed a decarbonization to a depth of 0.030 in.

To test the effect of repeated annealings, a test piece was first packed in a tube with a small amount of charcoal, air being given limited access. It was then repeatedly annealed for 10 hr. each time, at a temperature of 1,450° F. The decarbonization was measured after each annealing, the

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results being shown in the following table. The action was due to the $CO_2$ formed by the charcoal and the oxygen of the air.

| No. of Times Annealed | Depth of Decarbonization, Inches |
|---|---|
| 1 | 0.009 |
| 2 | 0.015 |
| 3 | 0.017 |
| 4 | 0.015 |
| 5 | 0.019 |
| 6 | 0.024 |
| 7 | 0.026 |
| 8 | 0.032 |
| 9 | 0.038 |
| 10 | 0.040 |

The curve plotted from these results shows that the rate of increase in depth of decarbonization with repeated annealings is a constant. (See Fig. 6.)

In all of the experiments referred to above, decarbonization has been effected by oxygen in the form of a gas. In order to determine if it is necessary for the oxygen to be in the form of gas to combine with the carbon in the steel, the following experiment was performed: A crucible of lead was saturated with lead oxide by the addition of litharge. A test piece was annealed in it for 3 hr. at 1,500° F. Microscopic examination revealed decarbonization at a depth of 0.024 in. This shows that easily reducible oxides such as $Pb_2O$, when dissolved in a molten bath, may also effect the elimination of carbon.

The conclusions to be drawn from these experiments are several:

It is shown that carbon is removed from the steel by a process of oxidation.

The oxidation is effected by either gases or liquids.

The principal active agents of the gases are air, $CO_2$, and water.

Of the liquids any good oxidizing agent dissolved in a liquid bath may be the active agent.

During the annealing the $CO_2$ necessary for the elimination of carbon may be formed through the reduction of iron oxide by CO.

No case has been observed where carbon was eliminated while the steel was heated below its critical point. This indicates that it is necessary for the carbon to be dissolved in the iron before decarbonization can take place.

It has been shown that the rate of decarbonization under uniform conditions is constant, without regard to the depth to which it has proceeded. From this we may infer that the carbon is transferred from the interior to the surface by diffusion through the solid solution. The oxidation takes place only at the surface.

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*Practical Applications*

The decarbonization of steel is of great practical interest to every user of tool-steel products. With but few exceptions, all tool steel furnished by the makers shows decarbonization to a greater or less degree. This is the result of various heating processes during manufacture. The decarbonized surface, or "bark," as it is commonly called, must be removed before the steel is used for cutting tools, wearing surfaces, etc. The depth of bark may vary greatly owing to oxidizing conditions during the hammering, rolling, and annealing of the rough material.

The most common method of inspection consists of hardening a piece of the material, breaking, and examining the hardened fracture by means of the naked eye or hand magnifier. The bark is thus distinguished by its fracture being coarser than the high-carbon interior. To examine the reliability of this method the following experiment was performed:

A bar of ½-in. round crucible steel was selected, having the same composition as the one used in the previous series of experiments. It was annealed and examined microscopically. The bark was found to be approximately 0.015 in. in depth. Pieces were cut from this bar and hardened at various temperatures. Microscopic examination was then made of a piece hardened at each temperature, the decarbonization being measured to the point where the normal structure of the interior began.

The depths of bark shown by the microscope on the various pieces were as follows:

| Piece No. | Temperature of Hardening, ° F. | Depth of Bark, Inches |
|---|---|---|
| 1 | 1,370 | 0.015 |
| 2 | 1,380 | 0.018 |
| 3 | 1,390 | 0.018 |
| 4 | 1,415 | 0.018 |
| 5 | 1,435 | 0.021 |
| 6 | 1,460 | 0.020 |
| 7 | 1,490 | 0.024 |
| 8 | 1,530 | 0.027 |

The increase in the amount of bark observed with the increase in temperature is of course due to the fact that an increased amount of cementite is held in solution by quenching from the higher temperatures.

The examination of the hardened fractures gave the following results.

No. 1 was only slightly hardened. The bark scarcely shows at all in the fracture.

No. 2 gives only a very slight indication of bark.

No. 3 no indication of bark.

No. 4 very slight indication of bark.

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No. 5 bark shows more distinctly but only in spots.

No. 6 bark shows fairly distinctly.

No. 7 bark very distinct.

No. 8 the fracture of the interior has become so coarse that there is no distinction between it and the bark.

This method of inspection is therefore reliable only when the hardening heats are carefully regulated and the results frequently checked by microscopic examination.

The method of inspection which the author has found to be the most satisfactory is by microscopic examination of an annealed specimen. The amount of carbon present and the exact condition of the bark can be determined very readily. Measurements of the depth of decarbonization may be made with a micrometer eye piece.

In the machining of tool steel, bark presents serious difficulties. It is a well-known fact among machinists that tool steel is frequently very difficult to machine near the surface. Also that after this hard bark is removed very little difficulty is encountered. This condition is almost invariably due to the surface decarbonization of the steel.

A typical example of this difficulty is found in turning twist-drill blanks from the round bar. Here it is sometimes observed that the outside of the bar, to a depth of several thousandths of an inch, is very soft. Beneath this is a ring about $\frac{1}{32}$ in. in thickness, so hard that it cuts a nick in the lathe tool. Below this ring the piece is uniformly soft.

Microscopic examination of such a piece reveals that the extreme outside of the bar consists of either ferrite or ferrite mixed with pearlite, which would of course machine as easily as machinery steel. The hard ring is found to consist of nearly pure pearlite in very coarse crystalline grains. The soft interior consists usually of finely divided pearlite and globular cementite, which also machines with great ease. Such a bark is shown in Fig. 7. The structure of the band of pure pearlite magnified 1,000 diameters is shown in Fig. 8.

In case a piece of tool steel is being machined in such a way that only a part of this layer of coarse pearlite is removed still greater difficulty is encountered. The surface of the work will be rough and the point of the tool will be worn out with great rapidity. The rapid wearing of the tool makes it impossible to hold a uniform size on any number of pieces of work.

The cause of this layer of coarse crystals is the difference between the critical points of eutectoid and hypereutectoid steel. At the correct temperature for annealing the hypereutectoid steel, the eutectoid is far enough above its critical point to allow the crystalline grains to grow to considerable size.

The fact that eutectoid steel with the pearlite arranged in coarse

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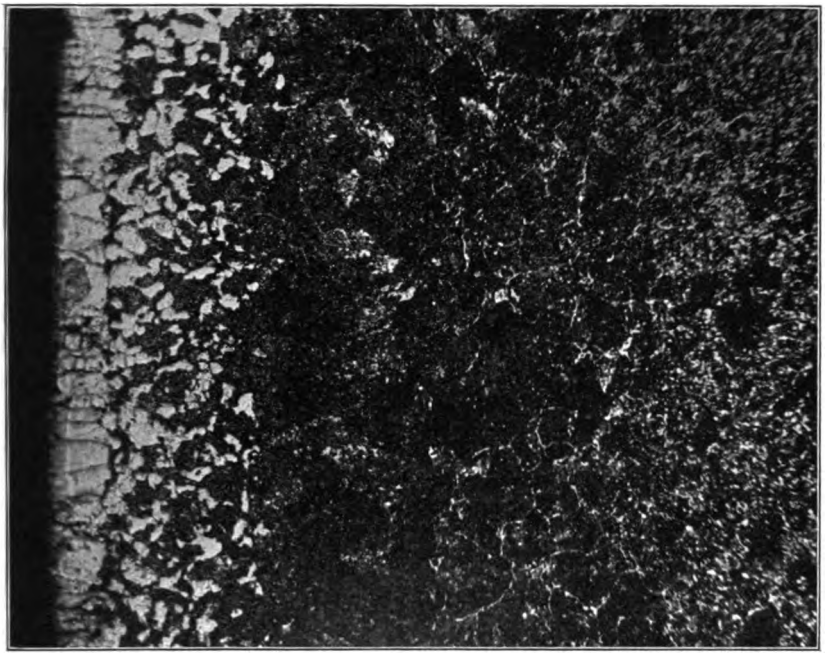

FIG. 7.—100 DIAMETERS.

FIG. 8.—1,000 DIAMETERS.

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grains is more difficult to machine than hypereutectoid steel in fine grains seems at first surprising. This is easily explained when we consider the manner in which the steel yields under stress and effect of the various micro-constituents upon the cutting tool.

It has been shown by several investigators that a tool when removing a chip from a piece of metal splits it off by an action similar to a wedge. In this case the hardest wear on the tool occurs not at the extreme point but a short distance back from the point where the pressure of the chip is greatest. Fig. 9 shows this condition on an exaggerated scale.

It will readily be seen that there is always a crack running ahead of the tool as it is forced through the material that is being cut. When the tool is cutting in coarse pearlite the crack will naturally follow the lines

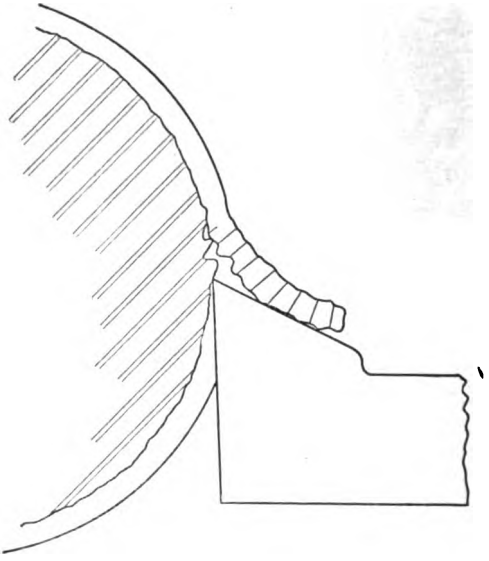

FIG. 9.—DIAGRAM ILLUSTRATING ACTION OF METAL-CUTTING TOOLS.

of least resistance. These are the boundary lines of the grains and through the grain in the direction of the laminations. This being the case, it follows that the larger the size of the crystalline grains and the more distinct the lamellar structure, the more irregular will be the crack and the rougher the resulting surface. Such a crack passing through a mass of coarsely crystalline pearlite is shown in Fig. 10.

The very rapid wear on the cutting tool engaged in cutting coarse pearlite is probably due to two causes:

First, the rough surface left by the crack running ahead of the tool greatly increases the amount of work and friction on the extreme cutting edge. The point of the tool is obliged to rub over many protruding grains

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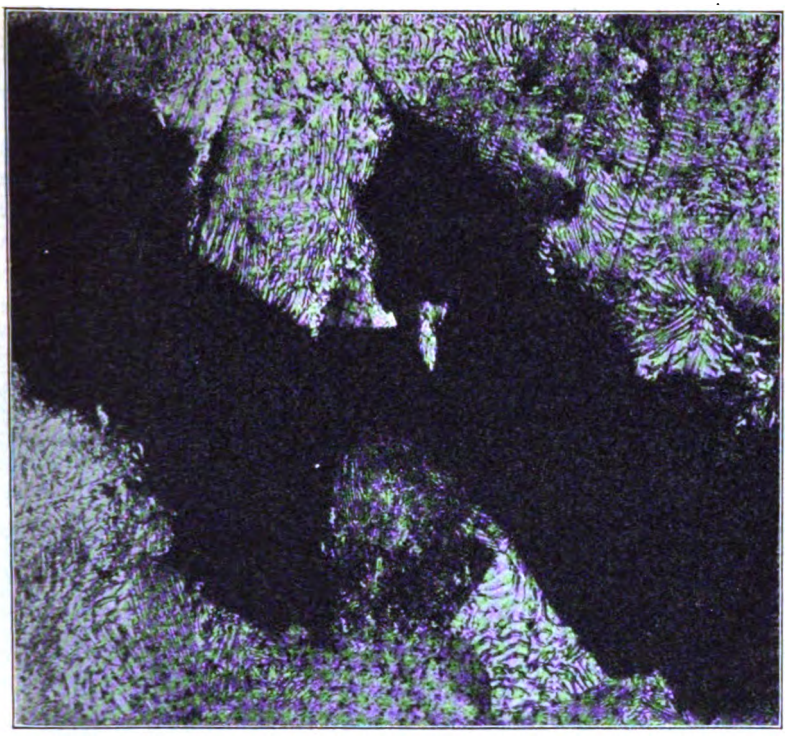

FIG. 10.—500 DIAMETERS.

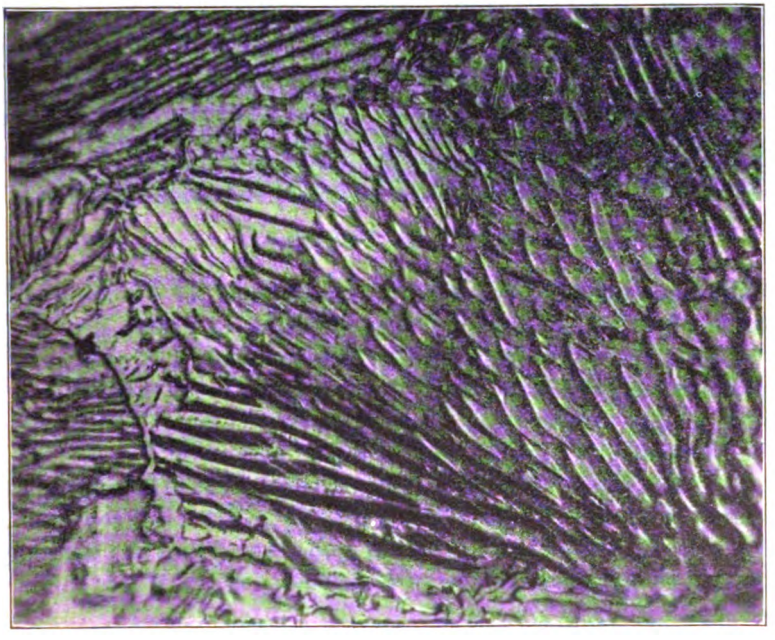

FIG. 11.—1,000 DIAMETERS.

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of pearlite and sometimes to work in and tear others out entirely. The result of this extra work is to round off the cutting edge rapidly and increase the friction, until the point is entirely destroyed by the heat generated.

The second cause of the extreme wear on a cutting tool working in coarse pearlite is the great abrasive quality of cementite when firmly held by the alternate plates of ferrite. The sharp angular plates of hard cementite in coarsely lamellar pearlite have a much greater abrasive action than the rounded masses of globular cementite in well-annealed hypereutectoid steel. A grain of coarsely lamellar pearlite exhibiting this feature is shown in Fig. 11.

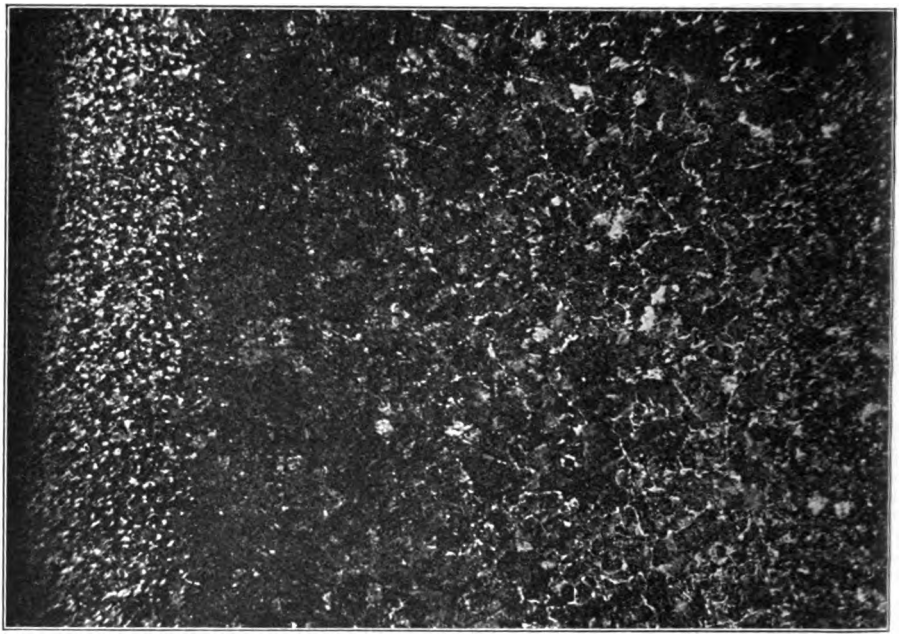

FIG. 12.—50 DIAMETERS.

A still more difficult form of bark to machine is one which has a band of coarse hypereutectoid steel below the band of coarse pearlite. In this case the coarse grains of pearlite are bounded by a cementite network, which seems to add greatly to the difficulty of machining. Fig. 12 shows a bark in which this phase is marked. At the edge of the bar is a band about 0.017 in. in width of mixed ferrite and pearlite in small grains. Below this is a band of pure pearlite about 0.030 in. in width, with large crystalline grains. Below this is a band of coarse-grained hypereutectoid steel about 0.030 in. in width, the grains of pearlite being surrounded by a coarse network of cementite. Below this is the normal

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structure of the interior, which is very fine-grained pearlite with small masses of globular cementite. In this piece of steel the band of ferrite mixed with pearlite and the hypereutectoid interior of the bar are very easy to machine. The bands of coarse pearlite and coarse pearlite surrounded by a cementite network are extremely difficult to machine.

If a piece of tool steel upon which a bark yet remains is hardened it is apparent that the surface will be hardened only in proportion to the carbon it contains. This may vary down to the point where the surface does not contain enough carbon to harden at all. In the great majority

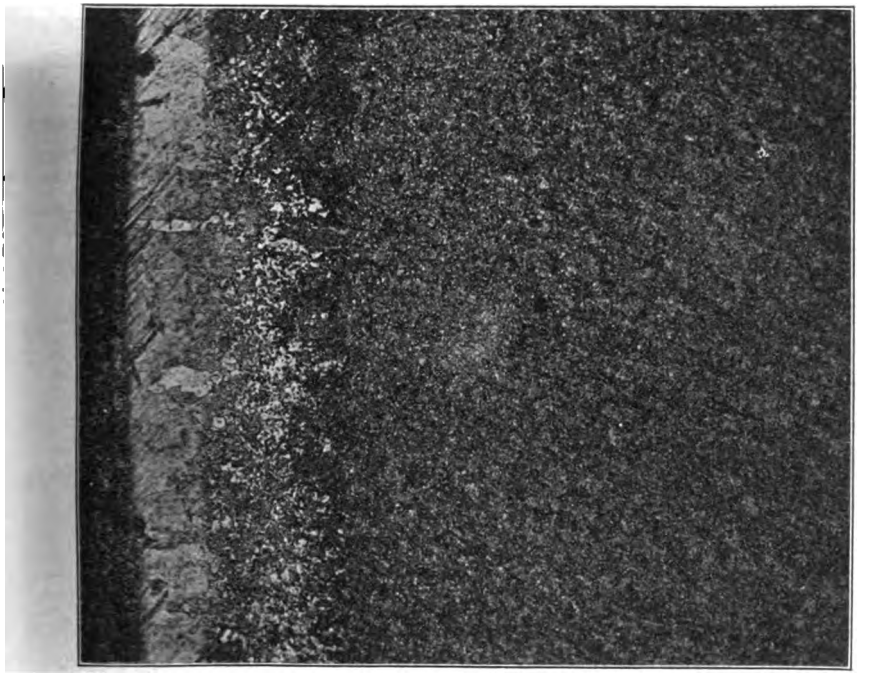

FIG. 13.—100 DIAMETERS.

of cases where tool steel is used for bearings, dies, and cutting tools the most severe wear must be borne by the surface or extreme edge. For instance, in the case of twist drills, a bark remaining on the tool would cause the outside point of the cutting edge, where the speed and wear are greatest, to be soft. This would result in the rapid wearing of the outside corner, causing the drill to run undersize and so increase the friction as soon to destroy the whole point of the drill. The effect of lowering the carbon content of the surface of a die or bearing even 10 or 20 points is very apparent.

Bark is also a frequent cause of checks and firecracks in hardened tools. One illustration of this difficulty is shown in Fig. 13, where there



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is a sharp line of demarcation between the bark and the structure of the interior. In this case, when the tool is quenched, a crack is liable to start and run along the boundary between the bark and the interior.

A second case in which bark left on a tool may start firecracks is where the coarse network of cementite, as shown in Fig. 12, is present. This cementite network is very liable to start a check or firecrack which will ruin the tool. The manner in which a cementite network starts hardening cracks has been more fully discussed in a previous paper. [4]

These remarks on the practical applications of a knowledge of the decarbonization of tool steel include but a few of the many difficulties which may be traced to this source. Each manufacturer and user of tool-steel products has of course his own difficulties with decarbonization, which can be solved only by research within his own shop.

In view of the effect of decarbonization upon the quality of tool-steel products it is surprising that so small an amount of attention has been paid it.

[4] *Iron Age*, vol. lxxxix, No. 8, p. 472 (Feb. 22, 1912).

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Salt Making by Solar Evaporation*

BY W. C. PHALEN, WASHINGTON, D. C.

(Pittsburg Meeting, October, 1914)

## Salt-Making Processes

THE production of salt in the United States divides itself at the outset into two distinct classes: (1) The mining of rock salt and its purification and separation into marketable sizes, and (2) the production of salt by evaporation from brines, bitterns, and other solutions.

The processes employed at the present time in the manufacture of salt by evaporation may be outlined as follows:

1. Solar evaporation.
2. Direct-heat evaporation (*a*) in open kettles; (*b*) in open pans.
3. Steam evaporation, (*a*) in jacketed kettles, (*b*) in grainers.
4. Vacuum-pan evaporation.

In this paper only the solar-evaporation processes practised in different parts of the United States will be described.

## Solar Evaporation

### *Localities*

In the Eastern States the solar-evaporation process is not generally employed for climatic reasons. In the Western States, particularly in California and Utah, the great bulk of the salt produced originates in the solar-evaporation process. Of the Eastern States producing salt, this process, so far as the writer is aware, is employed in New York State only.

## New York[1]

### *Introduction*

The manufacture of salt by solar evaporation began in the vicinity of Syracuse, Onondaga county, in 1789. A natural brine 68° to 80°

*Published with the permission of the Director of the U. S. Geological Survey.

[1] In the preparation of this portion of the chapter on solar evaporation, the author has made free use of the works of F. E. Englehardt; *Bulletin No.* 11, *N. Y. State Museum*, pp. 38 to 44 (1893), and T. M. Chatard; Salt-making Processes in the United States, *7th Annual Report, U. S. Geological Survey*, pp. 506, 507, (1888).

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salometer, with 17 to 20 per cent. sodium chloride, was and is still used. The brine is stored in glacial gravels and was evidently formed by the circulation of ground waters through adjacent beds of rock salt.

But a small part of the evaporated salt manufactured in New York is now made by the solar process. Its manufacture is limited to Syracuse and vicinity, where it has survived from the early days of the industry. Though it has now lost some of its former importance, up to the year 1880, the time when the beds of rock salt in the western part of New York State began to be utilized, Onondaga county supplied the whole output of the State in rock salt. A large part of the total now returned for the county represents the salt in brine consumed in the manufacture of soda. The solar salt produced is mainly coarse salt and is used for practically the same purposes as rock salt. It is marketed through the Onondaga Coarse Salt Association.

The wells are located on lands once included within the Onondaga Indian reservation; and until recently the State supplied the brine to the individual plants, exacting a small tax on the product to cover costs of pumping and supervision. The lands and wells were sold in 1908 to private companies, namely, the Onondaga Pipe Line Co. and the Mutual Pipe Line Co. of Syracuse, the transfer being made for a nominal sum. This transfer definitely terminated the historic connection of the State with the salt business.

Solar salt is made from a natural brine—the only instance of its use in New York. The process, of course, is carried on only during the spring, summer, and fall months, usually from the middle of March to the middle of November, depending on the weather.

### *Method of Manufacture*

*Former Method.*—The manufacture of solar salt in New York was formerly carried on in shallow wooden vats or "covers," provided with light movable roofs running on rollers. During the evaporation season the roofs were removed to one side, but replaced during rainy weather, and at the end of the salt-making season. There were ordinarily three sets of vats or "rooms," known respectively as "deep rooms," "lime rooms," and "salt rooms."

The deep rooms served for the reception of the brine from the well or pump house. When first pumped the brine was usually perfectly clear, but with the escape of gases ($CO_2$), ferrous carbonate separated, which oxidized to the hydrated ferric oxides, and settled out as a yellow mud, leaving the clear brine.

The lime rooms were constructed nearer the ground than the deep rooms, the clear brine, being drawn from the higher deep rooms into the lower or lime rooms. Evaporation continued in the latter until the brine became saturated and salt crystals began to separate. During this

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process, particularly when the brine was near the point of complete saturation, gypsum began to separate. The brine, now fully saturated and known as "pickle," went to the lowest set of rooms, known as the salt rooms. Here the salt and the remainder of the gypsum separated, sufficient pickle being added from the lime room from time to time to keep the salt crystals well covered. The term "lime room" in the above descriptions is somewhat misleading as no lime was used in the process. When sufficient salt collected in the salt rooms it was harvested. This took place generally three times during a season.

*Present Method.*—The process described above was the orginal method of solar evaporation. Great improvements have been made in it, one of the greatest being the replacement of the deep rooms and lime rooms, by "aprons," whereby the evaporating surface and the yield of salt per cover (288 sq. ft.), have been greatly increased. The aprons are large troughs from 15 to 20 ft. wide, about 3 in. deep, and varying greatly in length. They are generally erected over the deep rooms serving as roofs, thus making store rooms for the saturated brines of the deep rooms. They are built on piles or posts, and are so constructed that brine or rain water falling on them flows toward two sets of holes provided with wooden plugs, the grade of flowage being 1 in. per 100 ft. One set of holes communicates with the deep rooms. The brine run on to the aprons often becomes saturated in one good drying day. It is then discharged into the deep room below and its place taken by a fresh portion of brine. When rain is expected the plugs over the cisterns, or deep rooms, are drawn and the brine which flows into them is thus saved from dilution. The plugs are then replaced and the other set removed, thus allowing the rain water to flow to waste. When clear weather returns, the partially concentrated brine is returned to the aprons by pumps.

"The great advantage of this improvement lies in this—that the brine is concentrated and purified while being transported to the covers; the great length and width of the aprons, the shallowness of the layer of brine and its complete exposure to the sun, air, and wind greatly facilitate purification and evaporation while permitting the use of the entire number of covers for salt making, instead of having as formerly one-third of the total number taken up for lime rooms and deep rooms." In this way the production of solar salt during a given season has been greatly increased per cover. As the capital invested in covers is considerable, the economy of the process is readily seen. (Figs. 1 and 2.)

## UTAH

### *The Water of Great Salt Lake*

Salt is obtained from the water of Great Salt Lake. According to different authorities who have analyzed the lake water, the chief differ-

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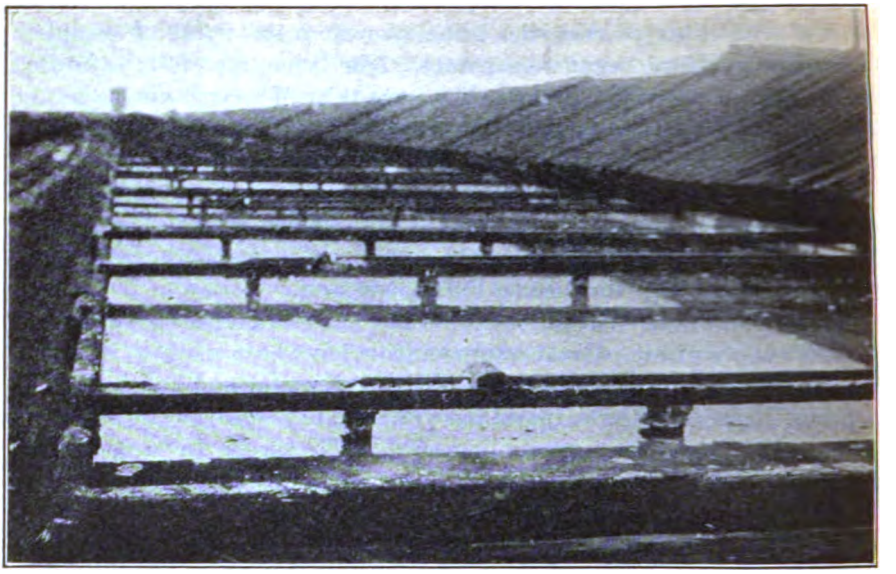

FIG. 1.—Yard of Salina Salt Co., Syracuse, N. Y.

FIG. 2.—Loading Wagons, Salina Coarse Salt Co., Syracuse, N. Y.

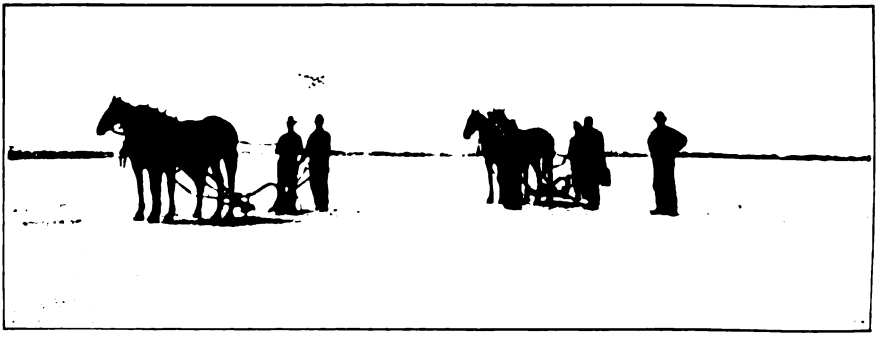

FIG. 3.—Harvesting Salt, Inland Crystal Salt Co., Saltair, Utah.

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ences in the results are in the degree, but not the character, of the salinity. It is to be expected that the former would change with greater or less dilution, as the waters of the lake rise or fall, from local or other causes. Although the degree of salinity of the lake is variable, ranging approximately from four to seven times that of ocean water, the composition of the saline matter contained in it is much like that found in ocean water. The following analyses show the composition of the salts contained in the water:

*Analyses of Salts in Water from Great Salt Lake*[2]

| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 |
|---|---|---|---|---|---|---|---|---|
| Cl | 55.99 | 56.21 | 55.57 | 56.54 | 55.69 | 55.25 | 55.11 | 53.72 |
| Br | trace | ..... | ..... | ..... | trace | trace | ..... | ..... |
| $SO_4$ | 6.57 | 6.89 | 6.86 | 5.97 | 6.52 | 6.73 | 6.66 | 5.95 |
| $CO_3$ | ..... | 0.07 | ..... | ..... | ..... | ..... | ..... | ..... |
| Li | trace | ..... | ..... | ..... | 0.01 | trace | ..... | ..... |
| Na | 33.15 | 33.45 | 33.17 | 33.39 | 32.92 | 34.65 | 32.97 | 32.81 |
| K | 1.60 | (?) | 1.59 | 1.08 | 1.70 | 2.64 | 3.13 | 4.99 |
| Ca | 0.17 | 0.20 | 0.21 | 0.42 | 1.05 | 0.16 | 0.17 | 0.31 |
| Mg | 2.52 | 3.18 | 2.60 | 2.60 | 2.10 | 0.57 | 1.96 | 2.22 |
| $Fe_2O_3$, $Al_2O_3$, $SiO_2$ | ..... | ..... | ..... | ..... | .0.01 | ..... | ..... | ..... |
| | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 | 100.00 |
| Salinity, per cent | 14.994 | 13.790 | 15.671 | 19.558 | [a]23.036 | 27.72 | 22.99 | 17.68 |

[a] More correctly, 230.355 g. per liter.

Compared with the analyses of sea water, the sodium is higher, and the lime and magnesia, especially the former, are distinctly lower. Carbonates are nearly entirely absent, but are present to an appreciable extent in ocean water. There are other variations in composition between lake and ocean water, but in general they are of the same type. Additional analyses appear in a monograph by Gilbert.[3] Gilbert states (p. 253) that the quantity of sodium chloride contained in the waters of the lake is about 400,000,000 tons and that the amount of sodium sulphate is about 30,000,000 tons. In the same volume Gilbert also considers the different surveys made of the lake, the depth of water, the oscillations of its surface, the changes and the causes of change in its area, the sources of its saline matter, together with the rate and period of its accumulation.

## *The Salt Industry*

Salt has been made by the solar-evaporation process for many years along the shores of Great Salt Lake, the industry dating in fact from the early settlement of the region. A small amount of salt is reported as

[2] Taken from F. W. Clarke: Data of Geochemistry; *Bulletin* 491, *U. S. Geological Survey*, p. 144.

[3] Gilbert, G. K.: Lake Bonneville, *Monograph*, *U. S. Geological Survey*, vol. i, pp. 253 to 254 (1890).

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having been made recently at Promontory Point, Box Elder county; Withee, Weber county; Syracuse, Davis county; and at Garfield, Salt Lake county. Fifteen miles west of Salt Lake City, near Saltair, is the plant of the Inland Crystal Salt Co. The operations at the latter place are by far the most elaborate ever attempted along the lake shore. The process of obtaining salt near Saltair is described below.

The brine is pumped from the lake into a flume in which it is carried about 3 miles to the ponds, where concentration and crystallization are effected. The ponds are designated from the processes taking place in them as: Settling ponds, stock or evaporating ponds, and crystallizing or harvesting ponds. The ponds are separated from each other by clay embankments held in place by boards set on edge. These clay levees are 2 ft. wide and 22 in. high. Pumping begins in April and continues until the first of September, excepting during times of storm. Evaporation goes on approximately at the same rate as pumping, which is about 5,000 gal. per minute, averaging 16 hr. per day from the middle of June to the middle of September—the season of greatest dryness and hence of maximum evaporation. Pumping is regulated so as to maintain the level of the brine in the ponds.

The brine from the lake goes first to the settling ponds, where it is allowed to remain five to six days. Here all suspended matter is deposited. The settling ponds are estimated to be about 75 acres in extent. From the settling ponds it goes to the stock ponds, which cover an area of about 1 sq. mile. Here it remains until concentration reaches complete saturation and salt is ready to deposit, the length of time depending upon the weather. The brine then goes to the harvesting ponds. During the summer of 1912, when the writer visited the region, 30 days elapsed between the beginning of pumping operations and the time the brine reached the harvesting ponds. The brine is conducted from one pond to the other by gravity, the flow being controlled by means of small gates.

In the harvesting or crystallizing ponds, as the name suggests, evaporation is carried to the point where all salt separates out. To insure a clean product, an underlayer, or salt floor, is allowed to crystallize out each year, unless it is left over from the operations of the preceding year. This underlayer serves as a clean floor upon which the subsequent crystallization takes place. When it remains from the previous year, as happens when the salt crop is larger than the market demands, the salt in it becomes more or less dirty. It is then used as stock salt though it is a first-class product in every respect. The salt which is marketed comes from the upper layer, the plane of demarcation between it and the floor being known as the "split." The average crop is about 3 in. of salt. The minimum crop is about 2 in.— the maximum usually 4 in., but a crop of nearly 5 in. has been obtained. Three inches is considered a good crop.

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During the course of the salt-making season the bittern formed in the harvesting ponds is drawn off twice, once during the middle of the season, and again at its end. In this way the salt is freed from the bulk of magnesium salts and sodium sulphate. Care must be exercised to draw off the bittern toward the season's end, before cold weather begins, to insure against precipitation of sodium sulphate, which crystallizes out when the water reaches a temperature of 20° F.[4]

When the bittern is drawn off for the last time and the salt is ready to be lifted, it is loosened by means of ordinary plows drawn by horses. (Fig. 3.) It is then stacked by means of scrapers, wheelbarrows, or hand cars run on tracks into large piles alongside the railroad which traverses the salt field. The salt is conveyed from the stacks to the cars, which are transported in turn by means of an oil-burning locomotive to the company's mill, where the salt is further refined A portion of the yield may be sold without further treatment, for the different purposes for which rock salt is used.

## California

*Localities.*—More than 97 per cent. of the output of salt in California originates from the evaporation of sea water along the coast. The main

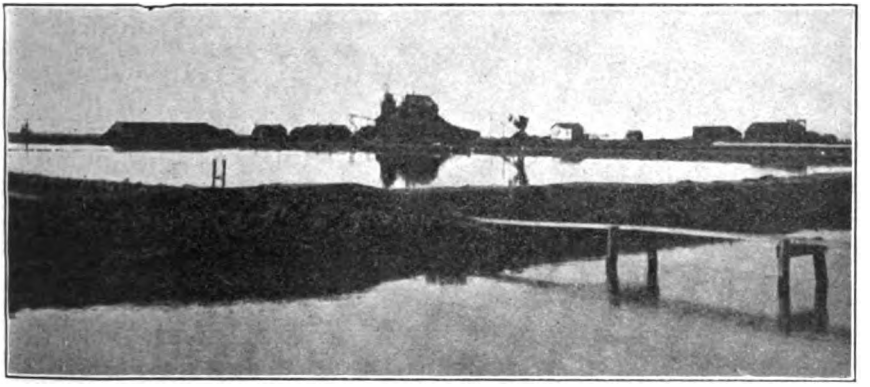

Fig. 4.—View of Salt Ponds. Oliver Salt Co., Mount Eden, Cal.

production of this solar salt comes from along the east and west shores of San Fancisco bay, in Alameda county, and San Mateo county, respectively. In Alameda county the centers of production are near Alvarado, Mt. Eden, Russell, and Newark; in San Mateo county, solar salt is produced a short distance south of San Mateo and near Redwood City. Other places where salt is produced by solar evaporation are near Long

[4] Gilbert, G. K: *Loc. cit.*, p. 253. Talmage believes the critical temperature of the separation to be within a few days of the freezing point of fresh water, say approximately, at 35° F.

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Beach, Los Angeles county, and on San Diego bay, San Diego county. In all these places the operations are conducted along similar lines, but they differ considerably from each other in the details worked out by individual operators in charge. There is also considerable diversity in nomenclature connected with the processes in their various stages. In the following account it is aimed to give general descriptions of processes with certain details observed at representative plants in the different salt-making localities.

*Salt-Making Season.*—The concentration of the water in San Francisco bay is variable, depending on the season of the year and other factors.

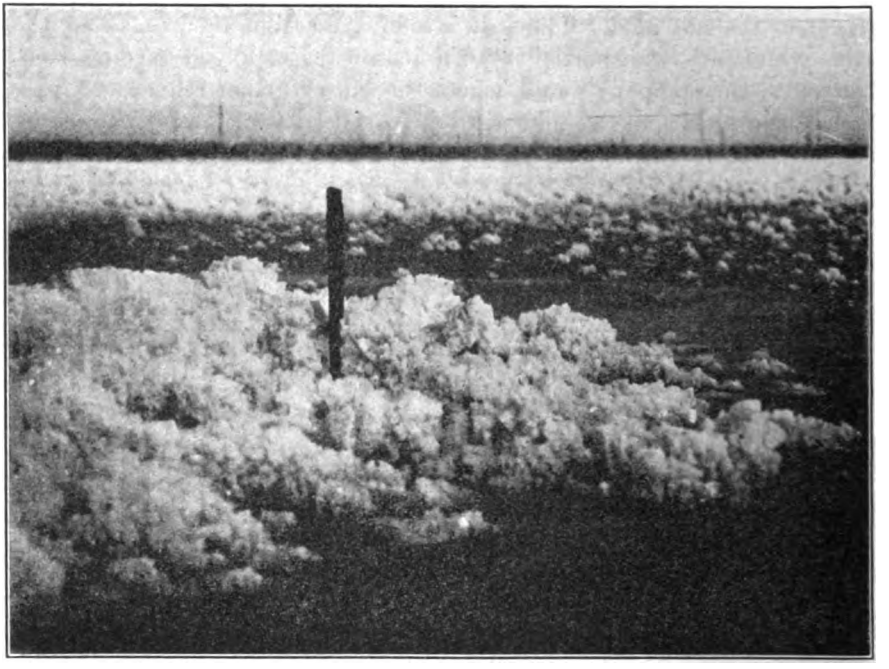

FIG. 5.—SALT CRYSTALLIZED IN A CRYSTALLIZING POND. LESLIE SALT CO., SAN MATEO, CAL.

The dilution of the bay water is caused in part by the fresh water from the San Joaquin and Sacramento rivers and is increased after winters of heavy snow which is not melted away completely until the summer is well advanced. Brine is taken into the salt ponds from about the middle of May to the middle of October, but the strongest brines are obtained after July first, as the dry season lengthens and the inflow of fresh water from the rivers and streams is appreciably reduced. Generally the salt-making season is over by September 15, as the rainless season lasts usually from May to September. A season as long as 210 days without rain has been known, however, along San Francisco bay.

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The season for 1911 may be taken as a typical one. During the summer of that year the evaporation and rainfall as observed at one plant were as follows:

*Rainfall and Evaporation in Inches, Season, 1911*

| | Rainfall, inches | Evaporation, inches |
|---|---|---|
| April | 1.36 | 3⅜ |
| May | 1.14 | 5$\frac{5}{16}$ |
| June | 0.67 | 6$\frac{9}{16}$ |
| July | none | 7$\frac{11}{16}$ |
| August | none | 7$\frac{13}{16}$ |
| September | none | 4$\frac{15}{16}$ |
| October | 0.77 | 2$\frac{15}{16}$ |

The water from the bay or ocean is not taken into the works continuously, but usually on two to six days each month, depending on the season when the tides are at their highest. Water is usually taken in at

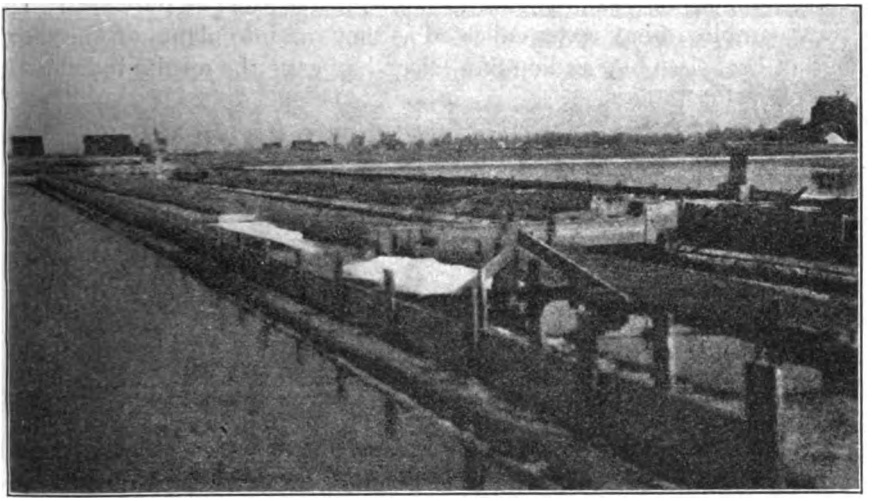

FIG. 6.—GRAVITY METHOD OF RUNNING BRINE FROM ONE POND TO ANOTHER. OLIVER SALT CO., MOUNT EDEN, CAL.

the period of the new moon. Sometimes the high tides come twice a month, but once a month is the more usual. The periods of highest tides are ascertained from the tide book issued by the U. S. Coast and Geodetic Survey. During the periods of high tide, water is not running into a salt works continuously, but only from 1½ to 3 hr. during a given high tide. These are statements based on general practice, but it is understood that at some plants, where a slough runs through the intake pond, salt water may be taken into a given system at any time or every day. Thus, at Long Beach the sea water runs into a ditch at every high

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tide, that is, twice each day. Salt water may therefore be pumped continuously day or night from this ditch into the plant.

In southern California, naturally, the salt-making season is longer than it is near San Francisco. At San Diego it is estimated to be about seven months, namely, from May to November. On San Diego bay the salt water also may be taken in continuously during the salt-making season. This is accomplished by draining the tide ponds, whereby water may be taken in on every tide above a given height.

From 1908 to 1912, the mean average date when crystallization of salt began along San Francisco bay, was as follows: 1908, May 1; 1909, May 13; 1910, May 3; 1911, May 25; 1912, April 18. The reason for the early date in 1912 was on account of the very few late rains during that year. Records are closed, that is, evaporation readings are stopped, each November 1, when the days are so short and the weather so cold and uncertain that there is practically no evaporation.

*Strength of Brine Used.*—The average density of sea water is 1.027, corresponding to a salinity of 3.72 per cent. regarded as pure salt. The three samples of bay water collected as they ran into plants on the shores of San Francisco bay and on San Diego bay gave the results indicated in the following table:

*Density and Salinity of Water used in Salt-Making Process on the California Coast*

| No. | Baumé degrees | Specific gravity[a] | Percentage of salt[b] | |
|---|---|---|---|---|
| 23 | 3.53 | 1.025 | 3.44 | Water from San Francisco bay, Mt. Eden. |
| 28 | 3.12 | 1.022 | 3.34 | Water from San Francisco bay, Alvarado. |
| 33 | 4.10 | 1.029 | 4.00 | Water from San Diego bay. |
| .. | .... | 1.027 | 3.72 | Sea water. |

[a] Specific gravities were taken at 22° C., but are referred to water at 15° C.
[b] Assumed to be pure NaCl, which is not the case.

Thus it will be observed that the normal salt water entering the salt plants along San Francisco bay is less concentrated than ordinary sea water, while the opposite is true of the bay water near San Diego. These samples of bay water were collected: No. 23, Oct. 5; No. 28, Oct. 8; No. 33, Oct. 17; *i.e.*, on days well advanced during the dry, or salt-making season. One of the reasons for the difference in density between the waters from San Francisco bay, and San Diego bay, is that farther south the warmer and dryer season lasts much longer. The evaporation therefore is more intense, hence at a given date during the dry season the southern waters would naturally be expected to show a higher degree of salinity. Other conditions which tend to reduce the salinity of the San Francisco bay water are the streams of fresh water which flow into it.

*Ponds.*—The salt-making operations are conducted in what are known as ponds, designated from the operations conducted in them. Thus

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there are: (1) The storage, the intake, receiving or tide ponds into which the salt water is received from the bay, (2) the concentrating ponds, and (3) the crystallizing ponds. By some operators all the ponds between the tide ponds and the crystallizing ponds are known as secondary ponds. Technically they may be known as pickling ponds. The term "lime ponds" is sometimes applied to those in which the bulk of the gypsum crystallizes out.

The salt or ocean water comes into the works generally through a slough, passing thence into the intake, receiving, storage, or tide pond, as the first pond is variously called. This pond is provided with large flood gates with automatic adjustments to open them to allow the water to run in, but which close when the tide begins to ebb. The salt water remains in the intake pond a variable length of time, in the meanwhile evaporating. It is pumped out before the next salt water is taken in. It then goes forward through the different ponds becoming more concentrated, until it finally reaches the crystallizing ponds. It is run into these latter when it reaches a strength of about 25° Bé, or when salt begins to form, usually to a depth of about 6 in. At some plants, when it reaches a strength of 29° Bé, the bittern with some salt still in it runs over into other ponds. Here it evaporates until a concentration of 32° Bé, is reached, when all the salt is out of it (see table below). The mother liquor remaining is then run into the bay and goes to waste at many plants, but it seems that some use should be made of the magnesia and potash salts contained in it. At some plants a part of it at least is saved and utilized as described later in this paper.

*Specific Gravity of Bitterns from Crystallizing Ponds*

| No. of Sample | Baumé reading at 22° C. | Specific gravity* | Locality |
|---|---|---|---|
| 25 | 32.50 | 1.286 | Oliver plant, Mt. Eden, Cal. |
| 26 | 29.60 | 1.256 | Leslie Salt Co., San Mateo, Cal. |
| 30 | 27.21 | 1.231 | California Salt Co., Alvarado, Cal. |
| 31 | 28.00 | 1.239 | Long Beach Salt Co., Long Beach, Cal. |
| 32 | 28.60 | 1.245 | Long Beach Salt Co., Long Beach, Cal. |
| 34 | 27.50 | 1.234 | Western Salt Co., San Diego Bay. |

* Specific gravities were taken at 22° C., but are referred to water at 15° C. They were made by R. K. Bailey, of the U. S. Geological Survey, with a Westphal balance.

Area is one of the dominant factors in the industry. There are plants where the total acreage of ponds is as low as 500 acres, but others are reported to have more than 2,000. The ratio of the acreage of crystallizing ponds to that of the whole system was found at two important plants to be as 1 to 10, but the older the plant, the more acreage it has for crys-

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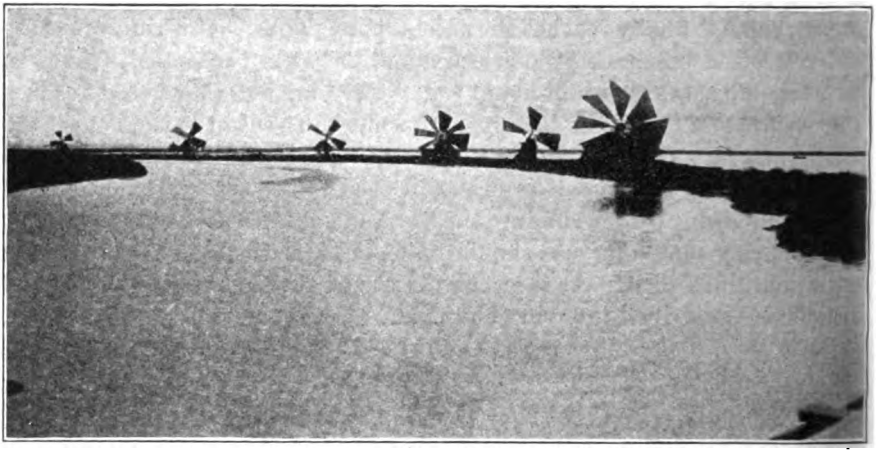
tallizing purposes, the reason being that the ground becomes thoroughly salted down with continued usage.

The ponds have mud floors. The salt in places is harvested to within 2 in. of the mud floors and removed to the main stock piles. The remaining 2 in .is shovelled up into smaller piles and this grade is used as ice cream and stock salt.

The ponds are separated from each other by low embankments known as dikes. The width and height of these dikes vary greatly. They may be as much as 3 ft. above the marsh level and 1 or 2 ft. above brine level. Some are very narrow, others, especially some of the outside dikes, are wide enough and sufficiently well constructed to serve as roadways over which teams and autos may be driven. The low narrow dikes or levees are not considered by all the salt makers as good practice as they require constant attention and repair owing to wash by the waves. Where the



'FIG. 7.—SLOUGH AND WIND MILLS. OLIVER SALT CO., MOUNT EDEN, CAL.

levees are too high the evaporating action of the wind is interfered with, but at San Diego there is not much wind. The dikes between the ponds were formerly flanked with sod, but this practice is now being done away with.

*Methods of Pumping.*—Pumping along San Francisco bay is done in part by Archimedes screws propelled by wind mills; also by means of paddle wheels. The average plant may have as many as 40 wind mills. Some plants use gasoline engines as an auxiliary power, especially in the fall when the trade winds die down. In August, the trade winds slacken or blow intermittently and therefore cannot be depended upon. The wind mills are set solid to the northwest from which the trade winds come and from their manner of construction they cannot turn around.

The wind mills are generally placed so as to pump from the outer

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or intake pond into the next pond. From the latter it is sought to have the brine move along by gravity, the flow from one pond to the next being controlled by small gates. Wind mills or other means may, however, be used for lifting in other parts of the system, when gravity is found to be inexpedient.

*Harvesting.*—The crop of salt of the preceding season is usually sold by July 1 of the following season. Enough salt is then lifted from time to time to supply the current demand until September 1, when harvesting is pushed as vigorously as possible. At many plants the first salt is lifted

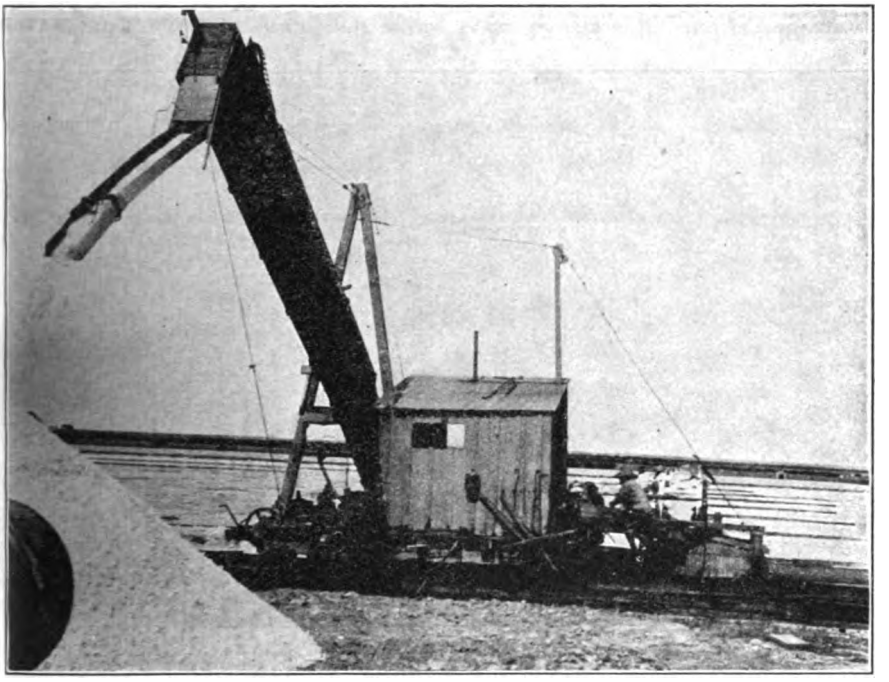

FIG. 8.—METHOD OF LIFTING SALT IN HARVESTING POND. CALIFORNIA SALT CO., ALVARADO, CAL.

about June 1, and the operation continued until about the first of January or February of the following year if the crop is a big one and labor is rather scarce. Ordinarily, however, lifting is finished by December. The salt is lifted by hand into small tram cars each holding usually about 1 ton. The salt thus lifted is pinkish in hue and the mother liquors are likewise pinkish. A small locomotive propelled by gasoline or electric power picks up eight cars, more or less, and hauls them to the mill; or they may be hauled to the salt piles or stacks and dumped. There may be one huge stack near the mill itself from which salt is used according to the demand. The latter method perhaps may

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be the more economical if practicable, for it reduces the surface exposed to solution from rains and also the surface which has to be broken up in the subsequent utilization of the salt. This latter is an important consideration, for the surface hardens so firmly that it is broken up only with the greatest difficulty; and in some cases the cost of breaking up a particularly old crust on a small stack costs more than the salt in the stack is worth. Sometimes old and small stacks may be discarded completely. In some places after standing in the stock piles the salt hardens so firmly that crosscut saws have to be used. These saws cut it into 4-ft. lengths which are pried apart and then crushed in the subsequent milling operations.

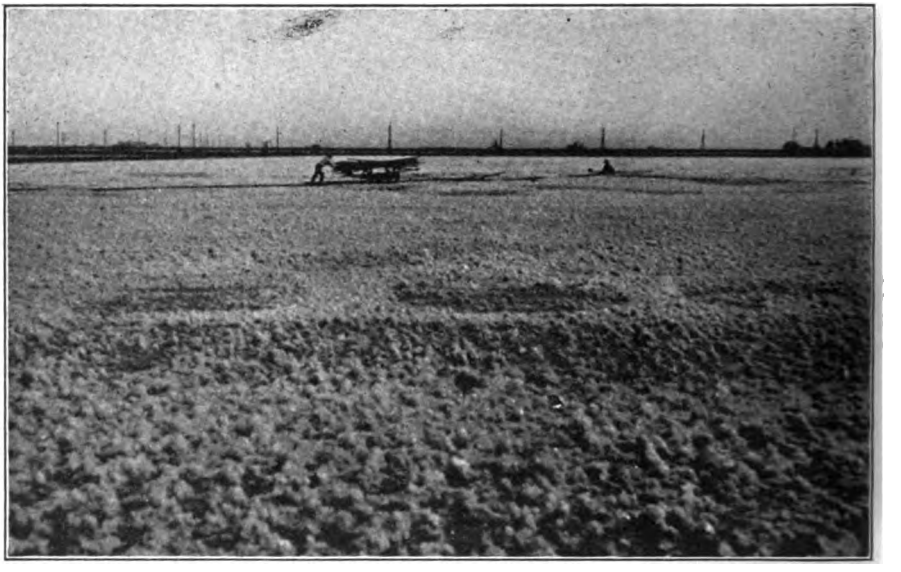

FIG. 9.—GENERAL VIEW OF CRYSTALLIZING POND AND THE LAYING OF TRACK. LESLIE SALT CO., SAN MATEO, CAL.

When the salt harvested is not taken directly to the mills, but to the stacks, it may be manipulated in various ways as outlined below. After reaching the works it may be dumped through a hopper on to a screw conveyor, passing thence to a bucket elevator, which lifts it to the main salt stack. Not every plant can use this method, because the ground has to be very firm to accommodate the enormous tonnage of one main stack, a condition not easily obtained in regions of marshy land. Sinking of the ground may occur and an appreciable part of the product be lost. An instance is known where 1 ft. of filling sank each year for a period of 20 years.

Instead of shovelling the salt into small cars to be hauled to the stacks, dumped and there elevated, a much simpler and comparatively inexpensive

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process may be used, in which the salt is shovelled by laborers, a line of whom extends the width of a pond, on to an endless belt 20 in. wide, running on rollers, which conveys the salt to one side of the pond. The power is furnished by a gasoline engine, but an electric motor might serve as well, or better. As the salt leaves the belt it is washed by a spray of salt water from the bay. It is next dragged up a perforated copper screen, during the ascent of which it is again washed with bay water. The endless drag chain is provided with forks to break up the larger lumps. The salt is then elevated to the stacks by a pan elevator run so that the open mouths of the pans are reversed to enable the excess of wash water to drain. By a washing device the main belt which conveys the salt from the pond is kept perfectly clean.

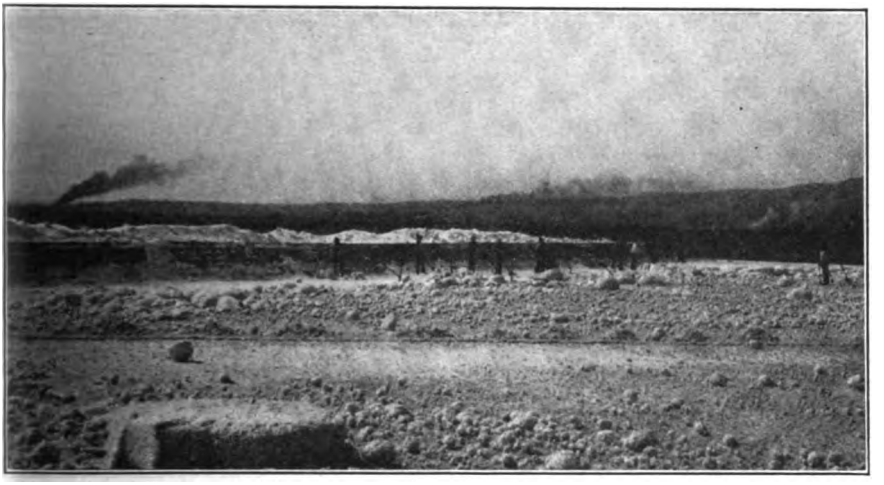

FIG. 10.—VIEW OF SALT POND IN WHICH THE SALT IS BEING HARVESTED. LONG BEACH SALT CO., LONG BEACH, CAL.

The method just described has many labor-saving advantages. As soon as a given area of salt has been removed from the crystallizing ponds the entire layout is moved forward on tracks provided for the purpose, and the salt pile may thus stretch out the length of the pond. The washing of the salt likewise is performed at a time when the adhering bittern is most readily and completely removed. The salt from the outside stock piles is brought up to the mill as needed in 5-ton cars and there subjected to the various operations such as crushing, etc., to fit it for the finer uses, or it is re-dissolved and prepared for the vacuum pans.

At Long Beach the salt is shovelled from the stock pile into small cars which are trammed by a rope tram to a hopper through which it passes, being ground in the process. It is next elevated by a bucket conveyor and dumped into a bin, which in this case is an ordinary freight



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car fitted up for the purpose. When the bin is filled the car is motored to the mill.

At San Diego the salt has to be picked as it lies in the crystallizing pond before it can be lifted with shovels. The greater firmness of the crystallized salt as compared with that along San Francisco bay, which makes picking necessary, is possibly due to the greater rapidity of crystallization. It is customary to begin harvesting just as soon as the bittern is run off the crystallizing ponds. This is due to the fact that the salt is easier to pick, shovel, and wash at this stage than at any other.

It is difficult to give the amount of salt which can be made during a given season, as the quantity is variable from year to year, depending upon the weather and other factors. Perhaps a thickness of 5 or 6 in. of salt per season would be a fair approximation for the region along San

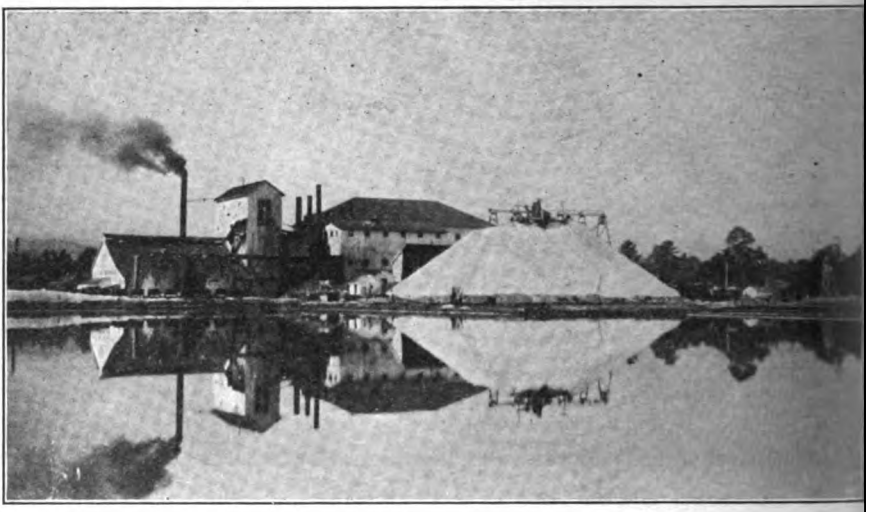

FIG. 11.—SALT PILE AND PLANT. LESLIE SALT CO., SAN MATEO, CAL.

Francisco bay, but in the southern part of the State owing to the longer and dryer season, it would be very much larger. At San Diego, for example, an average harvest is 6 in. of salt, but two such harvests are possible per season, in this region. Hence it is possible to make 12 in. of salt per year in this locality.

*Waste Bittern.*—The bittern from the crystallizing ponds has never been saved at many of the plants in the past and at most of them it is even now running to waste. Since the importance of potash salts has come to be realized some of the operators have in mind the utilization of the potash content of the bitterns, but at most of them nothing definite with regard to such use has yet been evolved. At one plant a small part of the bittern is saved, refined and used for medicinal purposes; at an-

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ther it has been used in the manufacture of "wood stone" and in the manufacture of magnesium oxychloride cement. The latter substance has been found very resistant to the passage of electricity, and hence has been applied in the manufacture of switch boards. It is understood that the bittern has been used by the Santa Fe Railroad to lay the dust along its road bed, but experience has proved it to be too costly for this purpose, as it has to be renewed too frequently.

*Milling.*—From the nature of the processes employed in the manufacture of solar salt, and more particularly from the differences between them and those employed in other parts of the United States, the first stages of the milling operations are different in California from the practice elsewhere. When the salt is lifted from the crystallizing pond, it is contaminated with considerable adhering pickle or bittern and dirt of various kinds. If it is not to be used at once it may undergo a preliminary washing after which it is conveyed to the stacks as already indicated under harvesting. If it is to be used immediately it is transported to the mill and there washed free from adhering impurities. This is accomplished in various ways. It may be dumped into a hopper and a current of hot brine from the pickling pond poured over it. This brine from the pickling pond represents bay water concentrated to complete saturation and from which no salt has been crystallized. For a concentrated solution it contains the least amount of mother-liquor salts. Its use results in the solution of a minimum amount of salt during the washing process. From the hopper the salt is lifted in an elevator, draining in the meanwhile. It is then passed through rolls where it is crushed into "half-ground salt" or "three-quarter ground salt." It then goes forward to other vats, where there is an artificial brine made from salt and fresh water and in which consequently there are no mother-liquor salts. The salt on removal from these vats is stacked in heaps to drain. The coarse salt is then sacked for one branch of the trade. If it is to be subjected to further refining processes the salt goes from the vats to a centrifugal machine where adhering water is removed. It is then conveyed to dryers where it is thoroughly dried by steam heat. The dryers are elongated cylindrical affairs containing steam coils. They are provided with fans to pump warm air through them, which greatly assists in the drying operation. From the dryers the salt goes directly to the rolls where it is crushed. From the rolls it goes to the sifters where it is graded according to fineness, depending on the purposes for which it is to be used. It is then ready to be sacked for the trade. At different plants the operations differ greatly in details to suit individual tastes and those of the customers.

At some of the plants the solar salt in the stock piles is re-dissolved and the finer grades are made by the vacuum-pan process. When the vacuum-pan process is employed the manipulation of the salt at the mill may be quite different from that outlined above.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Asbestos in Southern Quebec

BY JOHN A. DRESSER, SAULT STE. MARIE, ONT., CANADA

(Pittsburg Meeting, October, 1914)

## *General*

THE controlling supply of asbestos for the world is obtained from southern Quebec, 150 miles or less north of the international boundary line between Canada and the United States, and about 75 miles south of the city of Quebec. The principal production is furnished by eight mines, seven of which occur within a distance of 6 miles, but there are also several smaller properties in the vicinity. The industry was begun in a small way some 35 years ago and has advanced more or less regularly ever since. The annual production now exceeds 100,000 tons and its value is about $3,000,000. It represents over 80 per cent. of the world's production.

## *History*

Asbestos has been known in the eastern townships of Quebec since 1847, when attention was called to it in an official report by Sir William Logan, the first Director of the Geological Survey of Canada, but it was not until 30 years later that it came into commercial importance. The largest deposits, those of Thetford and Black Lake, were found in 1877 during the construction of the Quebec Central Railway. Work was begun upon them almost at once and has been continued ever since.

The Danville mine, the next largest producer, was opened in 1879, and the slip-fiber deposits of East Broughton were located shortly afterward.

For the first 15 years only the "crude" asbestos was recovered; that is, fiber long enough to be extracted by hand cobbing. Although this is still a valuable part of the production, it is now a relatively small part of the total output.

After several trials, a process of mechanical concentration was begun about 1893 by some of the pioneer operators of the district, which with many modifications has been successfully used ever since. Although there

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have been numberless changes in the operation and appliances, the present practice is a direct development of the first principles of the earliest attempts at concentration, and much credit is due to those who originated it. Its effect may be realized when it is stated that in leading mines to-day 95 per cent. of the quantity and 75 per cent. of the value of their total production is obtained by mechanical concentration.

The growth of the industry is best shown by quoting the production of a few years taken at regular intervlas:

| | Production, Tons | Value |
|---|---|---|
| 1878 | 50 | ..... |
| 1882 | 810 | $52,650 |
| 1892 | 6,082 | 390,462 |
| 1902 | 30,219 | 1,126,688 |
| 1912 | 111,175 | 3,059,084 |

### *Access and Location*

All the mines have easy railway access. The principal shipping stations are Thetford Mines, Black Lake, and East Broughton on the Quebec Central Railway, a part of the Canadian Pacific system, and Danville, on the Grand Trunk Railway. Thetford Mines is about 76 miles from Quebec, 67 miles from Sherbrooke, and 168 miles from Montreal. The other stations mentioned are from 4 to 18 miles from Thetford Mines, except Danville, which is some 50 miles distant, on the Grand Trunk Railway, 88 miles from Montreal and 86 miles from Quebec.

### *Geology*

The asbestos deposits are found in the hilly country of southern Quebec known as the Eastern Townships. Much of the district has been settled for upward of 100 years and is now generally occupied by small dairy farms. The hills are a continuation of the Green mountains of Vermont, a part of the Appalachian system.

The geological structure is complex. There has been intense folding, faulting, and regional metamorphism. Glacial drift conceals a great part of the rock surface, but glacial erosion has been an important factor in uncovering and exposing the deep-seated rocks in which asbestos occurs. The asbestos is in a series of basic igneous rocks which occur in stocks and sills, that have intruded sedimentary strata of Cambrian, Ordovician, and in places of Silurian age. These basic intrusives are part of the well-known and extensive series which appears at frequent intervals in the Appalachians from Georgia to Newfoundland, and yet in a distance of 2,000 miles the commercial production of asbestos is almost entirely limited to this small district.

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The principal types of igneous rocks are peridotite, pyroxenite, gabbro, and diabase. They are products of differentiation from a single magma, and are characteristically arranged in the order given above from the base upward in sills, and from the center outward in stocks. This order, it will be noted, is that of decreasing basicity and density. A relatively small amount of hornblende granite which is also present has generally been intruded a little later than the basic rocks.

Peridotite is altered in important amount to serpentine, and pyroxenite generally to soapstone, but in places probably also to serpentine.

### *Character and Mode of Occurrence*

The asbestos is entirely of the chrysotile variety, and has essentially the same chemical composition as serpentine, in which rock only it occurs. There are two types of asbestos in this district: namely, "cross-fiber" asbestos, and "slip," "parallel," or "mass fiber," asbestos, the three terms being used almost interchangeably.

*Cross-fiber Deposits.*—This variety furnishes the major production, both in quantity and value. It occurs in veins up to 2 1/2 in., or rarely 3 in. wide; the greater number are 1/2 in. or less in width. The fibers, as the name implies, lie crosswise the vein.

The veins rarely reach a length of 200 ft., but are usually very much shorter, the greater number being only a few feet in length. They run in all directions through the rock, in places cutting one another abruptly, but more frequently uniting at meeting. A careful examination shows that many of the larger and more persistent veins show an approach to a rectangular arrangement, and probably represent joints in the primary rock. Others have a roughly parallel order and denote fractures due to regional compression, while many smaller veins truncate the corners of rectangular joint blocks in shell-like form.

*Vein Structure and Origin.*—The veins are usually divided into two parts by a thin seam of iron ore, generally magnetite, which is parallel to the sides and near the center of the vein. Bordering the veins on each side there is invariably a band of serpentine about three times the width of the vein. The country rock near the veins is peridotite of the most basic phase (dunite) that occurs in the district. There is incipient serpentinization in all parts of the peridotite, but commercial asbestos is found only between walls of completely serpentinized rock whose thickness is proportional to the width of the vein. Microscopic and chemical evidences, as well as the distribution of the veins, point conclusively to the origin of the asbestos by alteration and recrystallization of the country rock *in situ*. Microscopically it is evident that the asbestos fibers have grown outward on each side from the seam of iron ore mentioned above; chemically

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there is no distinction between the asbestos and serpentine, while an essential difference between serpentine and peridotite is that the former contains 12 to 14 per cent. of water. The position of the veins and the deep-seated character of the rock make it impossible that such open fissures ever existed where the veins are now found. While much remains to be explained both as to the processes and their causes, the facts are self-evident that zones of the country rock have been altered to serpentine and proportionate parts of these have taken the form of asbestos veins. From the persistence of the seams of iron ore within, or at the sides of, the veins, and the correspondence of the position of the principal veins to that of joint and other fractures, the iron seams are believed to mark the original channels by which circulating waters have been introduced which produced the hydration of the peridotite and its alteration to serpentine, a certain portion of which has crystallized as asbestos.

The source of the water has given rise to considerable discussion. The fact that much excellent asbestos has been found near granite dikes has given rise to the opinion that magmatic waters accompanying the intrusion of these has been the cause. Locally there is much to support this view, but viewed broadly the small amount of granite as well as its variable distribution make it improbable that it has been the chief cause. Magmatic waters accompanying the peridotite itself are a more likely cause. Yet, this rock was solidified and was afterward fractured both by contraction (joints) and by regional deformation before the waters could have been introduced. On the other hand, the present water level is comparatively near the surface in this district, and, as the region is one that suffered great erosion in glacial times, the present surface may long have stood below ground-water level. It is not proved that meteoric waters are not a competent cause to produce these effects and at least a share in the cause may probably be attributed to them.

In the deepest workings yet made, which are a little more than 200 ft. deep, there is no apparent change in the quantity or quality of the product, and no deposit of serious importance has yet been exhausted. The principal deposits are found in the most basic parts of the peridotite (dunite); that is, near the base of sills or toward the central parts of stocks. Owing to glacial erosion having been more effective on the north side of the stocks several deposits are found near the north edge of the peridotite core of the principal stock.

*Slip-Fiber Deposits.*—Slip-fiber or parallel-fiber asbestos is a fibrous phase of serpentine in which the fibers are arranged parallel to adjacent cleavage faces in the rock. In places the rock is almost entirely in a fibrous condition and the name "mass-fiber" is sometimes used for such occurrences. It is probable that this class of asbestos has been derived from pyroxene.

Where slip fiber is abundant the proportion of asbestos in the rock is

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much higher than in the case of the cross-fiber occurrences. But much of the slip fiber is short and the proportion that is recoverable and useful, though higher, is not so greatly different from that obtained from the cross-fiber mines. Only a little "crude" asbestos is obtained from deposits of this class.

### *Mining and Dressing*

*Mining.*—The distribution of the asbestos is such that all the rock within the area mined must be handled. Except in one mine where underground work is quite extensively carried on, though principally in winter, or for development purposes, the mining, or rather quarrying, is all open-cast work. The ground is cut down in benches, generally 6 to 12 ft. high, which are carried across the floor of the pit so as to afford sufficient working face. Several of the pits have reached a depth of 200 ft. from the original surface and are from 600 to 1,200 ft. or even more in length.

*Handling.*—Hoisting is done by means of cable derricks with boxes carrying about a ton each. In one case a tramway enters the pit through an inclined tunnel and the rock is hauled out in cars drawn by a cable. Hauling on the surface is done by small locomotives with side-dumping cars of 4 or 5 tons capacity.

*Dressing.*—The separation of fiber from the rock commonly begins in the pit. Rock containing "crude"—that is, veins 3/4 in. or more in width and of good quality—is sent to the cobbing sheds for hand separation, and asbestos that is liberated by the breaking of the rock in the pit is collected in hand boxes; dead rock is taken to the waste dump, and the remainder, usually 35 to 60 per cent. of all the rock handled, goes to the ore bins, or directly to the mill for mechanical concentration.

The milling practice varies somewhat in different mills, but is very similar in all. It consists essentially of coarse crushing, drying, and alternate finer crushings and screenings. At each screening, the asbestos then liberated is drawn off through overhead pipes by suction fans and collected in settling tanks. When thoroughly screened from dust and classified according to length of fiber by means of a rotary screen, the different grades pass to their respective storage bins, or in some mills is mechanically bagged.

In the coarse crushing, jaw crushers are used, and gyratories and frequently rolls for the finer crushing. When rolls are used special appliances are needed for teasing out the fiber, which becomes compressed into matted sheets by the rolls. The final crushing of the rock is effected by a specially designed "cyclone." This consists of two "beaters" or fans of chilled iron, in shape like the screw propeller of a boat and weighing upward of 100 lb., which revolve at a speed of 2,000 rev. per minute, or more, in a closed chamber. From the rock fragments thus driven to-

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gether the smallest particles of asbestos are released and collected as before.

Suction fans for the removal of dust from the cyclone, the classifier, and sometimes from the mill, are important accessories to the equipment. Magnets are usually placed over the shaking screens to eliminate particles of iron ore. The average recovery of cross fiber seems to vary at different properties between 3 and 8 per cent. of the rock treated; slip fiber, perhaps 2 or 3 per cent. higher.

*Product.*—The fiber recovered in the mill is classified into three or more grades, the crude asbestos usually into two grades. The adoption of a standard classification has been discussed, but owing to local differences in the character of the fiber, as well as other causes, no standardization has yet been effected. Each mine follows its own grading and there is a lack of uniformity in the products of different mines. An arbitrary classification that has been adopted by the Department of Mines of the Province of Quebec is as follows:

*Crude Asbestos, Hand Cobbed*

No. 1.—Value $200 per ton or more.
No. 2.—Value less than $200 per ton.

*Mill Stock, Mechanically Separated*

No. 1.—Value $45 per ton or more.
No. 2.—Value $20 to $45 per ton.
No. 3.—Value less than $20 per ton.

The production of 1912 was as follows, according to the classification given above:

| Quantity of Rock Mined 1,870,608 Tons | | | Shipments | | | Stock on Hand | |
|---|---|---|---|---|---|---|---|
| Qualities | No of Men Employed | Wages Paid | Tons | Value | Av. per Ton | Tons | Value |
| Crude No. 1 | ..... | ..... | 1,914 | $510,785 | $263.16 | 867 | $221,215 |
| Crude No. 2 | ..... | ..... | 3,766 | 379,445 | 100.76 | 2,867 | 310,596 |
| Mill Sk. No. 1 | ..... | ..... | 3,682 | 237,203 | 64.42 | 2,370 | 137,106 |
| Mill Sk. No. 2 | ..... | ..... | 32,689 | 1,018,960 | 31.17 | 8,234 | 301,774 |
| Mill Sk. No. 3 | ..... | ..... | 69,097 | 912,691 | 13.21 | 6,838 | $131,206 |
| Totals | 2,910 | $1,377,444 | 111,175 | 3,059,084 | 27.52 | 24,176 | 1,102,206 |

The stock on hand at the beginning of the year 1912 amounted to 33,751 tons, valued at $1,583,076.

*Uses.*—A small proportion of the crude asbestos is used for making asbestos cloth and various fire-proof textiles. A much greater amount

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is used for covering and insulating purposes. Boards, shingles, and roofing felts for fire-proof construction, materials for electric insulation and protection from acids, and boiler and pipe coverings are the forms of manufacture in most common use. There is only one plant for the manufacture of asbestos goods in Canada, the Asbestos Manufacturing Co., at Lachine, Quebec. This plant makes shingles and other roofing materials, pipe covering, mill boards, and asbestos lumber for the Canadian and European markets. Its consumption is about 5 per cent. of the total output of the mines of Quebec. Approximately 75 per cent. of the output of these mines is exported for manufacture to the United States, and the balance to Great Britain, Germany, France, and other European countries.

### *Condition and Outlook for the Industry*

There has been a decrease of stock on hand during the past two years. This seems to indicate that the industry is emerging successfully from a critical condition into which it was thrown four years ago by a rapid overproduction, which resulted from some extravagant financial promotions.

The deposits are large and the principal mines seem to have almost inexhaustible reserves. Consequently the plants installed are of a durable character and around the mines substantially built towns have grown up. The labor is largly obtained in the locality. Many of the men own their houses, living conditions are favorable, and there have been no serious labor troubles in the history of the industry.

For the immediate future, or as long as mining can be done by open-cast methods, the cost will probably vary only with the nature of the ground, the depth of hoisting, and the price of labor and materials. But in methods of concentrating changes are more likely to take place. The present practice is the result of 20 years' experience, during which time many changes, and great improvements have been made. At present the practice of different mills varies considerably in details and probably also in efficiency.

The enlargement of the market for manufactured goods, which is steadily growing, depends largely on the skill and enterprise of the manufacturers and the supply of substitutes for asbestos. The constantly increasing price of lumber must improve the field for shingles and asbestos lumber, while the ingenious applications of asbestos for heat-resisting and insulating materials seem likely to give it a continued advantage over any substitutes at present known.

The absence of rival fields for so long a time is rather remarkable. Similar rocks in like associations are known in many parts of the world, yet the only sustained and growing production outside of Quebec is obtained from the Ural districts in Russia, the United States and South Africa

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being much smaller producers. In his report on Mining Operations in the Province of Quebec for 1912, T. C. Denis, Superintendent of Mines of Quebec, cites the following statistics for the year 1910, from the Colonial and Foreign Statistics of the Home Office, London. The quantities are given in metric tons and values in pounds sterling:

| | Production Metric Tons | Value |
|---|---|---|
| Cape Colony | 1,273 | £23,143 |
| Cyprus | 442 | 2,754 |
| India | 3 | 6 |
| Rhodesia | 301 | 3,320 |
| Natal | 2 | 15 |
| Transvaal | 70 | 2,575 |
| Russia (approximate) | 10,936 | 82,000 |
| United States | 3,350 | 14,036 |
| Total outside of Quebec | 16,377 | £127,849 |
| Quebec | 73,124 | £548,184 |

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Asbestos Deposits of Georgia*

BY OLIVER B. HOPKINS,† ATLANTA, GA.

(Pittsburg Meeting, October, 1914)

As prefatory to the body of this paper, a few general statements will be made (1) in regard to the history and importance of the asbestos industry, (2) as to the principal sources of the raw material, and (3) as to the types and modes of occurrence of asbestos.

While the modern asbestos industry is comparatively new and its growth during the last decade or two has been marvelous, asbestos has been known and used in a small way for probably 2,000 years—at least since the time of the Roman Empire. Pliny refers to it as "the funeral dress of kings" and Plutarch records its use as lamp wicks. In the middle of the 18th century the deposits of the Ural mountains were opened up and the first factory for the manufacture of asbestos goods was established. But, owing to the limited demand for the goods, the industry disappeared and interest lapsed until about 50 years ago. Since 1860, the asbestos industry has made wonderful strides, and each year sees new uses to which the material can be put, and hence an increased demand and an increased output.

Of the three great producers of asbestos, Canada, Russia, and the United States, Canada stands pre-eminent, producing from one-half to three-fourths of the world's supply; Russia comes second, producing in 1911 about two and one-half times as much as the United States. In 1912 the Canadian production,[1] including both asbestos and asbestic, was 131,260 short tons, valued at $2,979,384; while during the same year the production of the United States[2] was 4,403 tons, valued at $87,959. Canada holds the first place in the production of raw asbestos, while the United States holds the first place in its manufacture.

Although the production of asbestos in the United States is comparatively small there are deposits of varying importance in the following States: Vermont, Virginia, North Carolina, Georgia, Wyoming, Idaho,

* Published by permission of the Geological Survey of Georgia.

† Assistant State Geologist of Georgia. Non-member.

[1] *Summary Report of the Mines Branch of the Department of Mines of Canada*, year ending Dec. 31, 1912.

[2] J. S. Diller: *Mineral Resources of the United States*, 1912, Advance Chapter on Asbestos.

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Arizona, and California. Of these States, asbestos has been worked commercially in Vermont, Virginia, Georgia, and to a small extent in Wyoming, Idaho, and California. In production, Vermont ranks first and Georgia second.

### *Asbestos-Forming Minerals*

The asbestos-forming minerals come naturally under two heads, amphibole and serpentine. In the amphibole group there are a number of different minerals of quite variable compositions, while under serpentine there are only two which have practically the same composition, and only one which is of importance. The following tabulation shows the relationships of these minerals:

*Asbestos-Forming Minerals:*
- Amphibole Group.
  - Orthorhombic division.
    - Anthophyllite, $(MgFe)SiO_3$.
  - Monoclinic division.
    - Tremolite, $CaMg_3(SiO_3)_4$.
    - Actinolite, $Ca(MgFe_2)(SiO_3)_4$.
      - Mountain Leather,
      - Mountain Wood.
      - Mountain Cork.
    - Crocidolite, $NaFeSi_2O_6.FeSiO_3$.
- Serpentine, $H_4Mg_3Si_2O_9$.
  - Var. Chrysotile.
  - Var. Picrolite.

The most important of these minerals in point of view of quantity and accessibility are chrysotile, anthophyllite, and crocidolite; while from the point of view of commercial application chrysotile is by far the most important with anthophyllite probably second. The amphibole type of asbestos occurs in Georgia, Idaho, North Carolina, and Virginia; and the serpentine variety occurs in Vermont, Wyoming, Arizona, and California.

What are the properties of asbestos upon which its value depends? An answer to this question involves the knowledge of the physical properties of asbestos and gives an insight into the uses to which the material is put. The most important properties are its fibrous quality, resistance to heat and acids, and low conductivity. Under the head of fibrous quality the following characteristics may be considered: Fineness, length and flexibility of fiber, and tensile strength. Not only do the various types of asbestos vary widely in regard to these properties, but the relative value of the properties varies with the use to which the material is put. In regard to fineness, length, flexibility of fiber, and tensile strength, crocidolite is quite the equal of chrysotile, and superior in some respects; but in fire-resisting properties crocidolite is seriously lacking. Anthophyllite is equal to any other type of asbestos in regard

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to its resistance to acid and heat and insulating properties, but far inferior to chrysotile in regard to flexibility, fineness of fiber, and tensile strength. Thus it is that the anthophyllite, and the related types of amphibole asbestos, which constitute the most important deposits in the Southern States, are classed as low grade. Anthophyllite is not suitable for some of the purposes for which chrysotile is used, and hence it is considered of little or no value by some of the producers of higher grade material; but where quality of fiber is not important and where resistance to heat and acids and low conductivity are essential, anthophyllite is as valuable as any other type of asbestos.

Asbestos occurs in three forms:[3] as cross fiber, slip fiber, and mass fiber. When the asbestos occurs in veins with the fibers at right angles to the inclosing walls it is called cross fiber; when the fibers are parallel to the inclosing walls it is called slip fiber; and when the asbestos forms the entire rock mass composed of bundles of fibers arranged in all directions in respect to each other, often in somewhat radiating forms, it is called mass fiber. Cross fiber may be converted to slip fiber by movements of one wall in respect to the other. Slip fiber is found in the slipping plane, and the direction of the fibers records the direction of motion.

Chrysotile occurs as cross and slip fiber, but never as mass fiber; its characteristic mode of occurrence is as cross fiber. Fibrous tremolite and actinolite occur only in slip-fiber veins, while anthophyllite occurs in all three forms, but its characteristic form is as mass fiber.

Diller[4] has recognized four modes of occurrence of asbestos in the United States, which are as follows:

"The first mode is as cross-fiber veins of chrysotile in serpentine derived from peridotite, a deep-seated igneous rock, as near Lowell, Vt., and Casper, Wyo. So far as known, it is much the most important mode of occurrence and is well illustrated at the Thetford mines, in Canada.

"A second mode is as cross-fiber veins of chrysotile with serpentine in limestone. Its most important illustration is in the Grand Canyon of the Colorado in Arizona.

"The third mode of occurrence is as mass-fiber amphibole (anthophyllite), composing stocks and dikes of fibrous amphibolite, and is well illustrated in the deposits worked for many years at Sall Mountain, Ga.

"The fourth mode is as slip-fiber veins in rocks which for the most part are cortlandite and pyroxenite, but which locally pass into peridotite, as at Bedford and Rockymount, Va."

### *Location*

The asbestos deposits of Georgia are associated with the crystalline schists of pre-Cambrian age which occupy both the Piedmont Plateau and the Appalachian Mountain provinces. While asbestos has been

[3] Diller, J. S: *Bulletin. No.* 470, *U. S. Geological Survey*, p. 506 (1911).
[4] *Op. Cit.*, pp. 506, 507.

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found associated with basic igneous rocks over a large part of the area of crystalline schists, the distribution of commercial deposits is much more limited, and is, in general, confined to a belt extending from Rabun and Towns counties southwest into Alabama in the neighborhood of West Point. And even in this belt, which includes all of the deposits likely to be of commercial value, the most important deposits are grouped in its northern part, and more specifically in Rabun, White, and Habersham counties.

### *Associated Rocks*

The asbestos deposits of Georgia are limited in their distribution to the areas of basic, igneous rocks, which are represented in large part by the following types: Hornblende schist, hornblende gneiss, diorite, gabbro, peridotite, and pyroxenite. All these types of rocks, except peridotite and pyroxenite, have a wide distribution in this State. The hornblende schist and hornblende gneiss are found in varying quantities in every county in the crystalline area, occupying long narrow bands and folded with the rocks of the Carolina gneiss series, which vary from mica schist to granite gneiss. The gabbros and diorites are found associated with the hornblende rocks, into which they sometimes grade, occupying broader and less elongated areas.

Associated with these hornblendic rocks, diorites, and gabbros are small deposits of slip-fiber asbestos, which give little or no promise of commercial value, and the distribution of these deposits is practically coextensive with the distribution of those rocks.

The peridotites and pyroxenites, on the other hand, are more limited in their distribution and it is with them that the most important asbestos deposits of the State are associated. Their distribution is, in general, limited to the belt in which the most important asbestos deposits of the State occur.

### *Types of Asbestos Found*

Asbestos has been found in Georgia representing three of the four modes of occurrence described by Diller as occurring in the United States, namely: Chrysotile in cross-fiber veins associated with serpentine, which is derived largely from peridotite; mass-fiber anthophyllite in dikes or irregular masses; and slip-fiber veins in different types of basic rocks, varying from courtlandite to peridotite or pyroxenite.

Asbestos of the first type, as chrysotile in serpentine rocks of igneous origin, is found in Georgia in a limited number of places and in even more limited quantities. At none of the places examined were the veins more than $\frac{1}{8}$ to $\frac{1}{10}$ in. in thickness; the veins were very limited in distribution, and constituted a very small portion of the rock mass. It

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can be quite definitely stated that this type of material does not occur in commercial quantities in this State.

Amphibole asbestos is of much more general distribution and is present in much larger quantities. Slip fiber occurs at numerous localities over the crystalline area of the State, where it is found along shearing planes in the basic rocks described above. Its commercial possibilities, while somewhat more promising than the deposits of the chrysotile type, are as a rule unworthy of consideration because of the large amount of waste material which has to be removed, since the veins are usually not more than a few inches in thickness and seldom reach as much as 8 to 10 in., although they are exceptionally 16 to 18 in., in width.

Thus the only type left for consideration is the mass-fiber anthophyllite deposits. These deposits are unique in that more than 90 per cent., usually as much as 95 per cent., of the rock quarried is realized as fiber, while in the case of the chrysotile deposits of Canada the average extraction of fiber for 1912 was only 6.45 per cent. of the rock quarried. Although the extraction of the Canadian chrysotile is only about one-fourteenth that of the Georgia mass-fiber asbestos, the price is less than three times as great. This gives a decided advantage to the Georgia product, especially since the mining can be done more economically and the milling is much more simple.

| Canadian Asbestos[5] | Georgia Asbestos |
|---|---|
| Extraction, 6.45 per cent. | 90 to 95 per cent. |
| Average price, 1912, $27.79 per ton | $10 to $12 per ton.[6] |

Extraction, one-fourteenth that of the Georgia anthophyllite.

Price, two and four-fifths times that of Georgia anthophyllite.

The circumstances which keep the Georgia deposits from being developed are the limited demand for the material, the distance from the railroad, and the limited size of the deposits, features which will be dealt with later.

### *Description of Localities*

A detailed description of the more important asbestos deposits or even an enumeration of them would serve no useful purpose in this connection; however, a brief account of the best-known deposits, as typical of the class, may serve as a general picture of them all. Figs. 1 and 2 are views on the property of the Sall Mountain Asbestos Mfg. Co. and Fig. 3 shows a deposit at the mine of the Asbestos Mining & Mfg. Co., Hollywood, Ga.

[5] *Summary Report of the Mines Branch of the Department of Mines of Canada*, Year ending Dec. 31, 1912, p. 156.

[6] The figures here given represent the general range in value for several years and not the value for 1912, as that information is confidential, there being only one producer.



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### *Sall Mountain Mine*

The Sall Mountain mine is the oldest producing asbestos mine in the United States. It began operation about 1894, and until the last few years it was the largest producer in the United States. The mill has a capacity of 10 tons per day. It is the only mine, except the one at Kamiah, Idaho, which was worked a few years ago, where mass-fiber anthophyllite has been mined.

The rock is a fibrous amphibolite in which the bundles of fibers range from 1¼ in. down to a small fraction of an inch in length. But, owing to the lack of flexibility, the fibers break into short lengths on being

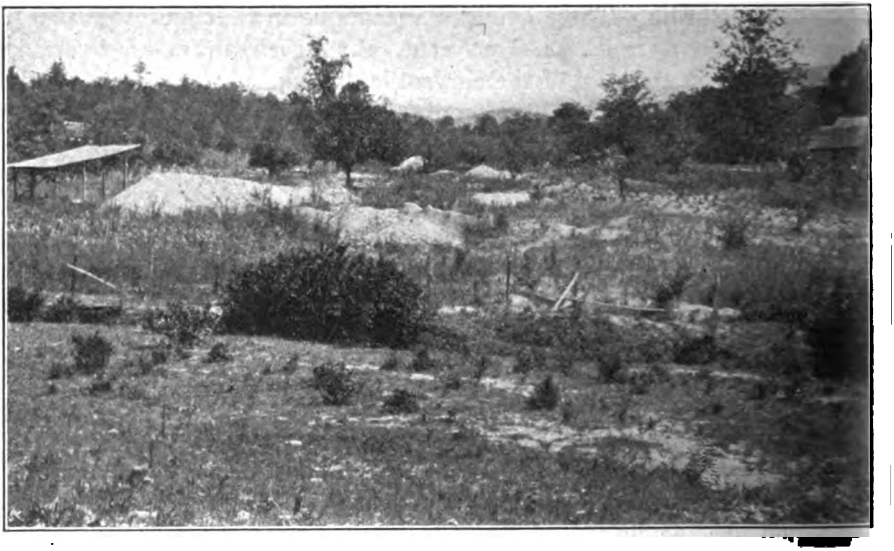

FIG. 1.—WEATHERED ASBESTOS ROCK DUG FROM SMALL PITS ON SALL MOUNTAIN COMPANY PROPERTY NEAR CLEVELAND, GA.

fiberized, so that the longest fiber is about ¼ in., and ranges down to $\frac{1}{10}$ in. and less. More than 95 per cent. of the rock mass is converted into fiber. Pyrite, magnetite, talc, and carbonate of lime and magnesia constitute the impurities. The asbestos is grayish white when fresh, but a large part of that produced has been stained yellow by surface water.

The country rock is a gneiss cut by more recent granite. The asbestos occurs in lens-shaped masses, of which six of varying size have been worked in an area of not more than 20 acres. The largest body was about 75 ft. long by 50 ft. wide as a maximum, and has been worked in an open cut to a depth of more than 50 ft. This company has largely worked out its deposits on Sall mountain, but owns others near Asbestos station on the Gainesville & Northwestern Railroad, where the material

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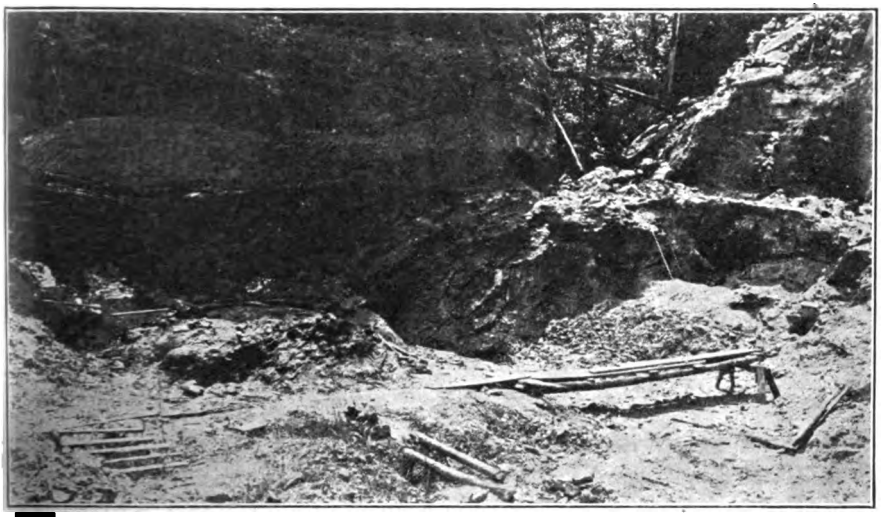

FIG. 2.—ASBESTOS MASS BENEATH DECOMPOSED GNEISS, SALL MOUNTAIN MINE.

FIG. 3.—ASBESTOS ROCK ON THE PROPERTY OF THE ASBESTOS MINING & MFG. CO., HOLLYWOOD, GA.

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is identical in association and character, except in the amount of disintegration which surface waters have produced.

Mass-fiber anthophyllite occurs at a number of other localities in White, Habersham, and Rabun counties, usually associated with partly altered rock, slip fiber, and talcose material. There are a number of places which give promise of yielding as much or more asbestos rock than the Sall Mountain property. The occurrence at all these places is very similar and the origin appears to be the same, but the quantity of asbestos at an individual place is difficult of determination because of the irregular shape of the bodies and the variation in the amount of alteration in the individual mass.

### *Origin of Mass-Fiber Deposits*

From the field relationships and the microscopic study of much of the material from Georgia the conclusion has been reached that mass fiber develops from olivine-enstatite rocks.

In regard to field relationship, it has already been stated that the important asbestos deposits, those of mass fiber, are limited to the belt of peridotites and pyroxenites. Wherever there is any rock associated with the asbestos which shows an original mineral or minerals it is universally found to contain olivine or enstatite, or both.

A microscopic study of the rocks in thin section has developed the facts that olivine and enstatite give rise to anthophyllite, and that all gradations from olivine-enstatite rock to anthophyllite are present. The alterations noted are briefly as follows: The development of slender needles of anthophyllite running in all directions in the olivine-enstatite rock and penetrating the crystals regardless of cleavage cracks and crystal boundaries; the increase in number and size of anthophyllite needles at the expense of the original minerals until only skeletons remain; and finally the anthophyllite rock is formed. But this alteration may have been stopped at any stage in its progress by changed physical conditions; by processes of hydration and oxidation, the anthophyllite, as well as the remaining olivine, may be partly converted into serpentine and the latter partly to talc; the result may be a rock containing olivine, anthophyllite, serpentine, talc, magnetite, and carbonate of calcium and magnesium.

### *Quarrying*

The mining of asbestos in Georgia has been entirely from open cuts or quarries. Rough, brownish boulders usually outcrop at the surface. By a study of the distribution of surface fragments and by digging a few shallow pits the size of a deposit near the surface may be fairly

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well outlined, although the irregular shape of the bodies precludes even a rough estimate of the amount of material present.

When work is started, the deposit is followed in all directions; but the removal of the rock is commonly begun along the contact with the country rock, as there the rock is usually soft and a face can be easily made to blast from. The removal of soil at all the localities where development has been undertaken is usually a small item. The rock is frequently soft enough to dig with a pick, although irregular masses of hard rock have to be drilled and blasted. The work is done largely by hand; the rock is loaded in small hand cars or wheelbarrows and hauled to the drying sheds.

### *Milling*

The crude asbestos is dried either in open sheds or over steam pipes; when sufficiently dry it is passed through a jaw crusher, and then through a rotary crusher, where the rock is reduced to 1-in. mesh and less. From the crusher, the rock is passed to a bin, from which it feeds into a Raymond pulverizer. As the rock is fiberized it is carried upward by a suction fan and separated by air currents, the coarse material returning to the pulverizer, the fines collecting in a bin which feeds into a packer, and the dust settling in a dust room.

### *Uses of Asbestos*

One of the principal uses to which the anthophyllite asbestos has been put is in the preparation of asbestos cement for boiler coverings and furnace linings. Twelve (?) car loads of the Georgia product were used at Ducktown in lining the flues. This type of asbestos may be used in the manufacture of the following articles: Furnace linings, fire bricks, steam-pipe coverings, asbestos shingles, wall plaster, asbestos tiling for floors, etc. Numerous experiments have been made with the amphibole asbestos, but, owing to the fact that the largest manufacturers of asbestos in this country are interested in the Canadian mines, it is difficult to get this material on the market in the United States. The greater part of the mineral now being mined in Georgia is sold in Germany. During the past year the manufacture of asbestos cement from Georgia asbestos was begun here and a considerable quantity has been sold for local use. Considering the uses to which it can be put and the cheapness of its extraction and milling, it seems that there should be a future for the material, notwithstanding the enormous output of low-grade asbestos from the chrysotile areas.

### *Summary*

Asbestos representing three modes of occurrence is found in Georgia. Chrysotile, occurring in serpentine which is derived from peridotite,

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is present in insignificant quantities in a few localities, where it gives no promise of commercial value. Asbestos of the amphibole variety in slip-fiber veins occurs at widely distributed points over the Piedmont area of the State; while mass-fiber asbestos, which represents the most important deposits from a commercial point of view, is restricted in general to the belt of peridotite and pyroxenites which crosses the State in a southwest direction from Rabun county to Harris county, but is relatively more important in Rabun and Habersham counties than in any others. Judging from the field relations and the microscopic study of the material it has been concluded that the mass-fiber anthophyllite is derived from enstatite-olivine rocks.

Mass-fiber asbestos, owing to the nature of its occurrence, is capable of being mined very economically, but owing to the slight demand little is being put on the market at the present time. With a good demand for the material, at from $8 to $12 per ton a number of deposits in this State could be worked with profit and a large amount of the material could be put on the market.

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# The Reserves of Iron Ore for the United States

BY JOHN BIRKINBINE, PHILADELPHIA PA.,

(Pittsburg Meeting, October, 1914)

EXTENDED discussions, by inviting attention to problems affecting the conservation of natural resources, have encouraged investigations as to their sufficiency, with the general result that the more thorough the examination, the less cause appears for anxiety as to the exhaustion or serious depletion of these resources. The extent of industrial development, the large plants and the enormous volumes of materials required, when compared with conditions a few decades ago, are sufficiently startling to enlist the attention of those who "fear the pace," and yet in but few instances has real solicitude for the future been warranted by a knowledge of true conditions.

In a former communication, presented 5 years ago,[1] I endeavored to emphasize the fact that true conservation represented the best utilization of reserves and not the locking up of resources for future use; a conviction which has become more pronounced as the subject was investigated. Prophecies of the early exhaustion of mineral resources have raised the question as to our permanent reliance upon these, and encouraged inquiries which have in most cases developed reassuring conditions. But the fact that the future appears less dark than prognosticated cannot excuse wastefulness or improvidence; and detailed study which demonstrates an apparent abundance of natural resources should encourage their best application.

In making inventories of resources one is liable to confine conclusions as to availability upon present circumstances and conditions in the past, or to formulate data for the future upon methods which are rapidly being displaced. In reserves which are not reproductive, such as minerals, the possibilities of exhausting the supply are startling if estimates assume that future extensions will follow methods which were less economical than those now employed, or to be used, and we dare not claim that the limit of advanced practice has been attained.

Examinations and estimates of iron-ore reserves may be taken as

[1] *The Conservation of Natural Resources, Trans.*, vol. xl, pp. 412 to 418 (1909).

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illustrative to determine whether an ample supply exists in our own country, or judge of the possibilities of liberal accretions which may be obtained from deposits of phenomenal character or size located elsewhere.

The prevalent method of estimating iron-ore resources has been to consider them as applicable to the present status and location of industrial development, to methods of utilization in vogue, and to processes which are accepted as the best in the light of existing knowledge.

A retrospect of the history of iron production in the United States demonstrates that the location of industrial centers dependent upon the smelting of iron ores has materially changed, and that many blast furnaces, formerly relying upon local ores, now obtain their supplies from distant points. The abundance and quality of mineral has caused the greatest development during the past 20 or 30 years at localities to which ores from the Lake Superior region have been transported from rail to vessel, and again from vessel to rail to reach smelting plants; but other industries have been created to supply a demand in sections of the country which several decades ago offered but a limited market for manufactured products.

As population increases, as towns and cities grow, as railroads are extended in various parts of the country more or less sparsely inhabited, as water transportation is cheapened, and as requirements from other countries may make drafts upon or furnish material for the products of the United States, changes in industrial centers may be expected. We, therefore, may not limit consideration of the iron-ore reserves solely as affecting the iron and steel industry as it now exists in this country, for railroad and water transportation cheapen the cost of delivering materials from distant points and facilitate the shipment of manufactured products.

Mechanical ingenuity, by displacing manual labor, admits of greatly augmented outputs and larger plants, while local industrial development is an early desire of growing communities.

Our iron-ore reserves are to be estimated as supplying our present industries and also as encouraging the development of new enterprises in a country of vast area, rapidly increasing in population and in markets for iron and steel products.

The history of the Institute, as set forth in its *Transactions*, shows how the centers of production and consumption have shifted, and illustrates changes which may be expected, at least in part, to be repeated.

Another factor apparently neglected is that but few iron-ore mines have been worked to exhaustion, for inquiries covering several years failed to develop any considerable number of iron-ore mines which could be classed as worked out. There are mines which have been abandoned because of accident, or by reason of apparent inferiority of the mineral to other ores latterly obtainable.

There are instances of mines flooded because of inefficient pumping

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apparatus which have ceased operations, and of others which have been wrought without modern labor-saving appliances and were not economically exploited, but some of which may be revived and made productive by utilizing equipment of modern design and capacity. Other mines lie dormant by reason of some constituent, which is or has been considered as deleterious, or because existing transportation facilities do not encourage immediate development, and limited financial ability of some who attempted to operate iron mines or blast furnaces, is also blamable for inactive deposits.

But a careful review of the iron-ore deposits of the United States, which have been or are wrought, shows remarkably few instances where the deposit has been "bottomed" or followed to its lateral termini; on the contrary, the testimony of those operating iron-ore mines, proves that in the majority of instances, the more a deposit is exploited, the greater are the apparent reserves. After nearly 60 years of activity, new "finds" of iron ore are reported from the Marquette range in Michigan, and the same is true of the Menominee range, which was opened 20 years subsequent to the Marquette range.

The Gogebic range of Michigan and Wisconsin, after a history of 30 years, has been demonstrated to possess enormous deposits of desirable iron ores below what had previously been considered a dike defining the depth of orebodies. And later discoveries between the Gogebic and Eastern ranges have proven large quantities of iron ore. In Minnesota, which 30 years ago supplied no iron ore but has grown to be the largest known producer, ore is now obtained in sections which until lately were considered beyond the productive limits; while explorations are developing additional ore supplies. One hard-ore property, which had been a large producer but abandoned because of its apparent exhaustion, is still contributing to the ore supply and the deposit has been proven to extend into the adjoining properties, where it is successfully wrought.

The magnetic iron mines of the Lake Champlain district, which have long been active, show no signs of exhaustion and new explorations are exposing orebodies whose magnitudes have been estimated in hundreds of millions of tons. The Cornwall ore banks in Pennsylvania, which have been supplying ore for over 170 years, have lately been equipped on a scale demonstrating that the owners are satisfied of liberal reserves for many years to come. Active mining, which has been prosecuted for about 40 years in the Birmingham district in Alabama, does not suggest exhaustion, but rather directs attention to the probabilities of a liberal supply of ore for the future. Iron ores have been determined as existing in liberal amounts either by exploration or by more or less continued exploration in the eastern and central parts of the country, also in the Rocky Mountain region, and the territory to the west. In Colorado an important iron and steel industry has been maintained for

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more than three decades upon ores mined from deposits, some of which are of known magnitude in the Rocky Mountain region.

The litigation between the National Government and the United States Steel Corporation brought forth sworn testimony from experts, among which were the following estimates of iron-ore reserves:

In Michigan: 170,000,000 long tons,[2] or if high-silica, low-phosphorus ores of 40 per cent. grade are included, 219,000,000 long tons.

Minnesota in 1911 was credited with 1,670,000,000 tons, the Cuyuna being then estimated at 40,000,000 tons and subsequently increased to 400,000,000 tons. The tonnage of reserves of the Great Northern lands was estimated at 235,416,756 tons.

If all ore above a minimum of 45 per cent. of iron is considered, there would be an additional iron-ore reserve in the Lake Superior district of at least 2,000,000,000 tons, and if the minimum is 35 per cent., 70,000,000,000 tons.

The Adirondack region of New York was estimated to contain 800,000,000 tons, which would yield 300,000,000 tons of furnace ore; and for southeastern New York 60,000,000 tons of reserves were estimated.

New Jersey was stated to contain 130,000,000 tons of reserves, of which 30,000,000 tons were shipping ore, and the balance would yield 40,000,000 tons of concentrates carrying 60 per cent. of iron.

In Alabama, the Birmingham district was estimated as having 1,300,000,000 tons, the Russellville district 100,000,000 tons, and eastern Alabama not over 25,000,000 tons.

Virginia was credited with 100,000,000 tons and Texas, in Cass and adjoining counties has 250,000,000 tons of brown ores.

To the above could be added reserves of considerable quantities in mines which have been exploited, and possibilities of known iron-ore deposits in sections of the country which have not heretofore supplied iron ores or offered a market for them.

In nearly all the political divisions of the United States iron ores of apparently desirable quality and quantity are known to exist and geological reports upon a number of important iron-ore deposits demonstrate the probabilities of liberal extensions beyond present exploitation in depth or length or both, and the working of mines has in many instances fully warranted the geologists' conclusions.

Iron-ore producing regions are generally areas of iron-bearing rocks, in which there are local concentrations carrying sufficient iron to encourage exploitation as merchantable iron ore, and the grouping of the deposits which have been or are now the sources of iron-ore supply demonstrate that the above may be considered as the rule rather than the exception, although exceptions may be recalled. Until these iron-ore

[2] The Michigan Tax Commission in 1913 made new estimates of the reserves of ore, showing a total of 189,467,621 tons.

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bearing areas are thoroughly explored, new discoveries of workable mines may be expected, and until exploitation discloses the limits of the mineral, their capacities cannot be calculated.

To the above may be added the foreign sources which may supplement the domestic supply of iron ores, upon some of which testimony was presented in the litigation above mentioned.

Cuba was estimated to have 3,000,000 to 6,000,000 tons of hard ore, and 3,000,000,000 to 3,400,000,000 tons of soft ores.

The Tofo mines in Chili had a mountain of ore, which for a distance of but half way from the top was estimated to have 150,000,000 tons of 67.5 per cent. ore, and if taken to the level of the valley would yield probably three or four times as much.

The Newfoundland iron ores were estimated at 3,250,000,000 to 5,000,000,000 tons, and in the Minas Geraes district of Brazil 7,000,000,000 to 8,000,000,000 tons of 60 per cent. iron ore were claimed.

Sweden is credited with 1,500,000,000 tons of reserves in Kïruna and 493,000,000 tons in other deposits.

Much additional detailed information concerning the future supply of iron ore appears in the volume entitled *The Iron Ore Reserves of the World*, a part of the proceedings of the International Geological Congress held at Stockholm, Sweden in 1910, and in published monographs on special districts or deposits, many of the estimates of which have been exceeded by later detailed reports.

Some iron industries formerly important have passed into memory and mines which for long terms of years produced mineral lie dormant, but on the other hand new developments are noted and mining operations are on a scale of magnitude attracting world-wide attention. Industries of moderate size which these effete installations supplied have expanded and demand iron or its manufactures in large volume. The mines abandoned may not have yielded all their ore, but in the march of progress the volume of business has been greatly extended and other sources are utilized. It is not improbable that some of the idle plants and dormant mines may be revivified and that the pendulum of production which in the United States swung geographically westward may return, at least in part, to the East. Localities could be cited where iron-ore mines have been or are operating, for which continuous life may be confidently expected, and the more thoroughly they are explored, their greater possibilities may be demonstrated.

Another phase in estimates of reserves is the changing demand for ores of specific compositions, as was instanced by the apparent scarcity of ore of Bessemer grade being the incentive for a liberal development of the basic openhearth steel industry, thus making available much mineral formerly considered under the ban and reducing the premium on Bessemer ores. Millions of tons of concentrated ore, treated to raise the iron con

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tent or eliminate undesirable constituents, are now annually used in the United States. Northern New York contributes large amounts of magnetic concentrates, Minnesota and the Southern States supply quantities of washed ore, suggesting that other deposits in which the minealr is lean in iron, or which contains objectionable ingredients, may, with modern appliances, add to the supply of the country.

Heat treatment of iron ores also increases the available supply by sintering or nodulizing finely comminuted material and driving off sulphur and volatile matter.

All methods of beneficiating iron ores add to the cost of the marketable product but the high grade obtained may offset this expense by commanding increased ton rates, and the outlay necessary to improve local ores may be less than the transportation charges on others from distant mines.

Metallurgy, which has mastered many difficult problems, may be expected to overcome successfully apparently unfavorable compositions of iron ores. Copper, which has been objected to as a drawback in manipulating iron associated with this element, is now desired by some manufacturers, and the presence of this metal within certain limits even suggests a premium due to its occurrence with the iron. Objection to the use of sulphurous iron ores is less pronounced than formerly, for these are fed to furnaces whose burden is adjusted and fluxed to practically neutralize this element. Titanium, chromium and other elements which have been under the ban are accepted as component parts of iron ores for use, and phosphorus, which was classed as most objectionable, has lost much of its bad reputation, and is even purchased to add to the charge fed to some blast furnaces which smelt iron ores deficient in this element.

Most estimates of iron-ore reserves assume that the present processes of smelting and conversion are to continue. Marvelous progress has been made in perfecting these processes, quantity and quality of product being obtained at costs far below those of a few decades ago, and this advance may be expected to continue, with possibly even more radical changes in the future than have heretofore taken place. The blast furnace has been the accepted appliance for producing iron for centuries, but the plant at the present time is quite different from one of even 50 years ago, not only in the quantity produced but in the materials used, the practice followed and the resultant economies. It is possible that another 50 years may so change the smelting process as practically to eliminate the blast furnace, and in estimating the iron-ore reserves we should consider that new methods may be employed for using ores which are now considered relatively undesirable.

The production of ferric alloys of titanium, phosphorus, chromium, silicon, manganese, etc., warrant the expectation that these may grow in importance, and that some ores in which the above or other elements are

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prominent, may be in much demand, for in many lines of industry steels of specific composition, in which some of the rarer elements are alloyed, are sought and the production of such steels has already reached an important volume. For such purposes ores of which iron is the base, but which are at present lightly esteemed because of other items of composition, may be sought and liberally employed.

While in the United States the electrical furnace has been primarily applied to refining steel, the possibilities of electrical smelting may command attention, especially where rich ores can be treated with electricity developed by water power or by fuel at low cost, a possibility influenced by the highly economic operation of turbogenerators.

Considering the various features above enumerated and remembering how much of our mineral wealth remains to be proven, there would seem to be little ground for anxiety as to the future supply of iron ores for the United States, especially if the question as to the years for which a supply should be calculated enters into the discussion. Such an estimate should take into consideration the factors which have been above outlined and also the financial problem; for it is fairly presumable that any deposit of iron ore demands an outlay of money for its purchase or leasehold; and for the capital invested in plant upon which interest and amortization must be provided, while the growing tendency to base taxation upon calculated reserves places on the iron miner the necessity of paying continuously the taxes upon mineral which he may not win for many years, or which may be ultimately lost by fire, flooding or other accident.

The return for the money invested in the iron and steel industry or in the mining of iron ore, will be affected by the duration of the demand for its products, and by competition which must be met from other developments or industries subsequently brought into activity. The investor will consider the extension of productive facilities elsewhere, the growth of communities and the local demand which these will make. If the ore produced is practically free from undesirable components it may have to meet competition from other ores from which these components are removed by processes of beneficiation, and the demand for metals of certain alloys may also influence the amount of mineral to be won and the use to which it is to be applied. It may then be questioned whether, in providing supplies of iron ores, one is justified in assuming that a specific deposit or series of deposits may be the dependence for an industry far in advance.

This paper could be extended by instancing individual deposits of iron ore or specific industrial installations as proof of the general conclusions presented, but if it invites attention to what may be classed errors in prognosticating reserves, or it awakens discussion upon future changes which may be expected, its purpose will have been served.

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The discussion has been confined to the iron-ore reserves of the United States or those which have been estimated as available to supplement the domestic supply and while there has been no attempt to consider the iron-ore deposits of the world, it is believed the sufficiency of these may also be demonstrated.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Maritime Features of the "Crude Petroleum" Problem

BY REAR ADMIRAL JOHN R. EDWARDS, U. S. NAVY, WASHINGTON, D. C.

(Pittsburg Meeting, October, 1914)

*Introductory.*—There are many interesting and important events connected with the petroleum problem. The remarkable men who conceived the thought of transporting petroleum by pipe line, conserving the by-products of the commodity, and conducting extended investigation and research whereby the flash point of the oil was raised, its cost reduced and its supply assured, are deserving of the gratitude of this nation.

It was primarily the success attained by Captain Lucas, after his comprehensive search for oil in Texas and Louisiana, that was the impelling cause which prompted the Navy Department to conduct the extensive series of liquid fuel-oil tests that were made from 1901 to 1903. The sensational discovery of oil at Spindle Top made a deep impression upon Rear Admiral George W. Melville, then Engineer-in-Chief of the Navy, and the discovery of the vast oil field in Texas convinced him that, within a generation, if not within a decade, crude petroleum was going to play a very important part in warfare, and that the commodity might possibly develop into a necessary munition of war.

*The United States' Commanding Position as a Petroleum Producer.*—In the study of the various problems relating to the world's production of crude petroleum, the position of the United States is one of commanding importance, whether viewed from the industrial or the military aspect.

The present yield of this country is practically double that of the rest of the world, and, in view of the productive and prospective fields possessed by various countries, together with the comprehensive and exhaustive search that has been made throughout the world for petroleum, it is extremely probable that this lead will be maintained for the next decade.

As regards the character of the crude-oil yield of different countries, America possesses the greater part of the supply that contains a paraffine base, and therefore a considerable portion at least of the American product produces the best illuminant. Of all the crude-oil distillates, kerosene is in many respects the most important, and the Appalachian product for this purpose is favored, particularly in the Far East, above that of any other petroleum yield of the world.

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We are undoubtedly in advance of any other nation in the extent and character of our prospecting and drilling facilities, and this is attested by the fact that American experts are found in nearly all the other oil districts of the world, directing the activities of production.

In methods of refining our position is on a parity with others, if not a commanding one. We have been able to obtain from the crude product various distillates of as high character, and at as low a cost, as can be produced by the foreign refineries. New and important uses are being found for the by-products of crude oil, and our discoveries in this direction may undoubtedly be regarded as important as, if not superior to, those of our rivals.

In methods of land transportation our lead is undisputed. This should not be surprising, considering the extended and diversified experience that we have had in the construction of pipe lines, pumping stations, and containing tanks, as compared with the relatively limited experience of the rest of the world.

*The United States' Weakness as Regards Ocean Transportation.*—There is one important feature of the fuel-oil problem, however, wherein we are lamentably weak, and that is the maritime distribution of the product from the terminals of the pipe lines to various important seaports of our own country, as well as to ports abroad.

The modern sea-going tankers that we possess are few in number as compared with those that fly the ensign of Great Britain. Our failure to convey or distribute crude oil in as efficient a manner as we produce, refine, and transport the product, constitutes a serious industrial reproach to our commercial abilities. When viewed from a military standpoint, our lamentable dearth of ocean tankers and sea-going barges constitutes a great military weakness.

*Our Lack of Sea-Going Oil Craft Entails Heavy Direct and Indirect Loss to American Industrial Activities.*—It appears incongruous that a British corporation should find it a profitable venture to spend millions of dollars in prospecting for oil in Mexico, Ecuador, Colombia, and other Central and South American republics, when our own capitalists ought logically to undertake the practically exclusive development of those fields.

The British corporations engaged in petroleum development cannot possibly have the intimate professional knowledge and extended practical experience that is possessed by our own experts. Our banking interests are likewise undoubtedly as ready to advance capital in the exploitation of legitimate fuel-oil enterprises as the financial interests of England. It appears evident that it is only because the American maritime feature of the fuel-oil problem is so unsatisfactory that it has been possible for the Pearson Corporation in Mexico to compete against the American interests in that country.

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As illustrative of the deplorable condition of affairs as regards the maritime feature of the crude-petroleum problem, it is pertinent to call attention to the fact that, when the Liquid Fuel Board of the Navy was conducting its extended series of fuel-oil tests from 1901 to 1903, an investigation that extended over a period of 30 months, and an investigation wherein there were expended at least a quarter of a million dollars, there were several periods when the President of the Broad was informed that, for two or three days at least, there was not one single American tanker carrying fuel oil between the Texas terminal ports and the leading seaports on the north Atlantic coast. Careful investigation was made to determine if such a condition of affairs actually existed, and, from the best information obtainable, there was one period, if not two, when it happened that every American sea-going tanker was either temporarily disabled, detained in port, or had met with some delay so that no oil vessel was steaming between the Gulf and north Atlantic ports.

It is possible that, at these two particular periods, one or two such tankers may have been proceeding with empty compartments to the Gulf for a cargo of oil. It may be stated as a fact, however, that in 1901 or 1902 a situation arose whereby, for at least a few days, not one gallon of crude oil was being transported by water from Texas to northern ports. There were, of course, during this special period, considerable quantities of oil being transported in barges from the terminal pipe lines and refineries at Baltimore, Point Breeze, Marcus Hook, and Bayonne to other points on the Atlantic.

As bearing upon the question of the world's dearth of oil tankers, it may be well to call attention to the fact that, about 18 months ago, the Paymaster-General of the Navy made an effort to find out the number and character of the tankers that could be chartered at short notice by the government. It was found at that time that it would not be possible to secure at short notice one single American tanker on the Atlantic coast; every tanker flying the American flag being required to convey the oil that was used for industrial and other purposes by regular consumers.

A still greater surprise was experienced when it was found that but one British tanker could be procured at that period, and that this vessel could only be chartered by the Navy Department by the payment of a rental of $575 per day, which rental did not include fuel and port charges. This rental had to be paid from the date the British tanker was chartered in London, and was to continue until the return of the vessel to the same port.

It seems surprising, if not astounding in view of the fact that we not only surpass the world as regards the yield of the crude-oil product, but that likewise we possess advanced methods of boring, improved



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processes of distilling, and incomparable methods of land transportation that the maritime feature of the oil industry should have been neglected by this nation.

*Profits of the Oil Industry have been in the Refining and Transportation of the Product.*—In the study of the extensive and far-reaching report of the Commissioner of Corporations, as contained in the government's brief against the Standard Oil Co., it is observed that the principal profits of that corporation have been in the refining and the transportation of the oil, and not in prospecting and drilling for the product. The distribution agencies, however, have been inadequate, except so far as they concerned districts fairly contiguous to pipe lines, refineries, and deep-water terminal points. At such particular points, which must necessarily be limited in number as compared with the rest of the country, the distributing facilities have been efficient, considering the expense, difficulty, and hindrances encountered in distributing all inflammable hydrocarbon products. There is no doubt, however, that every interest concerned either in prospecting, boring, refining, or distributing crude oil has been very materially injured by our failure to control the distribution by sea.

*The Importance of Petroleum as a Factor in Extending Our Foreign Market.*—Fuel oil has played a very important part in extending our trade with China. The writer, in the course of his duties as a naval officer, spent about 6 years in Asiatic waters, and interviewed various American and British Consuls, as well as the leading merchants of the China coast, in reference to the far-reaching influence of the petroleum industry. The study of our trade development with China must cause one to be impressed with the fact that the three distinguishing agencies which have done most to cause the Land of Sinim to abandon many of its traditions and much of its medievalism have been the missionaries, modern surgery, and the work of the Standard Oil Co. In distributing throughout that country an illuminant that has been a direct boon to the people, the Standard Oil Co. has exerted an indirect influence that has been of far-reaching consequence in extending our trade relations with that Empire.

Light is a great civilizer, and the remarkably convenient, economical, and safe manner in which the Standard Oil product was distributed, even to the wilds of Hunan, was the forerunner of the entry in large quantities of other American products.

In the distribution of refined petroleum in cases rather than in bulk, our country attained for some time considerable success in the employment of sailing vessels. We lost, however, a great opportunity in not effecting legislation that would have retained to us this trade by regulating the export of the product, and that would have given to this nation a paramount control of the distribution of petroleum.

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The writer was on the China coast about the period when the Russian oil interests, supported by the Rothschilds and other great banking interests of Europe, attempted to wrest the control of the petroleum industry from America. Modern steel oil tankers were built for the Rothschilds for this purpose, and large containing tanks were erected on shore at various points on the China coast. It was found, however, that the American product was so superior in quality as an illuminant, and it was so well contained and boxed, that it was practically impossible to supersede its use by the substitution of the Russian product.

It was not by a haphazard guess or by accident that the existing 5-gal. rectangular petroleum case was constructed. The form of the containing case, the material from which it is manufactured, the character of the soldering and the method of boxing and shipping constituted a development that represented heavy expenditure, extended experimentation, and practical knowledge upon the part of the Standard Oil experts.

In distributing this product, therefore, even unto the uttermost portions of the Chinese Empire, the Standard Oil Co. rendered a service of incalculable benefit, both to China and to America. If it had not been for the energy, efficiency, and tact displayed in developing our petroleum export trade, many looms in New England, and many furnaces in the Appalachians that are now finding it a profitable venture to export their output to the Celestial Empire would be idle to-day.

It is believed that the American export trade of the future to Central and South America, Australia, and particularly to China, is greatly dependent upon the distribution or maritime feature of the fuel-oil problem. This phase of the oil problem is therefore a matter that is specially worthy of the consideration of the American Institute of Mining Engineers, for it is the mining engineer who is most intimately interested in the primary feature of the oil industry—and that is the prospecting phase.

There is something lamentable in the thought that, despite the fact that this country has a commanding lead in all but one of the various phases of the petroleum problem, America should fall down at the last stage, and that we should see our British cousins, through their superiority as regards the maritime feature, wresting from us a maritime activity of great value—an activity that logically should be possessed by America.

*The Military Importance of the Maritime Phase.*—It is not opportune at this time to speak of the military features of the distribution problem, since there are phases of the matter that intimately concern national defense. The fact is well known, however, to British naval experts, and mention has been made of the subject in the English press, that our navy is lamentably lacking in the oil tankers that will be required to supply American battleships and other naval vessels that have been designed

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to burn oil exclusively. The various publications of the Congress that are distributed both at home and abroad, tell of the number and character of American naval vessels that have been designed for an exclusively oil-fuel installation, and therefore our weakness as regards the number and character of the merchant and naval oil tankers that we possess must be well known to foreign military experts. The various publications issued by the government even tell, in detail, of the tonnage, cargo capacity, and other important structural features of these ships.

*Our Inadequte Oil Reserve at all Our Industrial Centers in Considerable Part due to Our Maritime Weakness.*—The maritime feature is an important one, from whatever standpoint the question is considered. There are hundreds of industries in this country which would substitute petroleum for coal if an assured supply of the former combustible could be maintained. There are hundreds of industries that have experimented with petroleum with the intention of using it exclusively as a fuel, that have been compelled to return to coal, due to the impossibility of obtaining an assured supply of oil.

The cost and risk of transporting oil by cars constitute an insuperable bar to the distribution of the product in sufficient quantities to insure an adequate supply for large industries. Crude oil will, however, always be used for purposes wherein a substitute cannot be found, independent of the question of cost. It is, in fact, our dearth of water-transportation facilities that prevents the distribution of the product, and therefore limits the use of petroleum. There is no doubt that there are many important industries which, with the use of oil, would produce special articles that would find a ready and profitable sale in foreign countries, if they could be guaranteed that they would be able to get all the oil they required. It is well known that petroleum is an incomparable fuel for many industrial purposes, but no manufacturer dares to use the product except to a limited degree, unless his plant is so located that he can be assured of an adequate supply at other than a prohibitive cost.

*Should not an Export Duty be Placed upon Petroleum Products when Carried in Foreign Bottoms?*—When it is considered that the United States furnishes about two-thirds of the world's yield of crude oil, and that it is somewhat probable that, at least in this country, the maximum of annual production has nearly been reached, unless there are vast undiscovered pools of oil at lower depths than we are now able to drill, it ought to be possible for this nation to so regulate the export of the product as to conserve the industry in a manner that will best help ourselves, if not mankind. There are undoubtedly in the United States a sufficient number of steamship companies, railroads, and individual firms that would use all the oil produced in this country, if such plants could be assured of an adequate and reliable supply, and therefore the foreign export of oil may not be necessary in the disposal of the apparently large quantities produced.

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In many industries the cost of the product is not a serious matter as compared with the particularly beneficial and industrial results that could be expected to accrue. It is all-important, however, in every enterprise, that, once an industry adopts a special form of fuel, such plants should be assured that an adequate supply of that fuel could be promptly obtained

Many of the distillates of crude oil should even be regarded as munitions of war, and legislation should be enacted that would make it possible for us to prevent the export of oil at such times as we deemed advisable. Except for a brief period, it is extremely probable that, if we placed an export duty on petroleum, no permanent financial injury would accure to any one interested in the drilling, refining, transportation, and distribution of the product. The writer has been told repeatedly, during the past 10 years, by various manufacturers and individuals, that it was solely the lack of assurance than an adequate supply of crude oil would be available that prevented them from using the product.

Probably no better way of helping to give the American merchant marine a reasonable share of the foreign oil-carrying trade could be devised than by placing an export duty on every barrel of oil exported from the United States in ships carrying a foreign flag. We are one nation that is unreservedly adopting the policy of fitting our battleships with an exclusively oil-fuel installation, and this installation is of such a character that, except at enormous expense, the structural change to coal-burning arrangements cannot be brought about. One of the most effective ways, therefore, of providing a supply at various points for naval purposes is to conserve the petroleum industry, by taxing the export of the product and by exercising the prerogative that logically belongs to us in controlling, to a limited degree, the sale of the commodity.

International comity and the law of nations sanction the policy of a nation placing an export duty on a commodity that it substantially controls. We are therefore justified in taking some radical means to control the maritime phase of the matter.

In the consideration of the problem of imposing an export duty upon petroleum, there are several phases of the question that merit thoughtful consideration and investigation. If serious thought is ever to be entertained of the proposition of imposing an export duty upon crude oil, the question may well be asked: "How will it affect our trade relations with China? Is the American illuminant of such superior quality that the Chinese would pay the additional cost that would be the result of such action? Have the foreign competitors of the Standard Oil Co. effected such improvements as regards refining the product that they can now furnish as good an illuminant as that which can be obtained from the paraffine-base product of the Appalachian region? Are our foreign competitors already making serious inroads into the sale of the American product in various countries? Would the loss of the petroleum trade of

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the foreign countries, and particularly the petroleum trade of China, be followed by a marked reduction of other American imports into those countries?" The question of an export duty is therefore one which admits of a wide divergence of opinion.

*Conservation of Our Oil Resources.*—The conservation of our oil industry can only be brought about by some national organization like the American Institute of Mining Engineers taking advanced ground upon the subject, and urging that early and thoughtful action be taken in the matter. Various individual officials of the government, as well as individual experts, have urged such a course, but such individuals do not, in many respects, possess the machinery for being the proper central agency in bringing about such important action. The influence of the American Institute of Mining Engineers extends throughout the country, and, if it should recommend some thoughtful line of policy in the matter, there is no doubt that various commercial exchanges and maritime associations of the country would supplement the work of this Institute.

There is no doubt that the high purpose and urgent necessity of conserving the petroleum industry can be brought about in various ways, but the more study one gives to the matter the more impressed he becomes with the fact that, in the control of the maritime phase and in placing an export duty on oil shipped in foreign bottoms, it ought to be possible to do much, not only in conserving the industry, but in helping to bring about the restoration of the American Merchant Marine.

*Efficiency of Our Oil Tank-Car Facilities as an Agency in the Distribution of Petroleum Products.*—By reason of the wide extent of our country, the diversified character of its industries, and the limited number of existing oil pipe lines, the greater portion of the distribution of the crude-oil product will have to be done by cars. The existing design and arrangement of these cars represent extended and thoughtful development. The facilities for the safe and economical storage and handling of petroleum products are undoubtedly in many respects of the most efficient nature, and there is no doubt that all persons connected with the crude-oil industry are exceedingly receptive of counsel and advice that will improve the oil tank-car arrangements.

When there are considered the insurance and regulations, the danger attendant upon the transportation of oil by cars, together with the limited facilities for the storage of such combustibles in our various industrial cities, it becomes a question of paramount importance that wherever possible the car facilities should be supplemented in some manner, and that a greater distribution of the oil product can be brought about by barge or through some other maritime agency. However great may have been the shortcomings of those controlling the petroleum industry, there is no doubt that, so far as tank-car equipment and storage facilities are concerned, an exceedingly earnest effort has been made by

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the oil corporations to bring an adequate amount to the various industrial cities, and to store it in as convenient a location as possible. The oil-distributing companies, however, have been confronted with the problem that, at every important industrial center, there exists not only a dearth of oil tank cars, but a lack of freight-handling facilities. It has not therefore been possible, by reason of existing railroad terminal facilities, to give preference to the distribution and handling of crude oil when the railroad companies are confronted with the more serious and more important problem of handling the daily food supply of these great municipalities; and this, in general, is one of the reasons why manufacturers cannot get the oil they need.

*Our Lack of Merchant Marine a Serious Bar to the Development of the Crude-Oil Industry.*—One can well understand that it has been primarily due to the lack of suitable oil barges and tankers that makes the seagoing distribution, or maritime, phase of the problem a matter that concerns our industrial life. Under existing conditions the transportation of oil in American vessels to foreign ports is substantially a venture impossible of profit. On the Pacific coast it may now be possible; but, with the development of the Mexican fields and the opening of the Panama Canal, we may lose even that trade. The cost of operation of American-manned ships has been so great, as compared with the operation of foreign ships, that it is certain we cannot hold any foreign trade, and therefore it has fallen into the hands of our foreign rivals. The indirect financial loss to this nation in the loss of the petroleum export trade has undoubtedly been of vast extent. American crude petroleum is one of the products that is favorably received, if not eagerly sought, in foreign countries, and therefore the control of this special trade is of far-reaching importance to our manufacturing interests at large. Following the importation of petroleum products into foreign countries, there has almost invariably been a progressive increase of imports of other American articles to such countries. It therefore appears that, if there is one industry that needed the support of the government, or even that should have been heavily subsidized, it is that relating to the carrying of crude oil in American-registered ships.

*Nation's Reserve Stock of Crude Petroleum.*—It is significant to note that the petroleum stocks on hand in the United States at the close of 1913 approached about 130,000,000 barrels, a quantity that would not meet the nation's existing consumption for a period exceeding eight months. On Jan. 1, 1913, the reserve stock was actually less than that on hand Jan. 1, 1912. During the past year, however, there has been a slight increase in the reserve stock, although drilling for the product was stimulated in every State in the Union as a result of the increased price obtained for substantially every distillate and by-product of crude oil. It is of exceeding commercial, if not military, interest, likewise, to

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remember that the yield for 1913 from every oil-producing State except California and Oklahoma was less than the yield of some previous year, despite the fact that progressively increased prices were obtained for the oil in every district of the country, and that oil fields which had been abandoned in previous years are now able, under existing conditions, to yield a profitable return upon the capital invested.

A very distinguished authority upon the California product has recently declared that, despite the continuous and careful research that has been made for oil in that commonwealth, the possible productive area of that State appears to be about 2,000 acres beyond that established in 1909, when Secretary Garfield, then the executive head of the Department of the Interior, ordered a survey to be made. He furthermore said that in all probability, nearly every crude-oil basin within that State had been located, and that there would be a constant drain and reduction in the reserve stocks that could be maintained there. In fact, another expert has asserted that, due to water intrusion and certain wasteful methods of production practised in California, this country should be prepared to note a startling slump in the oil production of California. The wasteful manner in which drilling is being carried on in that State, taken in connection with the fact that the most important wells can henceforth be expected to show a progressively decreased supply of the product, affords some ground for the belief that the curve of production in that State may suddenly drop like the hump on the back of a camel.

As regards the thousands of miles of productive fields that are supposed to exist in Mexico, the United States Geological Survey, in its Advance Bulletin for 1914, states that, despite the phenomenal energy of crude-oil exploration work in Mexico, the fact remains that so far as the present supply is concerned the Mexican yield has been practically limited to a few large wells, and not to a very large number of smaller wells. It is questionable, also, whether many new wells are likely to be drilled under the peculiar conditions that exist in Mexico at present, and that may possibly exist for years, as regards shipments. It is extremely probable that the supply will not be sufficient to meet the demands of the numerous large-sized tankers that have been contracted for to carry the product from Mexico to England. It must further be considered that, with greater confidence developed as regards the Mexican supply, the price per barrel may increase rather than be reduced, because an assured supply from that country will not only stimulate refining, but tend to increase the oil consumption throughout the world. At present the Mexican yield is but 5 per cent. of the total yield of the world, and there is substantial evidence that most of the promises as regards Mexican development are not likely to turn into performances. It is even going to be for some years quite a serious matter to safeguard the Mexican wells from destruction at the hands of brigands and insurrectos.

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*The Military, if not the Industrial, Necessity of Installing Additional Pipe Lines to Certain Points on the Atlantic Coast.*—By reason of the fact that the vicinities of Cape Hatteras and Cape Cod constitute certain storm centers, the insurance companies of the United States charge additional rates on ships that pass near these storm points. These additional rates of insurance likewise apply to the cargoes carried by the vessels. While the rates for approved sailing-ship transportation are less than those demanded for approved steamship transportation, the fact remains that all classes of vessels, whether steam or sail, are made to pay an extra insurance charge of about 10 per cent. when plying within range of the lighthouses of Hatteras and Nantucket.

There are undoubtedly urgent military reasons, if there is not an industrial need, that the leading transporting oil companies should extend their pipe lines to some point on the Atlantic coast south of Hatteras, and to another point either on Narragansett or Massachusetts bay. An oil pipe line leading to New England ought to prove a paying venture at least in the course of a few years, by reason of the increased quantity that would be consumed there, and the higher prices that would be willingly paid by the manufacturers if they could be assured of an adequate and reliable supply.

In the manufacture of illuminating gas, crude oil is an important and essential constituent, and it ought to be possible, without in any manner impairing the financial interests of those engaged in oil transportation, to make the New England project a desirable investment. The number and character of the industrial cities between Bayonne and Boston, and the possible extent to which crude oil could be used in the industries of those cities, ought to warrant the building of a New England oil pipe line.

However numerous and however large may be the containing tanks that may be installed, but which must be supplied either by tank cars or by sea-going barge or ship, there will be periods, as long as we depend on tank cars and due to the railroad terminal conditions at all our leading industrial centers, when the reserve stock will be of such a limited nature as to prevent certain users from obtaining an adequate supply; and it is for this reason that there appears to be an industrial necessity for a New England oil pipe line.

The construction of a pipe line from the West Virginia fields to some port south of Hatteras—a port which possesses the deep-water facilities of Charleston, is a national necessity, and if necessary its construction should be subsidized by the government. The far-reaching influence of providing an ample supply of oil to naval vessels alone cannot be overestimated, and a pipe line to some point on the Atlantic coast south of Hatteras may eventually develop into a military necessity. Our sea-going battleships that are fitted with exclusively oil-fuel installations, together with numerous destroyers that burn oil exclusively, require

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terminal pipe line facilities at some point south of Cape Hatteras, and the construction of a pipe line from the West Virginia fields to Charleston ought to be given early consideration. Fortunately for the interests of the Navy, there are several oil pipe lines leading to Gulf ports.

*The General Problem of the Conservation of Petroleum Should be Considered by a National Commission of Experts, Representative of all Interests Concerned in the Question.*—Neither the direct value nor the indirect worth of the ocean oil-carrying trade to a nation may be within our power to estimate. The far-reaching importance, however, of the maritime feature of the petroleum problem is best evidenced and manifested by the fact that, although the petroleum yield of the British possessions does not comprise even 1 per cent. of the world's production, there was appointed on July 30, 1912, by the British Ministry, a Royal Commission on Oil Fuel for the Navy. This Commission was headed by the ablest naval officer, in some respects, that has been borne on the Navy List of England since the days of Nelson.

In order to demonstrate the importance which Great Britain attached to the work of the Commission, the personnel of that body included the Director of Naval Construction, the Engineer-in-Chief of the Navy, representatives of the shipbuilding firms and shipping interests, together with representatives of every important interest concerned in the extension and development of the petroleum industry.

While this Commission has made several confidential *ad interim* or preliminary reports, it is extremely important to note that this distinguished array of experts has not yet found it possible to submit its final report to the British Ministry. These experts evidently believe that the more study that is given the matter, the more important appear the possibilities.

If there were impelling reasons upon the part of the British Ministry to appoint such a Commission, although England's distinctive industrial interest in the matter is primarily based upon the distribution feature of the oil product, it would appear much more important that this country should give equal consideration to the problem. In fact, however, it is the military possibilities of the problem that concern Great Britain; and should we not be more concerned as to this phase of the problem?

The Liquid Fuel Board of the Navy, a Board which carried on the most exhaustive series of tests that have ever been conducted in determining the relative value of coal and oil as a fuel, made the following recommendation in submitting its report in 1903. It is pertinent to state that these tests continued over a period of 30 months, and that the direct and indirect expenditure involved in the work exceeded a quarter of a million dollars.

"In view of the fact that 48 per cent. of the world's output of crude petroleum is produced in the United States (it is now 63 per cent.), and that practically our entire

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yield is secured from fields which are in pipe-line communication with important maritime and strategic ports, the Board considers that a joint commission, representing commercial, manufacturing, and maritime and naval interests, should be authorized by the Congress, whose provisions it would be to formulate such rules and regulations as would provide for the economical, efficient, enduring, and safe fuel installation."

Far-reaching as may have been the above recommendation, there have been such important developments in the petroleum problem during the past 10 years that it now appears more imperative than ever before that such a commission should be appointed. Its duties should be of a much more extensive nature than recommended by the Liquid Fuel Board of the Navy. Its scope of inquiry should include every feature of development from prospecting for the product to the distribution of the commodity. Particularly should its work include the important problem of conserving this great industry to the benefit of our commerce and the defense of this nation.

This meeting of the American Institute of Mining Engineers affords an opportune time for giving some serious consideration to the question of conserving the petroleum industry, and to the appointment of a national Commission that would outline the nature of the work that should be entrusted to such a body.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Coal-Dust Explosion Investigations

BY M. J. TAFFANEL,* LIEVIN, FRANCE

(Pittsburg Meeting, October, 1914)

I AM very much impressed by this manifestation of international brotherhood; the mining engineers on both sides of the ocean have similar subjects to deal with, meet with the same difficulties, expose themselves sometimes to similar dangers; they are like companions in arms in the battle fought by mankind in order to subjugate the natural forces and to draw profit therefrom. There is no other field where this fraternity must manifest itself more actively than in the studies pursued with a view to improving the safety of mines and to rendering impossible in the future these mining disasters, which every year claim hundreds of victims. These are very difficult problems, the solution of which requires numerous and laborious experiments accompanied by very arduous scientific investigations; it is extremely desirable that the various mining countries, which have undertaken to solve these difficult problems, should unite their efforts in order to succeed as soon as possible and as well as possible. This idea has been fully grasped by your Bureau of Mines, which has brought me to Pittsburg; the daily exchange of views with the distinguished chief of the mine investigation division, George S. Rice, and with his excellent collaborators, the comparison of the results obtained at Lievin and Bruceton, the checking of these results by means of new tests, the profound study of the measurements, diagrams, measuring and laboratory apparatus, the cooperative investigation of new apparatus, or the preparation of new investigations, are, I hope, of such a nature as to bring about great progress.

I think that it will be interesting to you to know what has been done in France along this line.

It was the Courriere disaster which led in France, as well as in other countries, to the undertaking of experiments on a large scale with a view to investigating the danger of coal dusts. We have had in France, prior to 1891, numerous explosions in the mines; but beginning from that time, great progress has been made in the fight against firedamp in all the

*Non-member; Director of the Government Experiment Station, Lievin, France.

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mines where firedamp was found, even if in exceptionally large quantities. To meet this end mine ventilation was considerably improved by modifying, when necessary, the method of working and by substituting, for instance, the system of complete filling of the mine excavations for that of partial filling or none at all; use was made of improved safety lamps, particularly of the double-gauze and bonneted Marsaut type; the use of black powder for firing shots in the coal was forbidden; the escape of firedamp was observed with greatest attention by making precise gas analyses every week and sometimes every day in all the splits of the air current. Thanks to these measurements, which have already been made obligatory by the regulation of 1895, not a single large explosion took place between 1891 and 1906, and the proportion of miners killed remained very low, at about one per thousand annually.

But the development of the mines, the deepening of the workings, which became warmer and drier, the increase of the ventilation which dried up the galleries, and many other causes, have increased the dust danger. On March 10, 1906, at the Courriere collieries, which were absolutely free from firedamp, an explosion occurred which devastated the workings of one pit and part of the workings of two neighboring and connected mines and caused the loss of 1,099 lives. This disaster, the greatest that ever occurred in the mines of the world, has demonstrated in an indisputable manner the reality of the coal-dust danger.

Thus was shown in France the imperative necessity of studying this danger and of investigating the means of fighting it. Contrary to what happened in the United States, it was not the government, but the Coal Owners Mining Association of France, that took the initiative in making these investigations and experiments, establishing the station of Lievin, and furnishing all the necessary funds. However, this initiative had the full approval of the Government, and led to my appointment as Director, having formerly been a government mining engineer. Further, an administrative committee, the firedamp committee, appointed by the government, keeps in touch with the experiments and ratifies the results.

The first experiments of the experimental station of Lievin were made in 1907 on a restricted scale; but showed that the various kinds of coal dusts had quite different inflammabilities, the most inflammable having the largest proportion of volatile matter, and that by mixing a sufficient amount of stone dust with the coal dust the latter could be made non-inflammable.

From year to year the investigations have been continued on an ever larger scale. At present the testing gallery is 1,000 ft. long and has, at a distance of 750 ft. from the origin, a side branch of 230 ft. The gallery is entirely on the surface; the first 90 ft. had been made of reinforced concrete in order to have a rectangular cross-section; the following portion up to 600 ft. from the origin is circular in cross-section, in order to

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offer the best form for increasing pressures. It is a steel tube 6 ft. in diameter and $\frac{3}{8}$ in. thick. In one instance an extremely violent dust explosion generated in the last 30 ft. a pressure of about 280 lb. per square inch and caused the destruction of this part of the gallery. The extensions to 1,000 ft. have been constructed of special steel plates of $\frac{3}{4}$, $\frac{7}{8}$ and 1 in. thickness.

A branch tube connects the gallery with the ventilator, which is used for driving out the smoke after the explosion, and a trap-door protects the ventilator during the majority of the tests; it has been found, in fact, that even if the ventilation should have any slight influence upon the initial phase of the explosion, it does not have any influence on the propagation.

The experimental station also has a smaller steel gallery for the investigation of explosives. The two galleries are so arranged as to receive mixtures of firedamp and air in explosive proportions in an explosive chamber, or with a low content of firedamp in their entire length.

There are several laboratories, one of which is specially devoted to the study of explosives; in addition to various apparatus met with in the majority of laboratories of this kind, there is an extremely rapid cinematograph established last year by the Lievin experimental station, which gives a complete image of the flame of the explosion for every $\frac{1}{1000}$ sec.

The dust explosions produced in the large gallery are investigated with the help of numerous apparatus, chronographs, manometers, gas-sampling bottles, etc., which it would take too much time to describe here; I shall confine myself to explaining a diagram obtained with one of these apparatus. One of the curves, recorded photographically, gives the variation of the pressure as a function of time at the point where the apparatus was located; the first point to be noted is the arrival, at the manometer, of the shock wave produced by the detonation of the explosive; the pressure is maintained by the combustion of the dusts and rises in proportion as the explosion approaches a maximum. The passage of the flame is photographically recorded on the same film; its passage lasts but a fraction of a second. Another curve gives the velocity of the air; immediately after the passage of the shock wave the air is set in motion in the direction of the explosion, running from the center of combustion; this is the "pioneering" wave which raises the dusts; this first velocity amounts to from 50 to 100 ft. per second under the conditions of this test, but it rises rapidly at the same time as the pressure, and attains a maximum at the moment of passage of the flame; this passage, which takes place a little sooner toward the center of the gallery then on the wall, where the recording is done, has as its effect to reduce considerably the velocity of the air and often during violent explosions to change their direction; the reason thereof is to be found in the increase of volume of

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the gases in consequence of the combustion; there is, on the one hand, a blast in front of the flame, waves which maintain and reinforce the pioneering wave, and, on the other hand, a recoil of the burned gases toward the regions where the explosion was less violent and the pressure was lower; this reversion of the direction of the movement of the gases during the passage of the flame is sometimes sufficiently strong to give great dynamic effects in a direction opposite to the direction of the explosion; this explains the contradictions observed sometimes in the investigations in consequence of mine accidents.

An artificial gallery, like the one at Lievin, is easy to clean and to prepare, and allows of performing numerous tests at any season; in certain series of tests we have been able to make one test a day; the total number of tests in the large gallery exceeds at present 1,400. We have, therefore, collected a large number of results. We have investigated the laws of development of the dust explosions, which enabled me to establish in 1910 the theory of explosions based on the laws of combustion and of the dynamics of fluids. Numerous series of tests have been made with the dustless or watered zones, or with the pure-dust zones, or with zones containing variable proportions of stone and coal dust; these zones of from 300 to 600 ft. in length prove to be incapable of stopping a violent explosion. But I succeeded in 1909 and 1910 in stopping explosions, even the most violent, by means of arresting barriers where large masses of water, or of stone dust are accumulated at the point where the combustion is to be stopped; these masses are set in action by the pioneering wave. During the last 1½ years, in consequence of an explosion that occurred in the Clarence mine in Northern France, which seems to have propagated itself with exceptional slowness, I have been trying to improve the arresting barriers by increasing the quantity of the accumulated extinguishing materials, by securing their setting in action even by weak pioneering waves, and by making their discharge last sufficiently long in the case when the flame arrives several seconds late. I arrived last year at a solution which showed itself to be efficacious by means of a slow-discharging water trough or tank. I have been experimenting for several months on concentrated stone-dust receptacles, which so far have given me good results and will undoubtedly be soon ready to be adopted in practice.

Some hundred tests made between 1908 and 1909 and more than 500 tests made more recently, had as their object to investigate the relative capacity for propagation of various coal dusts mixed with different proportions of stone dust and deposited in variable quantities in the gallery, in the presence of various initial explosions, with or without watering, with or without the presence of various firedamp contents in the atmosphere of the gallery.

At the same time the inflammability of these dust mixtures was

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measured by means of a laboratory apparatus which was thus calibrated for the gallery tests.

I have thus arrived at an empirical law which, with the help of the "inflammator," a laboratory inflammability-testing apparatus, gives the relative degree of various dust deposits, according to the value of the multiple factors of the problem.

But there is a factor whose influence it is difficult to determine by means of tests made in a single gallery; it is the influence of the gallery itself, of its sectional area, of the nature of its walls, of the arrangement of the orifices, blind ends, ramifications; the theory indicates that this influence must be considerable. It is therefore essential to make comparative tests in other galleries and to check the results of the artificial gallery by means of tests made in a real mine; only tests made in an experimental mine, like the Bureau's experimental mine at Bruceton, Pa., can be considered as giving results that can be immediately applied to practice. In France we were impressed by this question about two years ago, and I was able to profit last year by the abandonment of a gallery in the Commentry mine, where I made 16 tests which gave results that agree fairly well with the tentative results obtained at Lievin. They showed the efficacy of the arresting barriers and of the new water tanks, and they also showed how bends or turns in the passageways favored the stopping of explosions. The tests were stopped by the destruction of the gallery and by the intrusion of water. In three months I shall have at my disposal a new experimental field in the workings of the Montvicq mine, which will soon be abandoned.

An ideal experimental mine, particularly because it is well equipped with recording apparatus of all kinds and is perfectly adapted to the experimental requirements, is the Bruceton mine, whose tests the Bureau of Mines has kindly permitted me to keep in touch with. I have been extremely interested in the study of the important results obtained so far; I have seen with the most cordial satisfaction the advances made by Mr. Rice. By means of investigations parallel to those pursued at Lievin he has greatly improved the method of stopping explosions, introduced new ideas and made arrangements which seem to be both very practical and very efficacious.

It is not sufficient merely to theorize about the coal-dust danger and the means of overcoming it; it is necessary to introduce in a practical way what is suggested by the experiments.

The majority of the dusty mines of France have made great advances along this line in the course of the last years. The first precautions, which have been taken and which are the most important, are those whose object it is to suppress or prevent the initial causes of inflammation: these are in the first place all the measures which I have indicated in the first part of my remarks relative to the firedamp danger; also, the exten-



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sion of the safety explosives to the dusty mines, the suppression in these mines of lamps with naked flames, the assignment of special employees to the shot firing, the recommendation of firing the shots preferably after or between shifts. The most important point after that consists in rendering the dust deposits as little inflammable as possible over the entire length of the haulage roadways and of the principal arteries of the mine; this neutralization of the dust is rarely done through watering, almost always through removal of the dust followed by stone or rock dusting. Practice has shown that, on condition that tight cars are used, the result of such an operation is to maintain the proportion of ash above 60 or even above 70 per cent. for two, three months, or sometimes longer; if the inflammability tests which are made regularly in the majority of the dusty mines show that the ash content must be maintained above 65 per cent.' the operation must be renewed almost every three months; with less inflammable coals and a less elevated limit of the ash content, the operations of neutralization could be made at longer intervals.

Finally, as a third precaution, the mine sections are separated by arresting barriers; about 2,000 of these devices are actually installed in the French mines.

Such is the organization against the dust danger, which is on the way of being put into practice in all the dusty mines of France, and which has already been put into practice where the nature of the dusts or the presence of firedamp rendered these measures more necessary.

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# Refining Petroleum by Liquefied Sulphur Dioxide

BY DR. L. EDELEANU,

(Pittsburg Meeting, October, 1914)

CRUDE petroleum is a mixture of various groups of hydrocarbons and some bodies containing oxygen or sulphur. These constituents possess properties differing considerably one from another and the proportion in which they are contained in the crude oils is also very different.

Owing to this complicated composition the practical use of crude petroleum is much restricted.

For example it is sometimes used as a liquid fuel. But its employment in this way is uneconomical because various valuable qualities of the individual constituents are not made use of.

The rational utilization of petroleum can only be accomplished if the various groups of hydrocarbons are separated. The mixture of hydrocarbons present in the crude oil must be split up in such a way that the hydrocarbons are divided into various groups having certain predominant characteristics and uses. To accomplish such a separation is the aim of the technical refining of petroleum.

## ORDINARY REFINING BY DISTILLATION AND CHEMICAL TREATMENT

Hitherto this has been accomplished by two processes, namely, fractional distillation and chemical treatment of the distillates obtained.

### *Distillation Process*

The process of distillation consists in the separation of the constituents of petroleum into various fractions, the boiling points of which lie within certain defined limits. The fractions so separated possess certain special qualities in common, which enable them to be applied for specific purposes in practice. For example, we have the fractions known as benzene, illuminating oils, heavy oils, lubricating oils, etc.

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The benzene fraction, which contains the readily volatile hydrocarbons, furnishes chiefly the well known solvents, and is now mainly used for producing the driving power for automobiles.

The intermediate fractions boiling up to about 300° C. are the oils used for purposes of illumination. This product must comply with several requirements. It should not smoke, should not possess any unpleasant odor, and should give a bright and steady flame which does not undergo unduly great alterations while burning. Now these conditions are only complied with by using oils of certain origin, while many of them do not fulfill the requirements in the least; it is therefore one of the special problems in refining to convert these inferior varieties into good burning oils.

The heavier fractions are used as gas oil or material for Diesel motors and the more viscous fractions are employed as lubricants.

For the use of lubricants these fractions must possess a viscosity which should be retained to certain limits of temperature variation, such as are commonly met with in the practical use. Moreover, there must be no deposit of free carbon and they must be freed from components which at high temperature would form a deposit of char and thus diminish the value of the lubrication.

Now the object of the petroleum refinery cannot be achieved by distillation alone. In fractional distillation it is inevitable that some varieties of hydrocarbons having the same boiling point will find their way into certain fractions, notwithstanding the fact that their chemical properties are distinctly dissimilar. Distillation alone is thus quite unsuitable. The problem to be solved is how to obtain only the valuable hydrocarbons, and to eliminate those with undesirable or injurious properties.

### *Refining with Sulphuric Acid*

Consequently recourse has been had to the chemical treatment of petroleum. In this treatment sulphuric acid is mainly used for the purpose of removing the resinous matters and certain hydrocarbons such as olefines. These constituents are separated out in the form of residual sludge—impurities which can be utilized in practice only with extreme difficulty or possibly not at all. For the removing of the aromatic hydrocarbons considerable quantities of sulphuric acid sometimes in the state of "fuming acid" have to be used according to the proportion of these hydrocarbons present in the distillate and this renders the process distinctly costly. The application of large quantities of sulphuric acid and especially in the state of fuming has a further drawback. It does not attack only the aromatic and unsaturated hydrocarbons, but also destroys a certain quantity of the saturated constituents. The process has thus the disadvantage of still further reducing the proportion of useful hydrocarbons finally obtained.

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In order to avoid the drawbacks of the treatment hitherto employed, it is necessary to abstain from the purely chemical treatment and to substitute a physical method of extraction. The constituents of the petroleum must be treated with a solvent which dissolves the hydrocarbons of the unsaturated aromatic group but leaves the saturated hydrocarbons undissolved. In this way the two classes of hydrocarbons can be separated from one another and employed in the way for which they are suited.

The most important point in the solution of this problem consisted in the discovery of an appropriate solvent and finally liquefied sulphur dioxide was found to be the best material. This substance readily dissolves the unsaturated hydrocarbons, which are responsible for the unsatisfactory burning and odor characteristic of certain illuminating oils, and leaves undissolved the useful illuminating oil. Frasch attempted to achieve the same object by the application of alcohol, but liquefied sulphur dioxide has certain advantages which alcohol has not. The chief advantage, apart from cheapness, consists in the fact that liquefied sulphur dioxide can be readily recovered from the solution, so that in practice the solvent can be used again and again in a cycle of operations, without any appreciable loss.

## New Refining Process with Liquefied Sulphur Dioxide

The new process makes use of this peculiar property of liquid sulphur dioxide. If the distillate is agitated with liquid sulphur dioxide at a low temperature the aromatic compounds are dissolved, but the paraffins and naphthalenes are unaffected. Moreover, owing to the difference in the specific gravity, two distinct layers are formed so that it is quite easy to separate them from one another.

The process is comparatively simple. A certain quantity of the sulphur dioxide is added to the petroleum distillate and is dissolved. This addition is continued until the distillate is saturated, and, from that point, each added portion of sulphur dioxide dissolves the aromatic constituents and separates out with them as a distinct layer. Thus, as the treatment is continued, the distillate becomes poorer and poorer in aromatic hydrocarbons until ultimately a refined distillate is obtained, which contains only a small amount of sulphur dioxide, and is practically free from the objectionable aromatic constituents.

It may be observed that the nature of this operation depends to some extent on the temperature. With a rising temperature the solubility of the paraffin hydrocarbons and the naphthalenes—which at low temperatures are practically insoluble—increases rapidly; whereas the aromatic hydrocarbons are soluble in sulphur dioxide at *all* temperatures. It is therefore important that the temperature should be main-

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tained at a suitably low level, particularly in the case of distillate rich in aromatic hydrocarbons. The degree to which the process is affected by the temperature naturally depends on the nature of the material treated. For instance, in the case of a distillate from Borneo, which contained as much as 40 per cent. of aromatic hydrocarbons, treatment with 66 per cent. liquid sulphur dioxide at $-7°$ C. produced no separa-

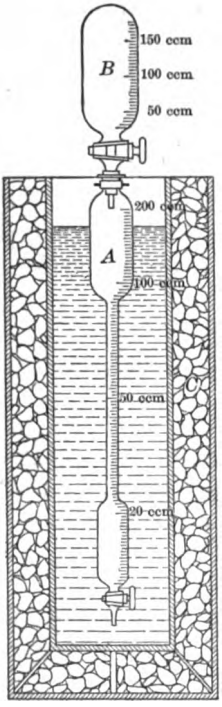


FIG. 1

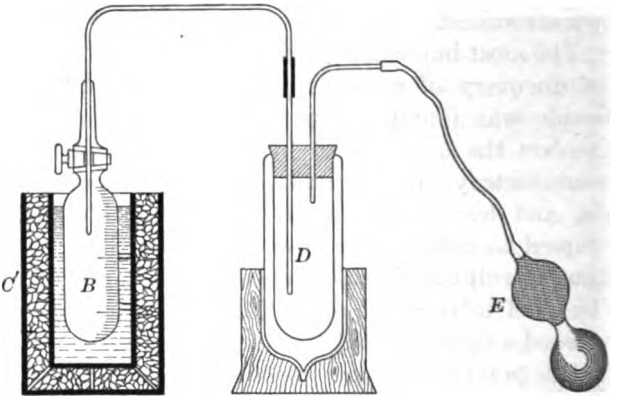


FIG. 2

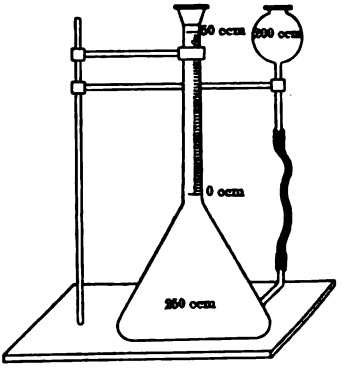


FIG. 3

tion; but a considerable proportion of the aromatic constituents could be separated at $-10°$. On the other hand, in the case of a Mexican distillate, which only contained about 17 per cent. of aromatic hydrocarbons, treatment with an equal amount of liquid sulphur dioxide produced separation even at a temperature higher than $+10°$ C. In practice it is found that each distillate requires special treatment, according to its composition, in order to obtain the best results.

This process, which is now being used on a large scale in the petroleum industry, has been worked out by the author also as a laboratory method. For the experiments he uses:

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1. A strong-walled graduated burette of 200 cm. contents drawn out pear-shape like on each end with a delivery cock protected against being forced out. (See Fig. 1, *A*.)

2. A strong-walled cylindrical graduated vessel of about 230 cm. contents with a delivery cock also protected against being forced out. (See Fig. 1, *B*.)

3. Two cooling baths for vessels 1 and 2. The cooling bath is insulated on the outside by means of cork sheets and consists of two tin cylinders which telescope into each other concentrically. The outer cylinder contains a freezing mixture, and the inner a liquid which conducts cold, preferably a distillate of illuminating oil. (See Figs. 1 and 2, *C* and *C'*.)

4. A flask with a graduated neck and a funnel attached to it. (See Fig. 3.)

5. A sprinkling flask for liquefied sulphur dioxide, consisting of an evacuated double-walled Dewar vessel. (See Fig. 2, *D* and *E*.)

To carry out the experiment, one proceeds as follows:

The graduated burette is filled with 50 cm. of the distillate to be examined, and the cylindrical vessel with double the quantity of the distillate of liquefied sulphur dioxide. Both liquids are cooled in the cooling bath to $-12°$ C. The filling of the cylindrical vessel with sulphur dioxide is demonstrated on Fig. 2. Then the cylindrical vessel is connected, by means of a tight-closing cork stopper, with the burette, and the connection is reinforced either by copper wire or by a suitable clamp. Then sulphur dioxide is allowed to flow from the cylindrical vessel into the burette until in the lower part of the burette, after shaking, a permanently remaining layer is observed. Then one-third the quantity of the balance of sulphur dioxide is allowed to run into the burette. It is again cooled to $-12°$ C., shaken, and then the lower layer is drawn off into a Dewar vessel. In the same manner, the remaining portion of the sulphur dioxide is introduced, and the sulphur dioxide solution is subsequently allowed to run into the same vessel. Finally, the non-dissolved saturated hydrocarbons remain in the burette.

In treating high-boiling distillates, such as gas oil or lubricating oil, it is sufficient to allow the sulphur dioxide to escape from the two separated products and to establish the ratio of the two groups of hydrocarbons, either by direct weighing or by specific gravity.

In treating relatively low-boiling distillates, such as benzene and illuminating oil, losses may be suffered if the sulphur dioxide is allowed to evaporate voluntarily. In order to avoid these losses it is preferable not to evaporate the sulphurous acid, but to absorb it in a closed vessel by means of an alkaline solution. In this case, one proceeds as follows:

After the sulphur dioxide solution has been drawn off, the burette is connected, by means of a cork stopper, with a glass vessel, and gradually

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under stirring the oil layer treated with sulphurous acid is allowed to run into an alkaline solution. After all the sulphurous acid has been neutralized and the contents of the vessel has been cooled, the stopper is opened and water is allowed to run in until the whole quantity of the oil has cooled in the graduated part of the neck. Then the volume of the oil layer is determined; also the specific gravity, and from the obtained data the quantity of the saturated hydrocarbon is established. The difference between this weight and the weight of the original distillate gives the difference of aromatic and cyclic unsaturated hydrocarbons.

Table I gives the results of a very large number of experiments with illuminating oil of various origin. Further experiments to extend this method also for the examination of crude oil are in the course of being worked out.

TABLE I.—*Experiments with Illuminating-Oil Distillates*

| Origin of Petroleum | A | B | C | D |
|---|---|---|---|---|
| EUROPE: | | | | |
| *Roumania:* | | | | |
| Bustenari | 0.8195 | 0.8030 | 75.0 | 25.0 |
| Câmpina | 0.8125 | 0.7968 | 82.9 | 17.1 |
| Moreni | 0.8185 | 0.8050 | 77.0 | 23.0 |
| Tintea | 0.8195 | 0.8090 | 82.5 | 17.5 |
| Filipesti | 0.8104 | 0.7984 | 84.7 | 15.3 |
| Policiori | 0.8073 | 0.7955 | 84.3 | 15.7 |
| *Galicia:* | | | | |
| Urycs | 0.8230 | 0.8105 | 80.6 | 19.4 |
| Rogi | 0.8120 | 0.8010 | 81.3 | 18.7 |
| Tustanovice | 0.8105 | 0.8000 | 80.8 | 19.2 |
| Boryslaw | 0.8166 | 0.8028 | 83.1 | 16.9 |
| *Germany:* | | | | |
| Wietze | 0.8200 | 0.8098 | 87.4 | 12.6 |
| Pechelbronn | 0.8058 | 0.7992 | 92.3 | 7.7 |
| *Scotland* | 0.800 | 0.7815 | 78.3 | 21.7 |
| | 0.7915 | 0.7807 | 80.4 | 19.6 |
| *Italy* | 0.8030 | 0.7902 | 64.7 | 35.3 |
| ASIA: | | | | |
| *Russia:* | | | | |
| Bibi-Eybat (Baku) | 0.8160 | 0.8085 | 92.7 | 7.3 |
| Bibi-Eybat (Baku) | 0.8145 | 0.8092 | 91.7 | 8.3 |
| Balachany (Baku) | 0.8192 | 0.8150 | 93.1 | 6.9 |
| Grossny | 0.8170 | 0.8072 | 85.6 | 4.4 |
| Grossny | 0.8155 | 0.8062 | 83.4 | 16.6 |
| Maikopp | 0.8165 | 0.800 | 71.5 | 28.5 |

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TABLE 1.—*Continued*

| | | | | |
|---|---|---|---|---|
| *British-India:* | | | | |
| Burmah | 0.814 | 0.799 | 80.7 | 19.3 |
| Burmah | 0.817 | 0.800 | 80.1 | 19.9 |
| Burmah | 0.840 | 0.8265 | 83.2 | 16.8 |
| *Persia* | 0.7988 | 0.7862 | 83.2 | 16.8 |
| *Holl. India:* | | | | |
| Borneo | 0.8480 | 0.8035 | 58.6 | 43.4 |
| Borneo | 0.8505 | 0.8090 | 55.5 | 44.5 |
| Sumatra | 0.7994 | 0.7865 | 83.8 | 16.2 |
| AMERICA: | | | | |
| *United States:* | | | | |
| Pennsylvania | 0.7960 | 0.7920 | 93.5 | 6.5 |
| Ohio | 0.7990 | 0.7926 | 95.1 | 4.9 |
| Ohio | 0.7985 | 0.7892 | 90.9 | 9.1 |
| Texas | 0.8240 | 0.816 | 87.5 | 12.5 |
| California | 0.8098 | 0.7982 | 85.0 | 15.0 |
| California | 0.8100 | 0.803 | 90.5 | 9.5 |
| California | 0.8124 | 0.8015 | 81.8 | 18.2 |
| California | 0.8190 | 0.8112 | 80.8 | 19.2 |
| California | 0.8195 | 0.8084 | 81.4 | 18.6 |
| California | 0.8194 | 0.8112 | 85.6 | 14.4 |
| *Mexico* | 0.8020 | 0.7926 | 84.0 | 16.0 |
| *Mexico* | 0.8030 | 0.7930 | 83.9 | 16.1 |
| *Mexico* | 0.8082 | 0.7972 | 82.9 | 13.1 |
| *Peru:* | | | | |
| Lobitos | 0.8174 | 0.807 | 81.8 | 18.2 |
| Lobitos | 0.8175 | 0.8122 | 89.5 | 10.5 |
| *Argentina* | 0.800 | 0.793 | 90.7 | 9.3 |
| *Argentina* | 0.810 | 0.804 | 90.4 | 9.6 |
| AFRICA: | | | | |
| *Egypt* | 0.8104 | 0.7976 | 82.5 | 17.5 |

A = Specific gravity before treatment with $SO_2$; B = specific gravity after treatment with $SO_2$; C = paraffin- and naphthene-hydrocarbons in per cent. by weight; D = aromatic and cyclic unsaturated hydrocarbous in per cent. by weight

The removal of the aromatic hydrocarbons from an illuminating-oil distillate causes not only a lowering of the specific gravity, but also the raising of the burning capacity and illuminating power. The richer a distillate is in aromatic and heavy hydrocarbons, the more distinctly stands out the improvement brought about by treatment with sulphurous acid. The effect of the new process is clearly discerned when treating a Bustenari distillate. Whereas the chemical process hitherto employed produces a very inferior illuminating oil which smokes badly, and when

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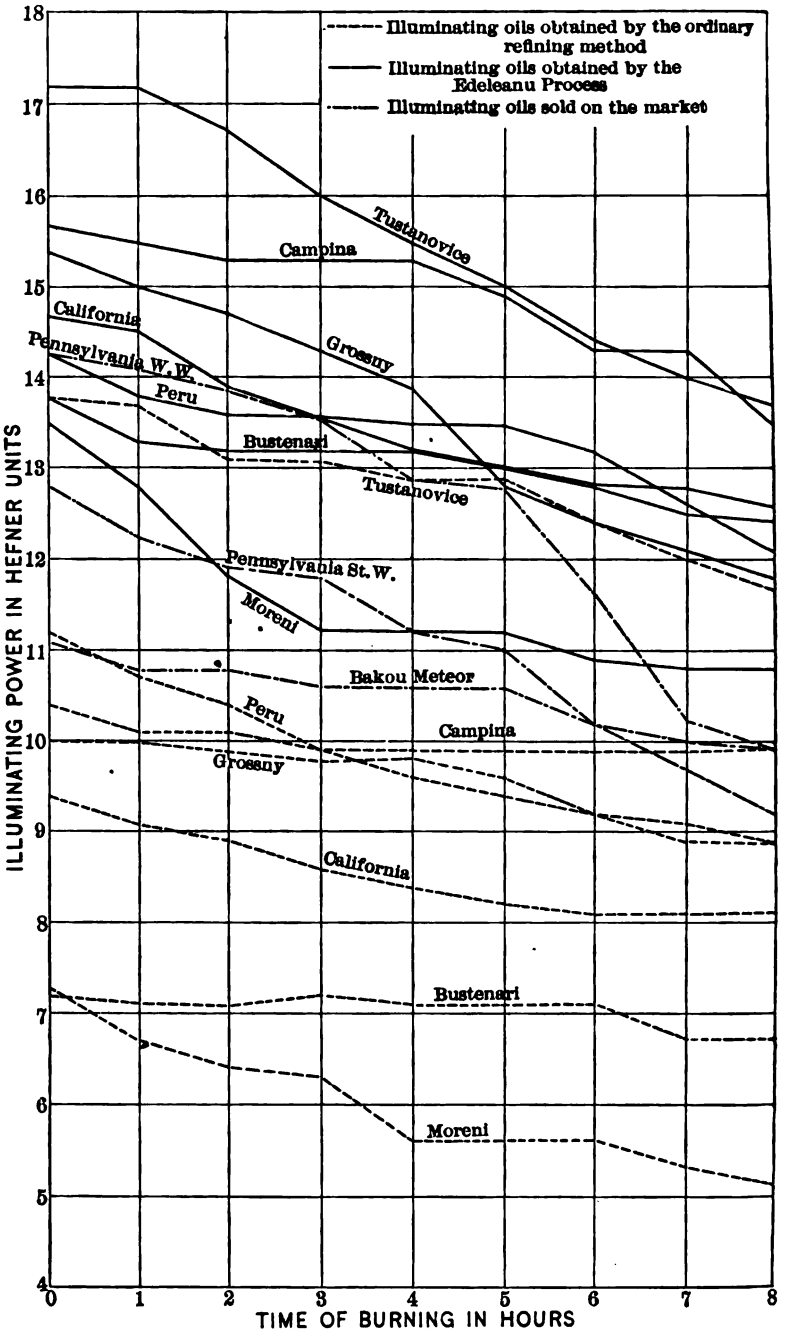


FIG. 4.—COMPARISON OF OILS FROM ORDINARY AND EDELEANU PROCESSES.

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burned in a 14-unit Cosmos burner develops only 7 cp., the treatment with liquid sulphur dioxide produces an oil equal in quality to the best American and Russian oils.

Table II illustrates the action of the old and new methods respectively on distillates of various origin. See also the graphical comparison in Fig. 4.

TABLE II.—*Comparison of the Qualities of Illuminating Oils Obtained by the Ordinary and by the Edeleanu Process*

| Origin | Refining Process | Specific Gravity | Illuminating Power: Photometric Measurements at each Hour with the Kosmos Burner 14. Illuminating Power in Hefner Units | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 |
| Bustenari... | Ordinary process... | 0.818 | 7.2 | 7.1 | 7.1 | 7.2 | 7.1 | 7.1 | 7.1 | 6.7 | 6.7 |
| | $SO_2$ process..... | 0.803 | 13.8 | 13.3 | 13.2 | 13.2 | 13.2 | 13.0 | 12.8 | 12.5 | 12.4 |
| Câmpina.... | Ordinary process... | 0.812 | 10.4 | 10.1 | 10.1 | 9.9 | 9.9 | 9.9 | 9.9 | 9.9 | 9.9 |
| | $SO_2$ process..... | 0.797 | 15.7 | 15.5 | 15.3 | 15.3 | 15.3 | 14.9 | 14.3 | 14.3 | 13.5 |
| Moreni..... | Ordinary process... | 0.818 | 7.3 | 6.7 | 6.4 | 6.3 | 5.6 | 5.6 | 5.6 | 5.3 | 5.1 |
| | $SO_2$ process..... | 0.805 | 13.5 | 12.8 | 11.8 | 11.2 | 11.2 | 11.2 | 10.9 | 10.8 | 10.8 |
| Tustanovice. | Ordinary process... | 0.799 | 13.8 | 13.7 | 13.1 | 13.1 | 12.9 | 12.9 | 12.4 | 12.0 | 11.7 |
| | $SO_2$ process..... | 0.786 | 17.2 | 17.2 | 16.7 | 16.0 | 15.5 | 15.0 | 14.4 | 14.0 | 13.7 |
| Grosny..... | Ordinary process... | 0.815 | 10.0 | 10.0 | 9.9 | 9.8 | 9.8 | 9.6 | 9.2 | 9.1 | 8.9 |
| | $SO_2$ process..... | 0.806 | 15.4 | 15.0 | 14.7 | 14.3 | 13.9 | 13.8 | 12.4 | 12.1 | 11.8 |
| California ... | Ordinary process... | 0.810 | 9.4 | 9.1 | 8.9 | 8.6 | 8.4 | 8.2 | 8.1 | 8.1 | 8.1 |
| | $SO_2$ process..... | 0.798 | 14.7 | 14.3 | 13.9 | 13.5 | 13.2 | 13.0 | 12.8 | 12.8 | 12.6 |
| Peru..... | Ordinary process... | 0.817 | 11.2 | 10.7 | 10.4 | 9.9 | 9.6 | 9.4 | 9.2 | 8.9 | 8.9 |
| | $SO_2$ process..... | 0.807 | 14.3 | 13.8 | 13.6 | 13.6 | 13.5 | 13.5 | 13.2 | 12.6 | 12.1 |
| Qualities of Illuminating Oil Brands Sold on the Market | | | | | | | | | | | |
| Pennsylvania W. W..... | | 0.7915 | 14.3 | 14.1 | 13.9 | 13.5 | 12.9 | 12.8 | 11.6 | 10.1 | 9.9 |
| Pennsylvania St. W..... | | 0.7995 | 12.8 | 12.2 | 11.9 | 11.8 | 11.2 | 11.0 | 10.2 | 9.7 | 9.2 |
| Bakou Meteor..... | | 0.8087 | 11.1 | 10.8 | 10.8 | 10.6 | 10.6 | 10.6 | 10.2 | 10.0 | 9.9 |

## *Utilization of By-Products*

The extract obtained in addition to illuminating oil contains, as mentioned, aromatic hydrocarbons and hydrocarbons rich in carbon. On account of their low burning capacity, they cannot be employed as burning oils; nevertheless, by dividing the extract into two products by

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redistillation, they become useful; the lighter part as turpentine substitutes or mixed with light benzene as a motor spirit of high efficiency, and the heavier as motor oil for Diesel motors. The extract is, therefore, as regards the possibility of turning it to account, at least equally as valuable as the distillate.

Further experiments have shown that the extract can be resolved by special means into the lower homologues of the aromatic hydrocarbons. By pyrogenic distillation the hydrocarbons contained in the extract are split up; and, besides a gaseous part, a large percentage of tar is produced. The latter is distinguished from tar derived from coal by its higher content of the lower members of the aromatic series of hydrocarbons.

Table III shows the products of gasification (pyrogenic distillation) of the extract as compared with other bituminous substances.

TABLE III.—*Comparison of Tar Extract and Coal Tar*

| No. | Composition of Tar Extract | | | | Average Composition of Coal Tar according to Prof. Kramer | |
|---|---|---|---|---|---|---|
| | Limits of Temperature ° Celsius | Density at 15° C | Products of Distillation | Per cent. Calculated for Tar | Products of Distillation | Per cent. |
| 1 | 80°—113° | 0.8700 | Benzène Toluène | 13.3 | Benzène et Homol. | 2.50 |
| 2 | 113°—140° | 0.8700 | Little toluène | ..... | Phénols..... | 2.00 |
| | | | with much xylène | 9.1 | Rèsidue..... | 0.25 |
| 3 | 140°—250° | 0.9870 | *a.* Naphtaline.. | 8.7 | Naphtaline..... | 6.00 |
| | | | *b.* Hydrocarb. | | Heavy oil..... | 20.00 |
| | | | arom. liquids... | 23.7 | Anthracène, Phénantrène..... | 2.00 |
| 4 | 250°—400° | ..... | *a.* Anthracène crude..... | 1.0 | Asphalt..... | 38.00 |
| | | | *b.* Anthracène oil..... | 20.0 | | |
| 5 | above 400° | ..... | Rèsidue..... | 21.2 | Coke..... | 24.00 |
| | | | Loss..... | 3.0 | Water..... | 4.00 |

While coal tar hardly contains 2.6 per cent. of aromatic hydrocarbons, the tar produced by pyrogenic distillation of petroleum contains between 15 to 23 per cent.

Among the fractions of this tar which distill above 200° a large quantity of naphthalene is found, which varies between 5 and 12 per cent., and of anthracene varying between 0.5 and 2.7 per cent.

The benzene which one obtains after the first redistillation of this tar is much purer than that which one obtains under the same conditions

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by redistilling coal tar. This result appears from Table IV which shows comparatively the composition of benzene obtained by the first redistillation of tar from the extract (treatment with dioxide of sulphur), and of that of the type of benzene used in commerce.

TABLE IV.—*Composition of Commercial Brands of Benzene*

| Commercial Benzene | | | | Benzene from Extract 80°—113° C. | |
|---|---|---|---|---|---|
| Limits of Temperature, ° Celsius | 30 Per Cent. | 50 Per Cent. | 90 Per Cent. | Limits of Temperature ° Celsius | Per Cent. |
| 80° | 0 | 0 | 25 | 85° | 0.0 |
| 90° | 2 | 4 | 70 | 90° | 13.4 |
| 95° | 12 | 26 | 83 | 95° | 48.0 |
| 100° | 30 | 50 | 90 | 100° | 74.0 |
| 105° | 42 | 62 | 94 | 105° | 88.0 |
| 110° | 70 | 71 | 97 | 110° | 96.0 |
| 115° | 82 | 82 | 98 | 115° | 98.0 |
| 120° | 90 | 90 | 99 | 120° | 99.0 |

As it appears from this table, the fraction of 80 to 103° which one obtains by the pyrogenic distillation of the petroleum extract corresponds to an intermediate quality which ranges between the 50 per cent. type and the 90 per cent. type, as sold in the market, and such as is obtained from coal tar by means of several redistillations. It must also be remembered that the benzene obtained by the pyrogenic treatment of the extract and of other products of petroleum is totally free from thiophen and only contains a few impurities. By means of a little soda it is obtained perfectly pure.

*Effect of Process on Distillates Containing Sulphur*

It is further to be noted that the liquid sulphur dioxide also possesses the property of dissolving out from crude distillates certain sulphur-containing constituents. For instance, by treating a Mexican distillate of 0.803 specific gravity and with a sulphur content of 0.6 per cent. an oil is obtained which shows a percentage of sulphur of only 0.08 per cent. and another oil of 0.79 specific gravity, with a sulphur content of 0.46, could be reduced to a percentage of sulphur of 0.04.

The process was first tested in a small experimental plant in the Vega refinery at Ploesti.

The results obtained with this small experimental plant were entirely satisfactory and shortly afterward the construction of the necessary plant for carrying out the process on a large scale was undertaken

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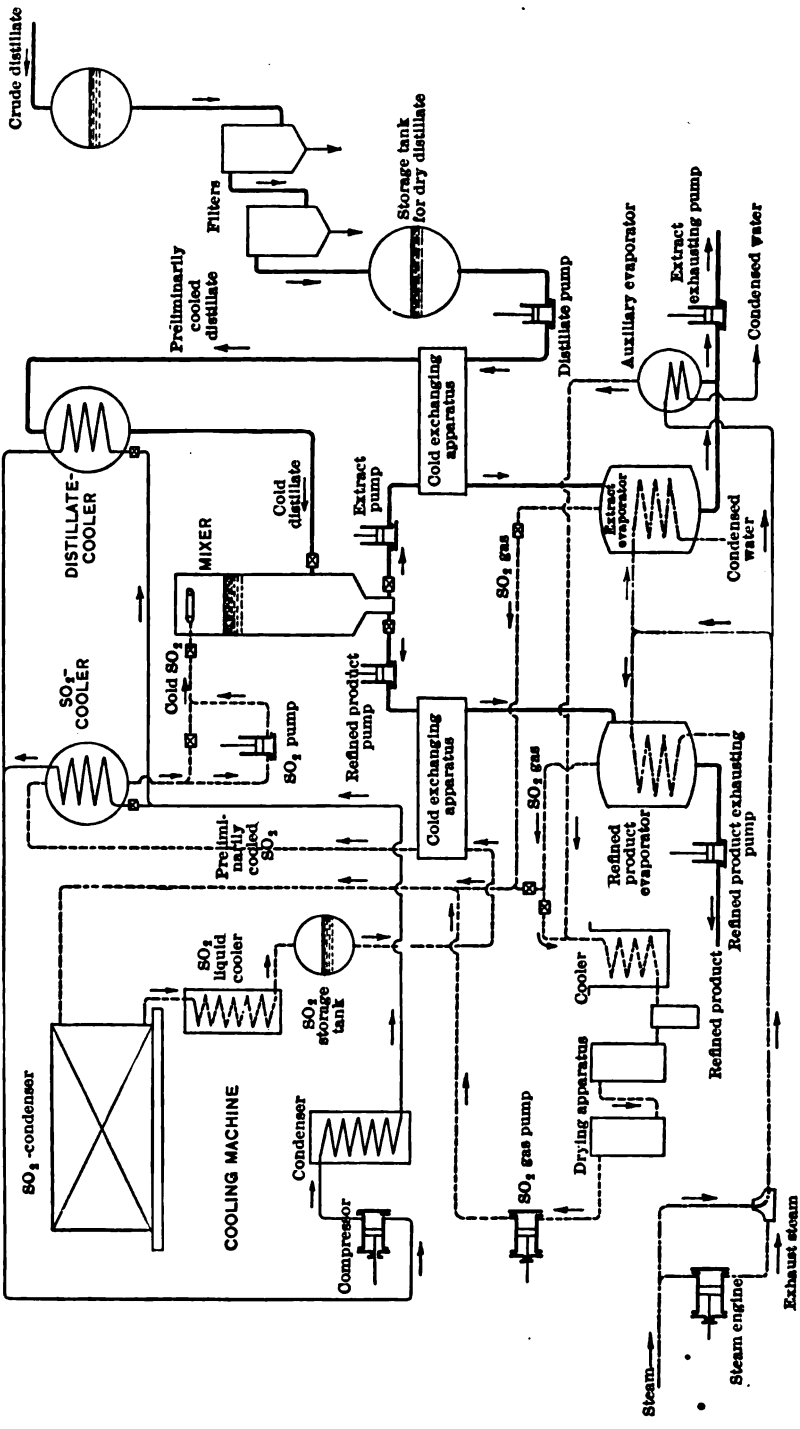


FIG. 5.—PLAN OF A REFINING PLANT FOR LIQUEFIED SULPHUR DIOXIDE PROCESS.

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by the firm of A. Borsig, of Tegel (Berlin). A refining plant was next installed in Rouen, and was followed soon after by the construction of a plant of similar size in Ploesti. The capacity of the plant at Ploesti, originally intended to treat up to 25 tons of distillate per day, was soon increased to 70 tons a day and the nature of the products is giving every satisfaction.

These trials were considered to demonstrate completely the practicability of the process on a large scale. One of the largest and most influential petroleum concerns has since ordered a plant with a daily capacity of 250 tons, and already an increase to 500 tons is being contemplated. There are now other large plants in course of construction for Galicia, Roumania, Borneo, India, etc.

The following is a description of the treatment of a Roumanian distillate from Bustenari.

The dried distillate to be treated passes through filters charged with salt, is collected in a receiver, and finally reaches the distillation cooler after going through a cold-exchanging apparatus. The temperature of the distillate is first lowered in the cold-exchanging apparatus, and it is then further cooled in the distillation cooler itself until it reaches the desired low temperature. Similarly, the liquid sulphur dioxide coming from the vessel passes through a second cold-exchanging apparatus, where its temperature is reduced somewhat, and from thence to the sulphur dioxide cooler itself, where it receives the necessary final cooling.

The arrangement is shown in Fig. 5.

The distillate to about $-10°$ C. is now let into the lower mixer, a cylindrical vessel which is provided with level gauges, and in its lower part with gauge glasses, so that the reaction in the liquid may be accurately observed.

The introduction of the liquid sulphur dioxide into the mixer begins immediately. The sulphur dioxide is at first entirely absorbed by the distillate, without any appreciable change of the liquid. As soon as the distillate is saturated with the liquid sulphur dioxide, dark cloud masses begin to rise, and the color changes suddenly to a deep brown. This change of color is brought about by the separating out of the liquid sulphur dioxide loaded with the aromatic hydrocarbons, and depositing at the bottom of the mixer. There are now two sharply divided layers which can be distinctly observed through the level gauges. As soon as there is a certain quantity of extract formed, the drawing-off commences; at the same time further quantities of liquid sulphur dioxide are supplied to the mixer.

The extract is drawn by the extract pump through the cold-exchanging apparatus into the extract evaporator. In the cold-exchanging apparatus it meets the comparatively warm distillate flowing to the dis-

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tillation cooler (as described above) by which its temperature is considerably raised. In this way the distillate receives a preliminary cooling and the extract a preliminary warming. The latter is then further warmed in the extract evaporator, which is provided with heating coils. This causes the rapid evaporation of the dissolved sulphur dioxide which flows directly into a condenser, where it accumulates in the form of a liquid; it is then stored in the reservoir, from whence it commences its cycle afresh.

As soon as the greater part of the sulphur dioxide contained in the extract has been driven off, the evaporation is gradually brought to a stop. The final remaining portion of the sulphur dioxide is held very tenaciously by the extract, and cannot be driven off merely by the application of heat. In order to prepare the extract evaporator for a fresh charge, the extract with its still remaining small contents of sulphur dioxide is run into an auxiliary evaporator. Here, the balance of the sulphur dioxide is drawn off by a sulphur dioxide gas pump under continuous application of heat, and sent to the condenser. This is continued until about 0.3 per cent. still remains in the extract, when the latter is removed from the auxiliary evaporator.

The refined product is pumped through the second cold-exchanging apparatus into the second evaporator. It meets in the cold-exchanging apparatus the liquid sulphur dioxide flowing from the tank to the cooler, and is raised to practically the same temperature. In this way the liquid sulphur dioxide receives a preliminary cooling (just as in the case of the distillate). The refined product is then collected in the evaporator and undergoes a similar process of evaporation as the extract. By means of heat and continued suction the sulphur dioxide is almost completely removed, only about 0.1 or 0.2 per cent. remaining; at which stage the refined-product evaporator is emptied.

The treated distillate, after being washed and neutralized with an alkaline solution, may be used as finished illuminating oil. In some cases subsequent treatment with very small quantities of sulphur dioxide is advantageous in order to obtain a "water-white" color.

The exhaust steam from the steam engine which drives the plant, is utilized for the heating of the evaporators.

While the evaporating process is still going on in the extract evaporator a new operation takes place, a freshly cooled distillate being admitted to the mixer and saturated with liquid sulphur dioxide, until again fresh quantities of extract settle out. By this time the evaporation in the extract evaporator is also finished and the apparatus emptied, so that the new operation is able to proceed without interruption.

The cooling of the distillate and sulphur dioxide collected in the two cooling vessels is effected in the usual manner with the help of cooling coils, which form a part of a separately working cooling machine. The cold-producing medium enters the cooling tubes as a liquid, evaporates on

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account of the continuous low pressure produced by the suction of the compressor of the cooling machine, and abstracts from its surroundings,

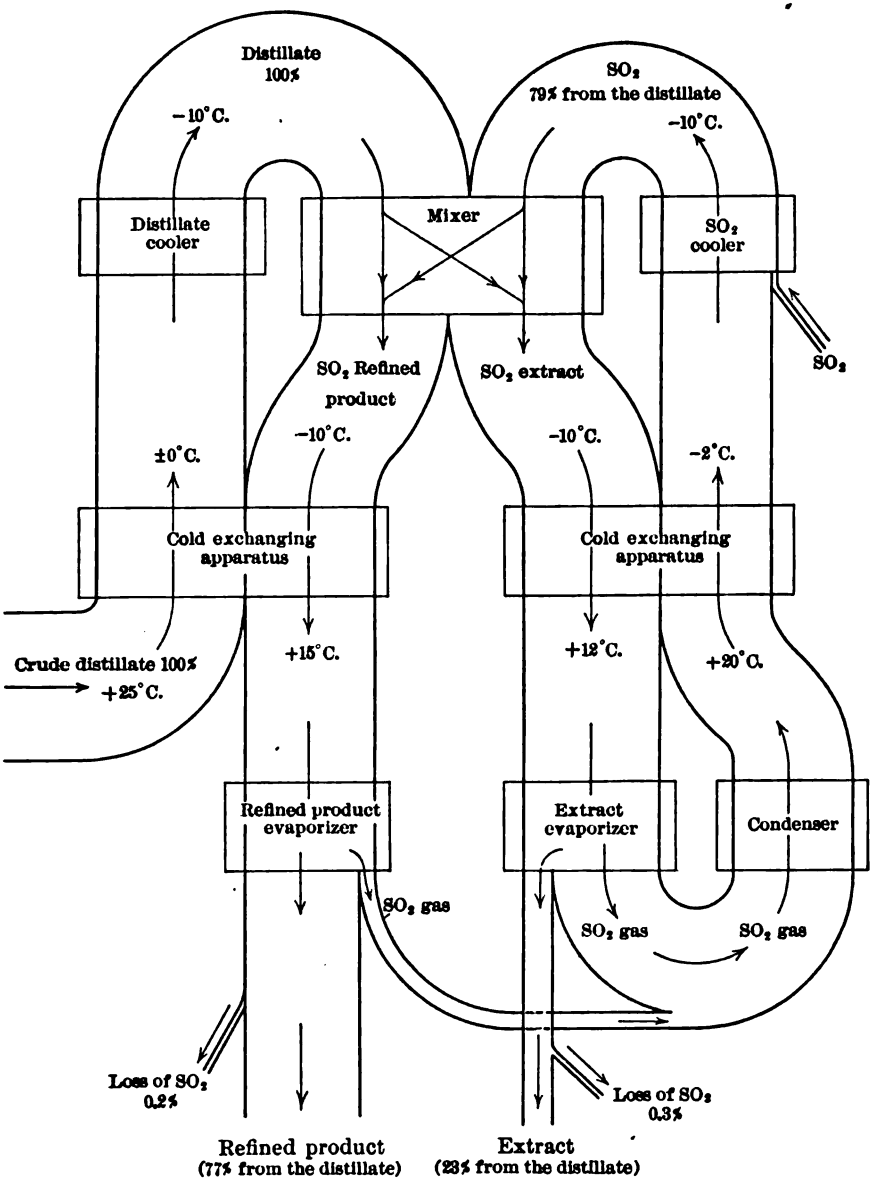


FIG. 6.—PLAN OF THE EDELEANU PROCESS.

i.e., the distillate and liquid sulphur dioxide, the heat required for its evaporation. The vapor of this cooling medium is afterward sucked up



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by the compressor and led to the condenser of the cooling machine, where it is again collected in liquid form.

The method of working is shown graphically in Fig. 6. In this diagram it is shown how fresh distillate is continuously entering the system, and how it finally leaves the system split up into refined product and extract, and how the sulphur dioxide, which effects the splitting up, is recovered with the exception of a very small quantity, and is then used again for the process, making a complete cycle. It may also be seen that of the distillate put into the system nothing is lost; the products treated, viz., the volume of extract and the refined product, together are equal in volume to that of the original distillates.

As the sulphur dioxide performs a complete cycle, it cannot when once entering the system in a pure and anhydrous condition, give rise to any undesirable complications. It is well known that liquid sulphur dioxide is quite inactive toward metals, so that there is no fear of the apparatus becoming injured or destroyed in the course of time.

However, great care must be taken that no moisture is introduced into the system with the distillate. The latter should, therefore, as mentioned above, pass through several drying filters before it enters the apparatus, so that all the moisture may be removed. Inasmuch as the system is perfectly airtight throughout, there is of course no possibility of moisture from the atmosphere penetrating into it.

The fact that the whole of the apparatus, pipes, valves, etc., are completely airtight is responsible for the fact that scarcely the faintest smell of sulphur dioxide is perceived in the plant. This is of the greatest importance for the welfare of the workmen who are in attendance at the plant.

The arrangement of the plant is such that from the operator's platform the coolers which are placed high and the evaporators which are below, as well as the mixer in the center, may be watched and attended to. As can be seen in Fig. 7, all the important pipes with their control valves are placed on a regulating table, so that the operator in charge of the refining process can take care of all the necessary regulating and connecting manipulations without leaving his place of observation.

### *Working Results*

The working results of a plant with an average daily capacity of treating about 65 tons of distillate per 24 hr., as were reached in the plant at Ploesti, are given in the following table:

| | |
|---|---|
| Specific gravity of the crude distillate to be treated..... | 0.820 |
| Average time of operation, minutes..... | 68 |
| Quantity of distillate treated at each operation, kilograms. | 3,115 |

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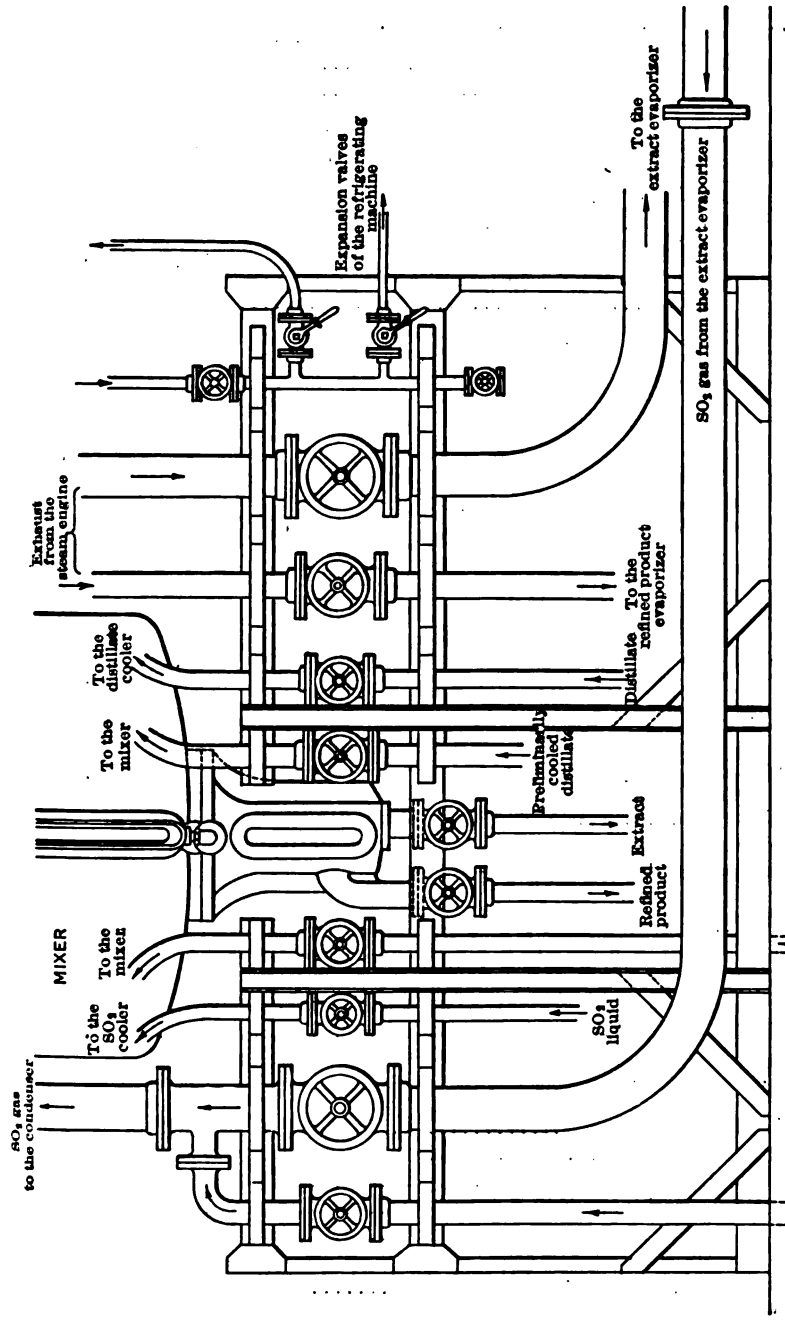


FIG. 7.—DIAGRAM OF REGULATING TABLE IN APPARATUS ROOM.

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*Working Results—Continued*

| | |
|---|---|
| Temperature in the mixer, degrees centigrade | −10 |
| Specific gravity of treated product | 0.8028 |
| Specific gravity of extract | 0.8691 |
| Sulphurous acid left in the treated product, per cent | 0.16 |
| Sulphurous acid left in the extract, per cent | 0.36 |
| Consumption of fresh steam per operation, kilograms | 1,200 |
| Consumption of fresh steam per 1,000 kg. distillate, kilograms | 385 |
| Consumption of exhaust steam per operation, kilograms | 648 |
| Consumption of exhaust steam per 1,000 kg. distillate, cubic meters | 208 |
| Cooling water used per operation, cubic meters | 17.1 |
| Cooling water used per 1,000 kg. distillate, cubic meters | 5.5 |
| Loss of sulphurous acid per operation, kilograms | 17.5 |
| Loss of sulphurous acid per 1,000 kg. (distillate), kilograms. | 5.6 |

From the above figures, the cost of treating a Roumanian Bustenari distillate can be calculated as follows:

*Cost of Treatment in a Plant of 65 Tons Daily Capacity*

| *Cost of Plant:* | Francs |
|---|---|
| Machines and apparatus | 130,000 |
| Buildings, construction, freight, duty and other expenses. | 100,000 |
| Total | 230,000 |

*Amortization and Interest:*

| | | |
|---|---|---|
| Amortization | = 10 per cent. of 130,000 | 13,000 |
| Amortization | = 5 per cent. of 100,000 | 5,000 |
| Interest on the invested capital | = 5 per cent. of 230,000 | 11,500 |
| Total | | 29,500 |

In 325 working days, at 65 tons per day, there will be 21,125 tons treated, so that the amortization and interest amount to Frs. 1,396 per ton, or per 100 kg ..... Frs. 0.140

| *Maintenance, Labor, etc.:* | Francs |
|---|---|
| Repairs, working materials | 6,000 |
| Wages and salaries | 8,000 |
| Insurance, 1½ per cent | 3,450 |
| Lighting and heating | 2,400 |
| Drying salt, unforeseen expenses | 3,000 |
| Total | 22,850 |

The cost of maintenance and labor, therefore, amount to Frs. 1.082 per ton, or per 100 kg ..... Frs. 0.108

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*Working Results.—Continued*

*Cost of Materials:*

| There are required for each 1000 kg. of distillate: | Francs | |
|---|---|---|
| 385 kg. fresh steam at Frs. 3.50 per ton | 1.35 | |
| 208 kg. exhaust steam at Frs. 1.50 per ton | 0.31 | |
| 5.5 cbm. cooling water at Frs. 0.04 per cbm | 0.22 | |
| 5.6 kg. sulphurous acid at Frs. 18.00 per 100 kg | 1.01 | |
| Cost of materials per 1,000 kg | 2.89 | |
| Cost of materials per 100 kg | | Frs. 0.289 |

The total costs of treatment, therefore, amount to ..... Frs. 0.537 or practically Frs. 0.54 per 100 kg. of distillate or 3.9 shillings per ton. It must also be taken into account that Frs. 18 per 100 kg. of sulphurous acid is a very high figure and refers to imported sulphurous acid. The cost of this material is very much lower when sulphurous acid is used which is produced in the country itself, or actually in the refinery.

*Cost of Treatment in a Plant of 500 Tons Daily Capacity*

| *Cost of Plant:* | Francs |
|---|---|
| Machines and apparatus | 1,100,000 |
| Buildings, construction, etc | 500,000 |
| Total | 1,600,000 |

| *Amortization and Interest:* | | Francs |
|---|---|---|
| Amortization | = 10 per cent. of Frs. 1,100,000 | 110,000 |
| Amortization | = 5 per cent. of Frs. 500,000 | 25,000 |
| Interest on the invested capital | = 5 per cent. of Frs. 1,600,000 | 80,000 |
| Total | | 215,000 |

In 325 working days, at 500 tons per day there will be produced 162,500 tons, so that the amortization and interest amount to Frs. 1.323 per ton, or per 100 kg ..... Frs. 0.132

| *Maintenance, Labor, etc.:* | Francs | |
|---|---|---|
| Repairs, working materials | 40,000 | |
| Wages and salaries | 30,000 | |
| Insurance, 1½ per cent. | 24,000 | |
| Lighting and heating | 10,000 | |
| Drying salt, unforeseen expenses | 20,000 | |
| Total | 124,000 | |
| Brought forward | | 0.132 |
| The cost of maintenance and labor, therefore, amount to Frs. 0.763 per ton, or per 100 kg | | 0.076 |

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*Working Results—Continued*

| *Cost of Materials:* | Francs | |
|---|---|---|
| There are required for each 1000 kg. of distillate: | | |
| 385 kg. fresh steam at Frs. 3.50 per ton | 1.35 | |
| 208 kg. exhaust steam at Frs. 1.50 per ton | 0.31 | |
| 5.5 cbm. cooling water at Frs. 0.04 per cbm | 0.22 | |
| 5.6 kg. sulphurous acid at Frs. 10.00 per 100 kg | 0.56 | |
| Cost of materials per 1,000 kg | 2.44 | |
| Cost of materials per 100 kg | | 0.244 |
| The total cost of treatment | | Frs. 0.452 |

or roughly, Frs. 0.45 per 100 kg. of distillate or 3.26 shillings per ton. In this case, the cost of sulphurous acid is taken at Frs. 10 per 100 kg. sulphurous acid produced on the spot being reckoned. The cost of production is amply estimated throughout.

The above tables are based on figures actually obtained from the existing plants. It is, however, to be anticipated, that with further perfection of the working of the process, the costs will be reduced even more.

In conclusion it may be well to summarize some of the chief advantages of this new process.

(1) The process makes it possible to obtain illuminating oil of good quality from the greatest variety of low-grade oils.

(2) The sulphur dioxide employed is cheap and practically all of it is recovered.

(3) Sulphur dioxide required for the process can even be obtained from the acid sludge available in the refinery in the manufacture of lubricating oils.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should be in the form of a new paper.

# Rolled Steel Roll Shells

BY JAMES C. H. FERGUSON, SAN FRANCISCO, CAL.

(Pittsburg Meeting, October, 1914)

THE fact that little if anything has appeared in the technical press or in the *Transactions* of the Institute on the subject of roll shells proper, used in various grinding appliances such as Cornish rolls or Chilean mills, may be taken as an incentive, if not an excuse, for the present article. No attempt is made to enter into a discussion as to the theory or mathematics of crushing as performed by rolls, the subject being treated in a general way with respect to the manufacture of roll shells and some salient points touching on their adaptability and wear in service.

While it stands to reason that skill coupled with judgment in the mechanical design of rolls, Chilean mills or other types of grinding machinery, with special reference to strength of parts, nicety of adjustment and other mechanical features, stand for efficiency in their operation and satisfaction in service, the exceedingly important item of roll shells or other wearing parts entering into their construction should not be lost sight of. In other words, a well-designed piece of grinding machinery is just as inefficient in service, when equipped with poor roll shells or shells made of the wrong or an inferior grade of metal, as good roll shells made according to the best practice the steel maker knows, working on an old or antiquated set of rolls.

Unfortunately there is a tendency occasionally for the machinery builder to blame the steel maker or *vice versa* if roll shells do not wear well or evenly, or if they break before being worn down to their ultimate thin point. There is no occasion for this attitude, as the product of each must stand on its own merits. But it is essential that the two work in harmony since the machinery builder usually does not furnish the shells for his rolls or grinding mills, but depends on the steel maker, who makes a specialty of this rolled product, to give the particular needed element of efficiency to his machine.

## HISTORY

Without going into an elaborate history of the subject, the following may be mentioned treating on the evolution of roll shells. During

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the early days of mining-machinery construction, cast iron was used for practically all such grinding devices as those under consideration, usually some good mixture of gray iron being the most suitable for wearing qualities. Sometimes, though not often, shells or muller rings, chilled to a depth of from ½ to 1 in. through the face, were used.

But it soon became apparent that cast iron, no matter how well selected as to grade or mixture, or how carefully molded or chilled as to physical appearance, was not a suitable material to use for grinding rings or dies, since, for the proper performance of their work, absolute homogeneity and freedom from blow-holes or other imperfections are highly important. Steel, that is cast steel, therefore, was next and most logically considered as being more suitable to fulfill the *desiderata* just mentioned. Although it was more expensive than cast iron, it was harder and tougher, and of infinitely better wearing quality. But even cast steel failed to give satisfaction particularly on the high-speed rolls of narrow face and large diameters designed by the leading manufacturers during the past two decades. It was for that reason that the steel maker soon recognized the importance of meeting the situation and has during the period mentioned evolved the rolled-steel roll shell. This was a natural evolution, and during the past 10 years practically no other material has been used except for some small or slow-running mills where the expense of steel shells was not warranted.

Therefore, it is with steel roll shells only that the steel maker is concerned when he furnishes either the machinery builder or the mill operator these wearing parts, and for a better understanding of the subject a few words regarding their chemical and physical requirements may not be amiss, before entering upon a consideration of their manufacture.

### *Chemical and Physical Requirements*

While as mentioned above the application of steel shells as wearing parts in the various types of grinding mills was comparatively new, the purely mechanical features of their manufacture was easily accomplished by the use of tire-rolling mills which for many years previous had been used for making locomotive driving-wheel and car-wheel tires. The mechanical manipulation being practically identical, it only required that the steel maker select the proper grade and composition of steel for the roll shell, not too hard, for danger of breaking, nor too soft, for fear of wearing out too fast. Of course it required much experimenting and many years' experience to attain the desired results.

With reference to the average chemical and physical properties entering into the manufacture of roll shells, the following may be given as forming the best practice by American steel makers:

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| Chemical Properties | Per cent. |
|---|---|
| Carbon | 0.65 to 0.80 |
| Phosphorus, not to exceed | 0.05 |
| Sulphur, not to exceed | 0.04 |
| Manganese | 0.60 to 0.85 |
| Silicon | 0.15 to 0.30 |

The above are the average ranges used. Shells which have given good satisfaction in actual operation had the following analyses in the heats from which they were produced:

| | Per cent. | Per cent. | Per cent. |
|---|---|---|---|
| Carbon | 0.70 | 0.73 | 0.78 |
| Phosphorus | 0.035 | 0.029 | 0.015 |
| Sulphur | 0.022 | 0.018 | 0.028 |
| Manganese | 0.741 | 0.786 | 0.745 |
| Silicon | 0.253 | 0.252 | 0.258 |

These results show that the shells run very uniform in composition.

| Physical Properties | |
|---|---|
| Tensile strength, pounds | 125,000 |
| Elastic limit, pounds | 72,000 |
| Extension, per cent | 10 |
| Contraction of area, per cent | 15 |

The above physical properties are given as being of general interest but they should not be taken as having any particular bearing on the successful working of roll shells, as they do not in any way exhibit the qualities of the metal in its resistance to abrasive wear. After all, what is desired, as previously stated, is a shell of a metal hard enough so as not to wear out too quickly and yet not so hard as to break before wearing out.

*Manufacture*

Briefly, the process of manufacture of roll shells is this: A long cylindrical steel ingot is cast by pouring molten metal into a cast-iron mold as illustrated in Fig. 1. These ingots are about 7 ft. high and the circumferential measurement of the ingot varies according to the size of the roll shell desired. When the metal is cold the molds are stripped from the ingots, which are then sliced cold in a lathe into a number of sections called billets (Figs. 2 and 3), each piece resembling a large cheese. Sometimes the ingots are sliced or cut into billets while hot, but of course cold slicing is a vastly superior method as it gives an opportunity to make a thorough examination of the center of the billet.

The top portion of these ingots, in which the segregation of impurities predominates, is discarded, and only those portions used which are free from impurities. It is a well-known fact that there is a tendency in all castings while cooling for slaggy materials and impurities to segregate and form toward the center, where the shrinkage produces a pipe or axial

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opening. The aim, therefore, in the manufacture of good roll shells must be to retain only those portions of the ingot which are on or near the outer circumference, which having cooled the most rapidly necessarily have the most uniform fiber texture and are the freest from impurities.

In order to illustrate the practice of slicing these ingots with regard to discarding the top section, reference is made to Fig. 4, which illustrates a segregation of impurities and defects due to the shrinkage of the metal and shows that when cooling, these impurities concentrate at the center and near the top of the ingot. It conclusively shows the necessity of making

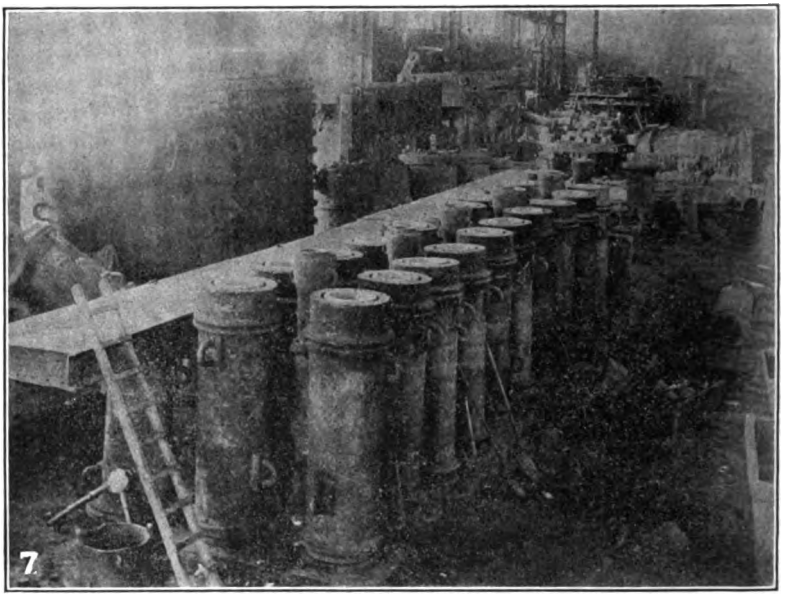

FIG. 1.—MOLDS FOR CASTING STEEL INGOTS FOR ROLL-SHELL MANUFACTURE.

a proper discard from any steel castings, in order to insure against troubles due to impurities.

These billets after being sliced, although not of the proper dimensions, actually represent the weight of the finished product desired, plus the proper allowance for loss in heating, forging and rolling, and for machining afterward if the shell calls for machine work. The billet is heated and forged out roughly and flattened under a steam hammer to the approximate diameter and face desired. A hole is punched through the center in the same operation and then, at the same heat, "beaked" on the horn of the anvil; this means hammering the rough ring thrown over the horn, producing an increase of the outside and inside diameters, and giving the ring the proper ratio, roughly, of diameter to face required.

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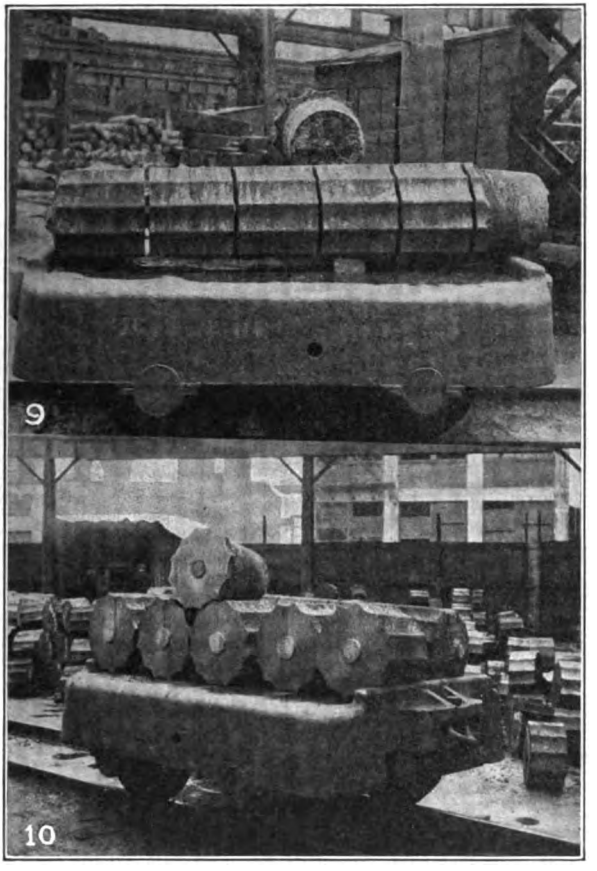
These rough rings, after being thus forged, are again placed in a heating furnace and the temperature raised to the proper degree for rolling.

A modification of this hammering process of the billets, is used by one of the manufacturers who has the most recent and up-to-date plant, and who, after heating the billet to the required temperature, works it down




FIGS. 2 AND 3.—STEEL INGOTS CUT INTO BILLETS.

under a 5,000-ton hydraulic press, which gives the metal a good reduction. During this pressing, which is done in three operations, the billet is flattened, a hole is punched in the center, and the rough ring prepared for the rolls, similarly, but unquestionably more thoroughly, than by hammering the billet.

The rolling out of the shells to the desired size is done on a tire mill, where they are revolved through pressure rollers, either in a horizontal or vertical plane. During this operation all faces of the shell are subjected

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to a very high hydraulic roll pressure, which insures a thorough reduction of the metal, giving it the necessary work to develop an ideal structure for the severe service to which roll shells are usually subjected in actual use. In this rolling the inside and outside faces and the two sides of the shell are engaged at the same time, thereby increasing the diameter of the shell by squeezing out and lengthening its circumference until the desired size is obtained, both as regards diameter and face.

As stated before, this rolling is done on the same mills used for rolling locomotive driving-wheel tires or car-wheel tires. This method has now

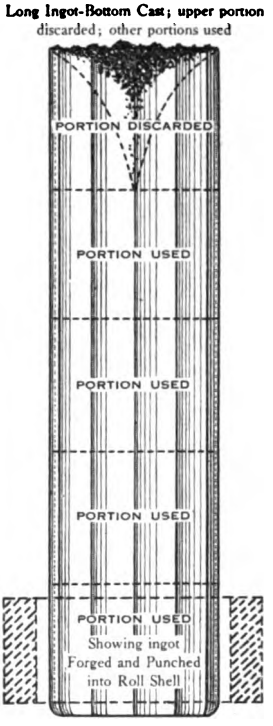


FIG. 4.—DIAGRAM SHOWING RELATION OF FINISHED ROLL SHELL WITH REFERENCE TO CENTER LINE OF INGOT.

reached a very high degree of efficiency and in a number of cases shells are put into service just as they come from these rolls, the outside diameter, together with such tolerances as may be permissible, being so close as to require no further machining on the face. This leaves a hard skin due to the quicker chilling of the metal on the periphery, which is very useful in grinding and enhances the life of the shell considerably. The inside of the shell is left rough if it is to be secured to the center of the rolls by wooden wedges for wet grinding. If the bore of the ring is to be machined either straight or tapering, proper allowance is left for that

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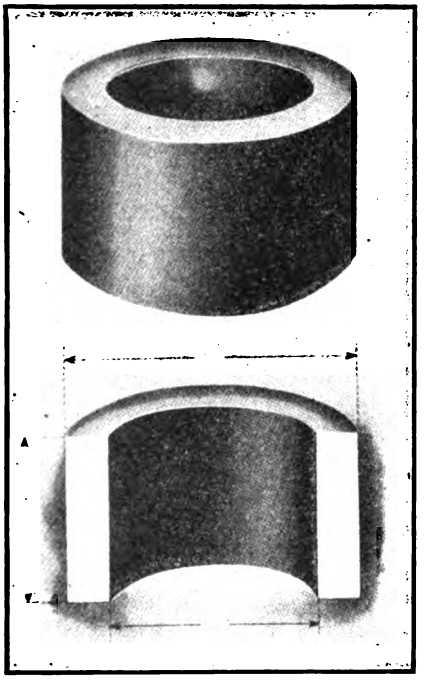

FIG. 5.—STRAIGHT BORE.

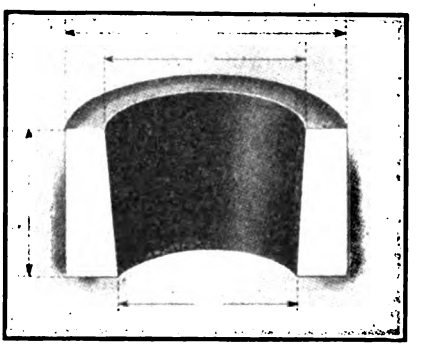

FIG. 6.—SINGLE-TAPER BORE.

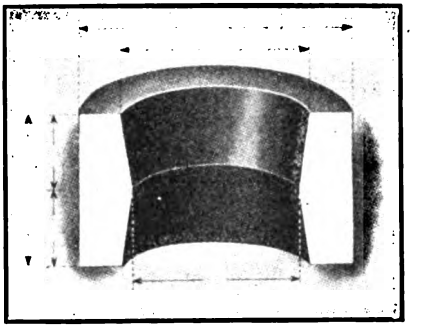

FIG. 7.—DOUBLE-TAPER BORE.

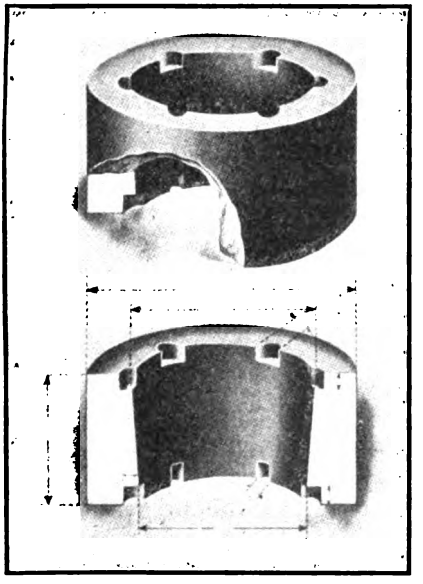

FIG. 8.—RECESSED FOR DRAW BOLTS.

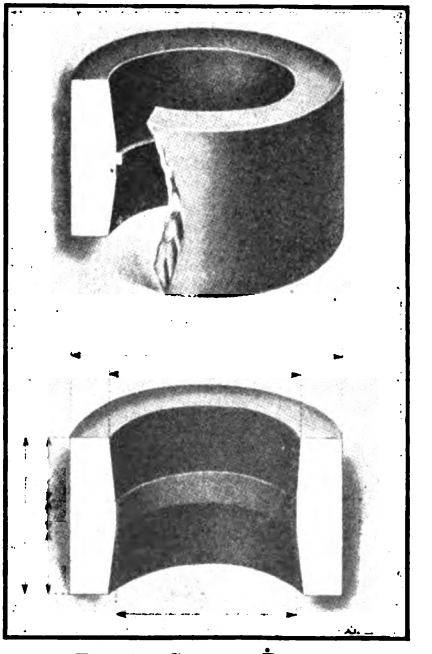

FIG. 9.—SPECIAL BORE.

FIGS. 5 TO 9.—PREVAILING TYPES OF BORES OF ROLL SHELLS.

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purpose on the inside, but the outside or wearing face of ring, being finished practically smooth by the rolling operations, should preferably be left in that condition, by reason of the advantage mentioned above.

Figs. 5 to 9 illustrate some of the prevailing types of bores of shells as used in rolls manufactured by the principal mining-machinery builders in the United States and are usually designated as follows: Fig. 5, Straight bore; Fig. 6, Single-taper bore; Fig. 7, Double-taper bore; Fig. 8, Recessed for draw bolts; Fig. 9, Special bore.

With regard to the rolling of shells, there would seem to be a limit of efficiency governed by the width and thickness of shells. Manifestly a mass of metal thicker than 5 in. cannot be sufficiently worked by the pressure of rolls, unless the rolls can be made very much more powerful than those in use at present, particularly when the combination of rolling and pressing naturally has its relative limits. Then again, the width, regardless of thickness, would have its limit at about 15 or 16 in., because a very wide mass, even thinner than 5 in., would experience the same difficulty in being thoroughly worked. It may therefore be assumed that shells over 5 in. thickness and wider than 16 in. cannot be rolled advantageously in a tire mill and produce homogeneous and dense metal, which will give good service and wear.

The following gives the limits of sizes for rolling as used in the practice of one of the largest tire manufacturers in the United States: Roll shells 8 to 10 in. wide, 108 in. outside diameter by 3 in. thick; over 10 to 16 in., or under, 68 in. outside diameter by 3 in. thick.

Shells over 16 in. wide and thicker than 5 in. and weighing say 5,000 lb., or over, are produced to the best advantage under a hydraulic forging press of adequate power, as a strictly forged, instead of a rolled product. Sufficient stress cannot be laid upon the necessity of thoroughly working the metal in these forgings all the way to the center by the use of machinery of this kind, which is sufficiently heavy and powerful to perform the necessary forging operation and is better able to do so than rolls. The writer is convinced that most of the cases of uneven wear in roll shells of extreme widths, where not due to improper setting or improper mill feeding, or both, is largely owing to the fact that the metal is not of uniform density, resulting from being worked in a tire mill, which was too light for the heavy and thick section passing through the rolls.

### *Life of Shells*

The limits of this article do not permit as exhaustive a treatment of this sub-heading as the writer would like to submit, based on data gathered during the past 13 years in furnishing the mining public in the West with steel roll shells and material of kindred nature, but for the sake of generality as well as brevity the following examples taken from users of roll shells in the widely separated states of Montana, Colorado,

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Sonora, Arizona and Utah will illustrate what may be considered good average records of the life of steel roll shells and Chilean mill wearing parts.

*Mill No.* 1 (*Montana*).—The mills used were 6-ft. Evans-Waddell and 6-ft. Monadnock type, being fed by revolving distributors in the center. The ore crushed was very hard with about 65 per cent. silica. The size of feed was 3 mm. and discharge, 16 mm. The rate of crushing was 80 tons per 24 hr. or 2,400 tons per month. The life of the tires was 4 months per set of three, running continuously; the life of dies, 2 months, running continuously. The amount of rock crushed per pound of steel consumed was 1.06 tons.

*Mill No.* 2 (*Colorado*).—At this plant, 6-ft. Evans-Waddell mills, fed by revolving distributors at the center, were used. Roasted ore, assaying 60 per cent. silica was fed, sized as follows: On 5/32, 4 per cent.; on 10 mesh, 15; on 16 mesh, 16; on 20 mesh, 7; on 30 mesh, 11; and through 30 mesh, 47 per cent. The discharge gave the following screen analysis: On 30 mesh, 1 per cent.; on 40 mesh, 13; on 60 mesh, 20; on 80 mesh, 12; on 100 mesh, 17; on 150 mesh, 7; and through 150 mesh 30 per cent. Crushing was at the rate of 173 tons per 24 hr. The average life of tires and dies determined by life of dies was: Steel maker No. 1, 183.3 days; steel maker No. 2, 164.2 days. The average tonnage crushed per set: Steel maker No. 1, 31,711 tons; steel maker No. 2, 28,407 tons. The average weight of shell, 1,488, set (3), 4,464 lb.; of die, 3,029 lb.; total, 7,493 lb. The number of tons crushed per pound of steel consumed was: No. 1, 4.25 and No. 2, 3.8, which gives the number of pounds of steel consumed per ton of ore crushed as follows: No. 1, 0.236 and No. 2, 0.264 lb. The average weight of scrap per set was 1,640 lb.

*Mill No.* 3 (*Sonora*)

| Dept. | No. of Pieces | Diameter, Inches | Face, Inches | Weight per Set, Pounds | Life Days | Tons Ore Crushed | Pounds Steel per Ton Ore | Size Feed | Size Discharge | |
|---|---|---|---|---|---|---|---|---|---|---|
| A | 10 | 36 | 16 | 13,615 | 270 | 308,000 | 0.044 | 1½ in. | 1 in. | Roll shells |
| B | 4 | 36 | 16 | 5,444 | 135 | 81,000 | 0.067 | 1 in. | ¾ in. | Roll shells |
| | 8 | 27 | 14 | 7,256 | 80 | 118,400 | 0.061 | ⅞ in. | 2 mm. | Roll shells |
| | 3 | 54 | 7 | 7,892 | 182 | 35,100 | 0.224 | ..... | ..... | Tires and die 6 ft. mill. |
| | 2 | 70 | 7 | | | | | | | |
| | 5 | 60 and 44 | 7 | 6,502 | 425 | 21,625 | 0.304 | ¾ in. | 1½ mm. | Tires and die 5 ft. mill. |
| C | 4 | 36 | 16 | 5,444 | 195 | 25,798 | 0.210 | 1 in. | ¾ in. | Roll shells |
| | 2 | 60 | 7 | 6,502 | 240 | 69,960 | 0.093 | ¾ in. | ..... | Die rings, Bryan mill tires. |
| | 3 | 44 | 7 | | | | | | | |

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## *Mill No. 4 (Arizona)*

Data on shells from two sets of 54 by 24 in. rolls (dry crushing). One shell, 54 by 24 in, (in two pieces, 54 by 12 in)

| | |
|---|---|
| Outside diameter, inches | 54 |
| Inside diameter, inches | 43⅛ |
| Original thickness, inches | 5 1/16 |
| Ultimate thickness, inches | 1 |
| Wear, inches | 4 1/16 |
| Original weight (two pieces), pounds | 5,408 |
| Scrap weight (two pieces), pounds | 1,130 |
| Loss, pounds | 4,278 |
| Tons through crusher | 1,266,000 |
| Tons crushed each set of rolls (Feed, 3 in. size, product, 1 in. size) | 538,000 |
| Loss of steel per ton crushed, pounds | 0.0159 |

*Note.*—These shells were never taken out of the machines since started. They were not entirely worn out but had been in use 2½ years. During that time they were trued up occasionally in the machine by attaching a lathe tool holder to the main frame and then turning the rolls backward by a back-geared electric outfit through the medium of the regular roll belts.

During the period that the two sets of rolls were in use they crushed 1,076,000 tons of ore from 3 in. size, which had been produced by a No. 8 McCully crusher (with the fines screened out *en route* to the rolls), to a product approximating 1 in. size. Approximately 25 per cent. of the crusher product was taken out by the screens ahead of the rolls.

## *Mill No. 5 (Utah)*

Record of Roll Shells in use at Sample Mill, June, 1914

| Section Number | Roll Size Inches | Thickness of Shell, Inches | Date Installed | Date Replaced | Period of Service, Mo. | Dys. | Estimated Tonnage | Size of Crushing, Inches | Cause for Change |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 42 by 15 | 5 | 8–1–12 | 2– 9–13 | 5 | 8 | 54,000 | ½ | Surface corrugated |
| 1 | 42 by 15 | 3½ | 2– 9–13 | 6– 8–13 | 4 | | 38,000 | ½ | Surface corrugated |
| 1 | 42 by 15 | 2½ | 6– 8–13 | 11– 6–13 | 5 | | 36,000 | ½ | Surface corrugated |
| 1 | 42 by 15 | 3½ | 11– 6–13 | 4–27–14 | 5 | 21 | 50,000 | ½ | Worn out |
| 1 | 54 by 24 | 5 | 8–28–13 | Still on | ..... | | 95,000 | ⅝ | |
| 2 | 42 by 15 | 3½ | 2– 3–13 | 5– 1–13 | 3 | | 27,000 | ½ | Surface corrugated |
| 2 | 42 by 15 | 5 | 5– 1–13 | 9–27–13 | 4 | 26 | 40,000 | ½ | Surface corrugated |
| 2 | 42 by 15 | 3½ | 9–27–13 | 12–26–13 | 3 | | 24,000 | ½ | Slipped off. |
| 2 | 42 by 15 | 3½ | 12–26–13 | Still on | ..... | | 58,000 | ½ | ..... |
| 2 | 54 by 24 | 5 | 6–28–13 | Still on | ..... | | 116,000 | ⅝ | ..... |

Material in 42 by 15 in. shells is rolled tire steel, the 54 by 24 in. shells are forged throughout.

Four sets of rolls are provided for two machines, and as a different set of shells is placed in machine at each replacement it is not possible to secure tonnage figures for the entire life of any one set of shells.

Lead ores, limerock, slag, iron ore, etc, through No. 1 section from 40 per cent. of the total tonnage through the mill.

Copper ores, the greater percentage of lead matte, lining ores, etc., go through No. 2 section, forming 60 per cent. of the total crushed.

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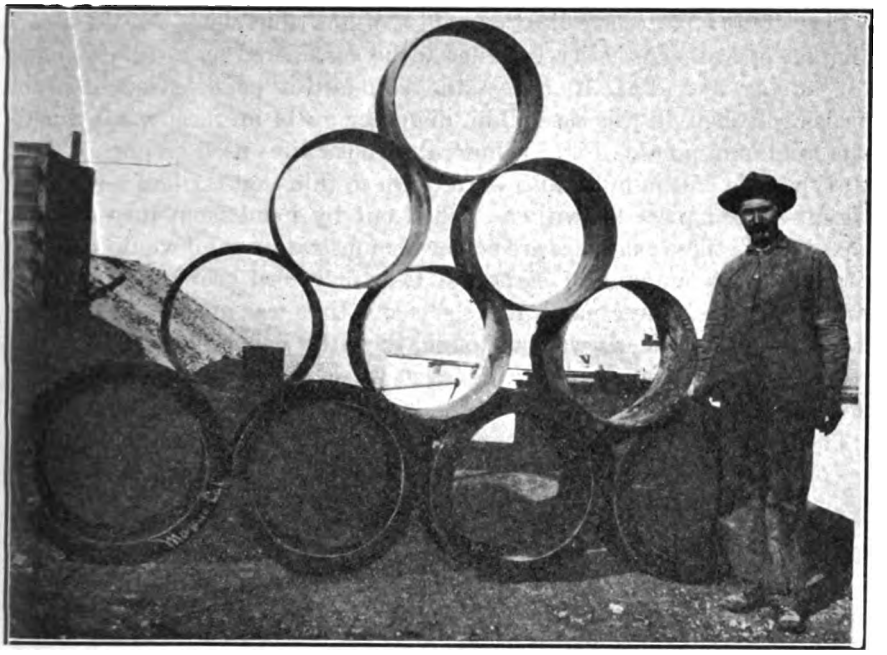

FIG. 10.—CORNISH ROLL SHELLS SHOWING NEW SHELLS AND THOSE WORN TO MINIMUM THINNESS.

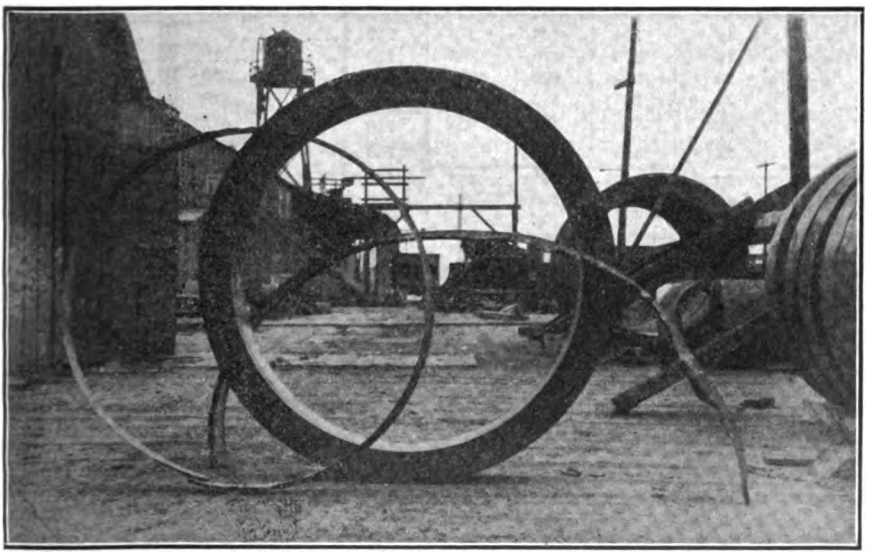

FIG. 11.—CHILEAN MILL TIRES. SHOWS SHELL WORN SO THIN THAT CRESCENT-SHAPED PIECE WAS PULLED OUT BY HAND FROM ITS CIRCULAR SHAPE.



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Figs. 10 and 11 are excellent examples of how thin shells should wear in actual operating practice in order to be considered as most efficient and satisfactory. Fig. 10 shows the standard type of straight-bore Cornish roll shell, in this case 30 in. diameter by 14 in. face, worn down to its minimum point. Fig. 11 illustrates rolled tires used on one of the best types of Chilean mills, also worn down so thin that the half-round or crescent-shaped piece shown was pulled out by hand from its circular shape. Both these examples are ocular demonstrations and would prove, more than any amount of elaborate tests or record keeping, that the material of which these particular shells and tires was made, was homogeneous and durable as well as tough, the latter being indicated by its strength and resistance to breaking, even when worn to the thin section shown.

The tabulated list of weights compiled by the writer, for the various sizes of roll shells given below, may be of interest in a general way in connection with this article.

WEIGHTS OF ROLLED-STEEL SHELLS

(*Straight Bore*)

| | Diameter in Inches | Face | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | 6-in., Pounds | 8-in., Pounds | 10-in., Pounds | 12-in., Pounds | 14-in., Pounds | 15-in., Pounds | 16-in., Pounds | 18-in, Pounds |
| 2½ in. thick | 12 | 126 | 168 | 210 | 252 | ..... | ..... | ..... | ..... |
| | 18 | 205 | 270 | 335 | 410 | ..... | ..... | ..... | ..... |
| | 20 | 230 | 300 | 375 | 460 | 630 | 675 | ..... | ..... |
| | 24 | 285 | 380 | 475 | 570 | 780 | 840 | ..... | ..... |
| | 26 | 310 | 415 | 520 | 730 | 850 | 915 | 975 | ..... |
| | 27 | 325 | 435 | 540 | 760 | 890 | 1,135 | 1,300 | ..... |
| | 30 | 430 | 575 | 715 | 860 | 1,005 | 1,375 | 1,465 | 1,650 |
| 3 in. thick | 36 | 525 | 700 | 870 | 1,050 | 1,225 | 1,690 | 1,805 | 2,030 |
| | 40 | 590 | 785 | 980 | 1,180 | 1,375 | 2,000 | 2,030 | 2,285 |
| 4 in. thick | 42 | 805 | 1,070 | 1,340 | 1,610 | 1,880 | 2,015 | 2,145 | 2,415 |
| | 44 | 850 | 1,130 | 1,415 | 1,700 | 1,985 | 2,125 | 2,270 | 2,550 |
| | 48 | 930 | 1,240 | 1,550 | 1,860 | 2,190 | 2,345 | 2,500 | 2,790 |
| | 54 | 1,060 | 1,415 | 1,770 | 2,120 | 2,475 | 2,655 | 2,830 | 3,180 |
| | 56 | 1,100 | 1,470 | 1,840 | 2,200 | 2,570 | 2,755 | 2,940 | 3,300 |

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# The Occurrence, Preparation and Use of Magnesite

BY L. C. MORGANROTH, PITTSBURG, PA.

(Pittsburg Meeting, October, 1914)

## *Magnesite both Massive and Crystalline*

MAGNESITES are of two general classes—massive and crystalline. Massive magnesite occurs in serpentine, being formed by the breaking down or decay of serpentine-making minerals or rocks as olivine, etc. (All the magnesia contained by the olivine is not liberated, as a portion goes to make up the serpentine; the breaking up of this serpentine later forms more magnesite.) Serpentine in various stages of transformation is found in the vicinity of magnesite deposits.

*Massive magnesite* is very widely distributed. It is found in Greece, California, Africa, India, Australia, Mexico, South America, and deposits of varying sizes occur in several of the European countries. Those on the Island of Euboea, Greece, are probable the most important deposits of this character of magnesite. They have been worked for a number of years, the principal quarries or mines being 3 miles in the interior from the village of St. George. (St. George is on the west coast of the island, between the towns of Limni and Khalkis.) (Figs. 1 and 2). The magnesite occurs in veins or lenses in serpentine. Some of these lenses are very large; one worked several years ago was approximately 50 ft. in thickness, 75 to 100 ft. high and 300 ft. or more in length. The magnesite is pure white and makes a very prominent feature of the landscape.

While massive magnesite is widely scattered over the world, many of the deposits are in veins too thin to work. The usual mode of occurrence is in the form of stock work, the deposits in Venezuela, Mexico, and many of those in California, being of this form. The veins vary in thickness from the thinness of paper up to several inches, running in all directions, occasionally leading to a mass or nucleus of some size. The spaces between the veins are filled with a disintegrated serpentine in the form of sand. Several of the deposits in California would probably be workable, if they were more favorably situated with reference to transportation, but by far the majority of the deposits are too poor to work. At one deposit,

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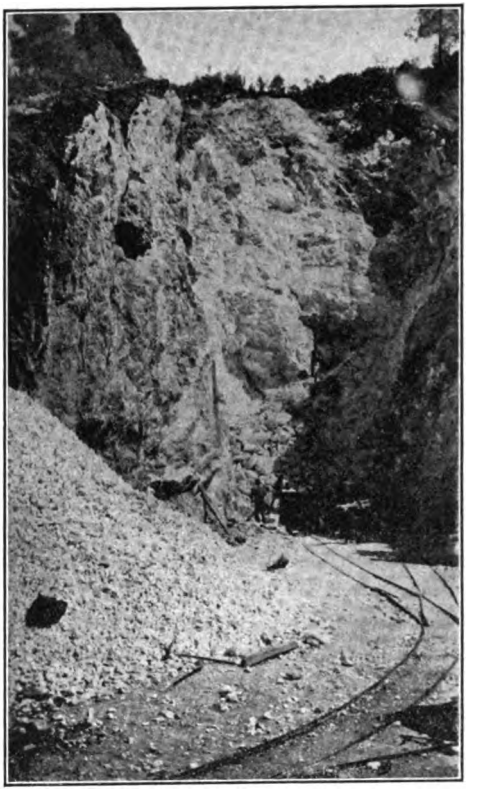

FIG. 1.—MAGNESITE QUARRY ON ISLAND OF EUBOEA, GREECE.

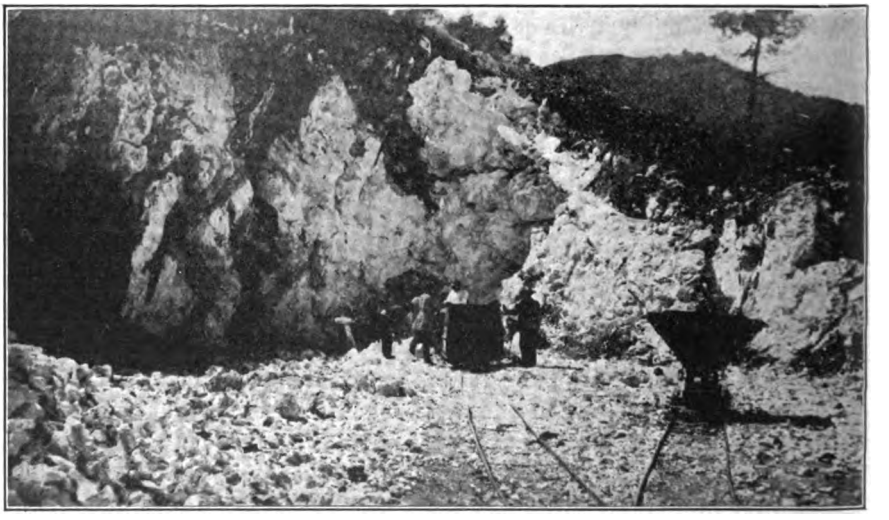

FIG. 2.—MEN AND WOMEN WORKING IN MAGNESITE QUARRY, ISLAND OF EUBOEA, GREECE.

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worked several years ago, they were handling 4 tons of refuse to 1 ton of magnesite.

In the preparation of this magnesite for use the process is practically the same as that which will be described under preparation of the crystalline variety; with the exception that the magnesite is not dead-burned being burned only to what is known as the "caustic" state. In this form it contains 2 to 5 per cent. $CO_2$. The reason for this is that caustic magnesite is used for a different purpose than that of the crystalline.

### *Crystalline Type Found only in Austria-Hungary*

*Crystalline magnesite* is found practically only in Austria and Hungary. It occurs in dolomite to which it owes its origin. The steps in the formation of the magnesite are represented as follows:

Magnesium bi-carbonate penetrating limestone forms dolomite. A further percolation of magnesium bi-carbonate in the dolomite forms magnesite; later, the more soluble calcium carbonate is washed out. The finding of a large number of crinoidal segments in the magnesite and dolomite attests to limestone being the origin. This magnesite is rarely pure, nearly all containing carbonate of iron. Iron solutions were penetrating the limestone at the same time as the magnesium bi-carbonate; in fact, it is considered that the magnesite occupies one end of a chain of minerals, and iron carbonate, siderite, the other. (Important iron-ore deposits are found in the same dolomite bed.)

Metamorphosis probably plays an important part in the structure of magnesite. It is probable also that this has a bearing aside from the chemically combined iron on the particular adaptability of this magnesite for use in the metallurgical industries. Crystals when examined under the microscope show distortion due to pressure from the folding of the beds, also indicating that various changes have taken place after crystals were formed. The magnesite from its crystalline structure is sometimes known as pinolite magnesite. The crystals of magnesite are often inclosed in a thin skin of talc, which accounts for the $Al_2O_3$ and the $SiO_2$ found in all analyses. Talc is sometimes found in large amounts in the same beds, but there is no direct connection between magnesite and talc, except that they occur in the same limestone beds. The talc is generally found between the magnesite and the limestone. At one locality in Hungary a quarry originally worked for talc was later operated for magnesite.

Geologically the crystalline magnesite probably belongs to the Carboniferous period. Carbonaceous slates are found with it and small isolated veins of coal have been noted, while graphite is mined at various points. The basal bed is a black clayey slate.

The crystalline variety of magnesite differs markedly from the massive. The massive is a white chalky-colored mineral, but the crytalline is a blue-

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gray or drab, or rather a mixture of crystals of various shades of grays and drabs. Both the massive and the crystalline magnesite may sometimes have a yellowish tinge, due to surface stains. The massive variety has a rough, earthy fracture; the harder varieties break with sharp edges and a more or less concoidal fracture. The crystalline varieties break more irregularly. The crystalline variety may have pieces of black slate scattered through it giving it a decidedly mottled appearance. Dolomite in the form of splotches or veins shows distinctly light or white against the gray or drab background.

The beds in which the magnesite is found in Austria and Hungary follow more or less a northeast and southwest line that would about cross between the cities of Vienna and Budapest. The valleys of the Danube and its tributaries divide these deposits into two groups. The Austrian group extends across the province of Steiermark, with its western extremity near the city of Salzburg and its farthest eastern deposit near the town of Gloggnitz. The biggest deposit in this group is near the town of Veitsch. The Hungarian deposits have their western extremity near the village of Divin; the eastern extremity is not clearly defined and may extend into Russia. The largest deposits in Hungary are midway between the towns of Jolsva and Nyustya. It is not to be inferred that magnesite is found throughout the entire district between these extremities in either country. The magnesite occurs only in isolated lenses, and while some of these lenses are very large, many are too small to be worked profitably. It is probable that commercially workable deposits in Austria and Hungary do not exceed a dozen in number.

### *Mining and Preparation*

The method of mining and preparing magnesite in Austria and Hungary is practically the same in all plants. Following is a description of the Veitsch operation, as this deposit is the oldest and one of the largest found in Austria-Hungary. The deposit is a huge lens found in an isolated hill surrounded on all sides by barren rocks. It is on the outskirts of the village of Veitsch, which is about 65 miles southwest from Vienna. The entire top of the hill is magnesite. From the top to the base of the magnesite is from 700 to 800 ft., the bottom of the magnesite being about half way down the hill. The quarry is worked in a series of steps or levels about 50 ft. apart vertically, which are clearly indicated in Fig. 3.

The material is blasted out of the solid in the ordinary manner of rock quarrying. The magnesite is broken to one-man pieces, or less, and any particles of dolomite and quartz carefully sorted out. In the best deposits there is always a large amount of this gangue. Practically one-half of what appear to be levels of the quarries are spoil banks. The magnesite, loaded on tram cars, is lowered by gravity planes

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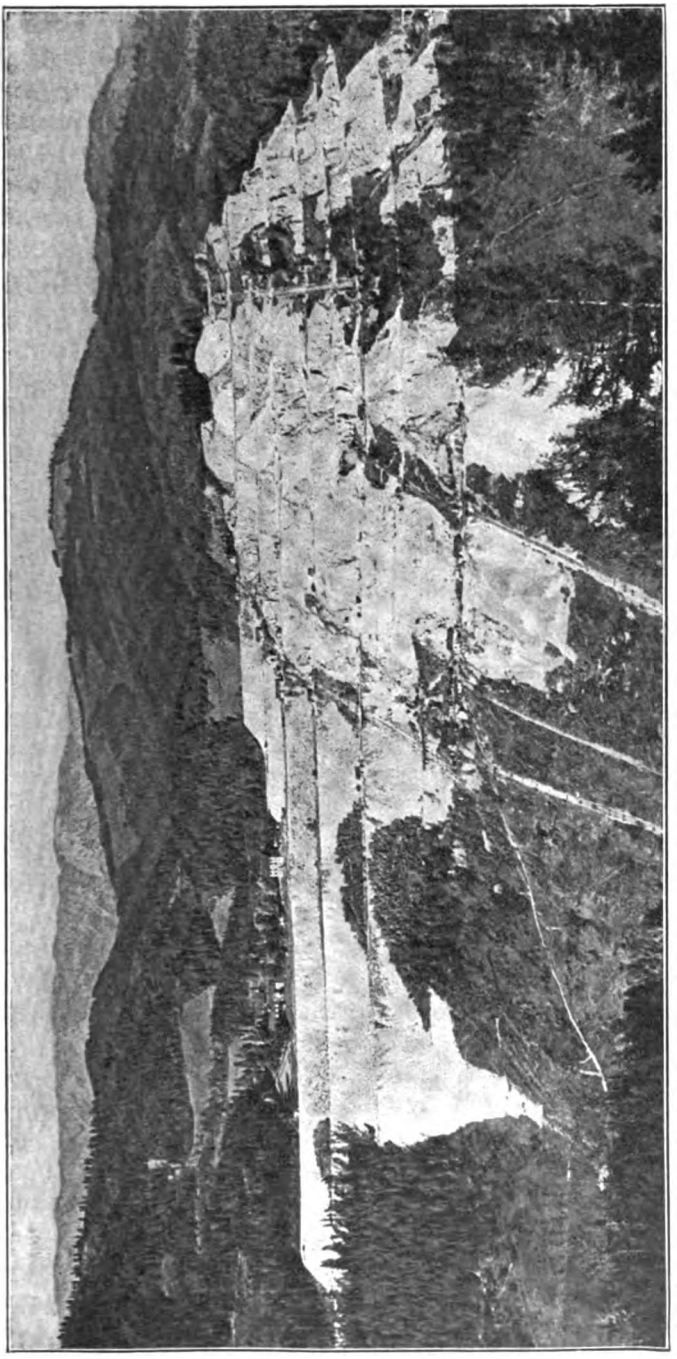

FIG. 3.—FAMOUS MAGNESITE QUARRY AT VEITSCH, AUSTRIA.

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to the base of the hill, where the works are situated. Here it is burned in continuous kilns of the bottle variety. These kilns burn on the average 12 to 15 tons in 24 hr. As a rule, producer gas is used as fuel. Within the last few years several plants have installed kilns of the rotary or cement type to burn the magnesite, powdered coal being used as fuel. The capacity of these rotary kilns is probably 50 to 60 tons in 24 hr. The magnesite can be burned as thoroughly in these kilns as in the bottle kilns, but they have one disadvantage, in that a larger percentage of fines is produced.

The magnesite as burned in the bottle kilns is drawn about every 6 hr. (Fig. 4). It is quenched with water and then passes through a crusher, which reduces it to the size of a walnut or less. It is then screened, being sepa-

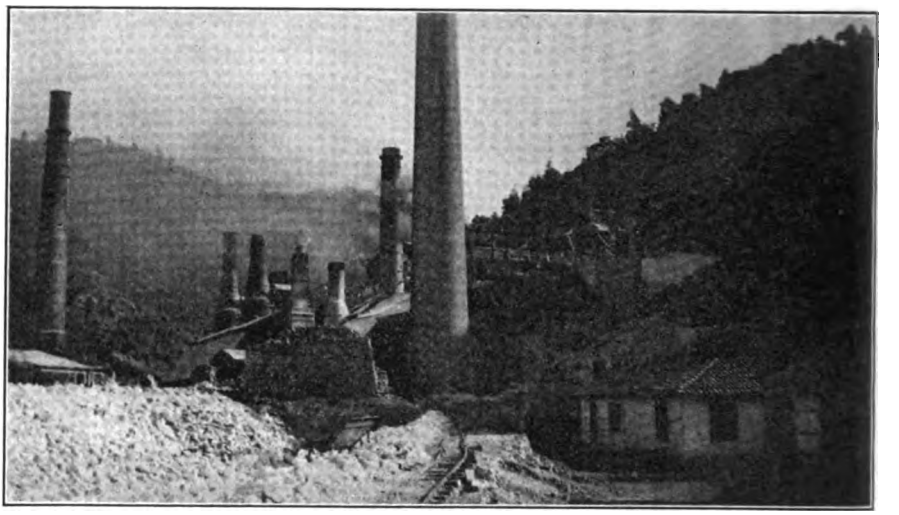

FIG. 4.—CALCINING PLANT, ISLAND OF EUBOEA, GREECE; CALCINED MAGNESITE IN FOREGROUND.

rated into three sizes. From the screens it goes to the picking tables, where the underburned pieces of magnesite, together with any dolomite or quartz, which were too small to be removed at the quarries are picked out. The largest sizes of magnesite are again crushed and these smaller pieces are re-picked. The magnesite finally is crushed to the size of corn, again picked over, and then put in sacks holding from 150 to 200 lb. In this shape it is shipped to the user. Magnesite in burning loses about half its weight in the passing off of the $CO_2$. No effort is made to save this gas. Within recent years an economy has been effected in the picking or sorting of magnesite by the installation of magnetic separators. Magnesite containing iron is readily separated in this manner.

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### *Distinct Type in Norway and Sweden*

A distinct type of magnesite is found in Norway, near the town of Snarum. It occurs as small veins in serpentine, but unlike other magnesite in serpentine it is crystalline. The crystals are small and uniform in size and are distinctly different from the crystalline variety of Austria-Hungary. It is white or tinged green or yellow from the particles of serpentine which are more or less intimately mixed with the magnesite. At present there is only one deposit worked, and this in a very small way. It is doubtful if they will ever amount to much commercially, on account of the limited amount of magnesite of good quality.

In Sweden similar deposits are found. On account of their situation, operating and transportation charges would be excessive and it is doubtful if they will ever be able to compete with the much cheaper magnesite of Austria-Hungary.

### *Uses of Magnesite*

Magnesite is used in three forms: Crude, or unburned; burned to the caustic state; and dead-burned.

*Crude Magnesite.*—The principal use of crude magnesite has been in the manufacture of carbon dioxide gas. The use of magnesite for this purpose is decreasing rather than increasing, due to the fact that the manufacture of gas from this material can only be a financial success if the residue of burned magnesite can be disposed of. This residue contains too high a percentage of $CO_2$ for use even as caustic magnesite. It is also too impure to be used in the manufacture of various forms of magnesia used medicinally.

*Caustic Magnesite.*—This is magnesite that is not thoroughly calcined. As it comes from the kilns it may contain 3 to 4 per cent. of $CO_2$. It is burned at a temperature of about 1100° C. In this form when mixed with magnesium chloride it forms an exceedingly strong cement, which is used in the manufacture of artificial marble, flooring, etc. Various fillers such as talc, sand, saw dust, etc., are used. These cement mixtures are known under various names, such as "sorel cement," "oxychloride cement," etc. Probably over 90 per cent. of the white or massive magnesite is used in the caustic state for the above purposes. Caustic magnesite deteriorates rapidly by the absorption of $CO_2$ and moisture from the air, so that in the course of 4 or 5 months it will contain two or three times as much gas as it did when it first came from the kiln.

*Dead-burned Magnesite.*—This is magnesite that contains less than 1 per cent. of $CO_2$ or that has been so thoroughly calcined that there is no deterioration or reversion on exposure to the air by the absorption of $CO_2$. It is this form of magnesite that is used in the making of the

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bottoms of open hearths and for the manufacture of magnesite brick. The crystalline variety of magnesite is used exclusively for this purpose and Austria-Hungary may be said to furnish it all. (It is a difficult matter to dead-burn the white or massive variety, due to the fact that it decrepitates or burns to powder before all of the gas is driven out. In Greece, where formerly a small amount of dead-burned magnesite was produced, it was only possible to dead-burn it by including lumps of magnesite that had small pieces of serpentine adhering to them. In the kiln this serpentine fusing held the magnesite in more or less lump form and allowed of a more thorough burning.) Austro-Hungarian magnesite on account of its containing chemically combined iron, is much more readily calcined to the dead-burned state. It is also on account of this iron content that it is far more suitable in the making of open-hearth bottoms and in the manufacture of brick, for the iron fusing causes the magnesite to set. It may be possible to get the same results by using the purer massive magnesite by mixing some iron with it, but dead-burned white massive magnesite cannot compete with the Austrian or Hungarian variety in point of price. A few magnesia brick are made from the massive variety where a brick of special purity is desired or where a more highly refractory brick than one made from the Austrian or Hungarian magnesite is required, as in certain metallurgical processes or in electric furnaces.

The Austro-Hungarian calcined magnesite received in this country is in the form of grains, ranging in size from particles the size of corn to powder. Approximately 50 to 60 per cent. of the magnesite is used in this form, *i.e.*, is not manufactured into brick, this use being in the making of bottoms of open-hearth steel furnaces. Although the bottoms of the furnaces are made of magnesia brick, the bath is not allowed to come directly in contact with the brick, a bottom of 15 to 20 in. of magnesite grains intervening. This bottom is made by sintering the magnesite grains in place in layers of ½ to 1 in., until the requisite thickness is obtained. From time to time erosions occur in the bottom and along the side walls, which are repaired by putting in more magnesite grains.

Magnesite bricks are used in electric furnaces, melting, heating, welding furnaces, etc. During the last couple of years the adoption of magnesia brick for linings of copper converters has become almost universal. The siliceous lining good for three or four blows, formerly in use, has now been replaced by magnesia brick lasting a year or two and good for 15,000 to 20,000 tons of matte.

The United States Government reports a total consumption of magnesite for the year 1912 as 103,000 net tons.

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# Barytes as a Paint Pigment

BY H. A. GARDNER,* AND G. B. HECKEL*

(Pittsburg Meeting, October, 1914)

THE principal use of barium sulphate is as an inert paint pigment. For this purpose, the ground material is used both in its natural and in its artificial forms. Probably the largest amount is used in the production of a chemically precipitated pigment called lithopone which consists of 70 parts of barium sulphate and 30 parts of zinc sulphide. An enormous tonnage of this pigment is at the present time being used in the manufacture of interior flat wall paints of the oil type, such paints having almost entirely supplanted the use of corroded white lead which was at one time used for interior painting. The more dense nature, lower price and sanitary value of lithopone was responsible for this change. Lithopone is, moreover, used to a large extent in the manufacture of oil cloth, shade cloth and linoleum. Its whiteness makes it of particular value for this purpose. Barium sulphate is also used of itself in admixture with other pigments in the manufacture of prepared paints. The chemically precipitated form of barium sulphate, which is generally called *blanc fixe*, is also used for this purpose.

Much controversy has raged in the paint world over the question of inert pigments such as barytes, silica, asbestine and china clay. These pigments are probably called inert because they have no chemical action upon the linseed oil in which they are ground to produce paints, and on account of the fact that they become transparent in oil. It is well known that in the manufacture of paints, opaque white pigments, such as basic carbonate white lead, basic sulphate white lead, zinc oxide etc., are generally used as the grinding base. These base pigments, when ground in oil, react therewith to some slight extent, and they produce white paints which have excellent hiding or obscuring value. The inert pigments so-called, when ground in oil and spread out into a thin film, form transparent coatings which have no hiding power. It has been contended, therefore, that the use of barytes or similar inert pigments in paints is resorted to simply for the purpose of cheapening the cost of the product.

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It is, of course, true that the natural tendency of manufacturers in the past has been to use inert pigments liberally, on account of their low cost, and the consumer has in consequence frowned upon this practice and has been prone in some instances to condemn the use of paints which contain even small amounts of inert pigments. That such inert pigments do have a proper and legitimate place in the manufacture of prepared paints, has been shown by practical tests. The degree to which they may be used, however, is a question which some believe not to have been fully determined. Several years ago, at the laboratories of the Pennsylvania Railroad Co., in Altoona, Pa., Dr. Charles B. Dudley conducted a long series of carefully made paint tests on a practical scale. From the results of these tests, Dr. Dudley formulated a series of fundamental principles governing the quantity and use of inert pigments in prepared paints. Briefly paraphrased, these principles may be stated as follows:

In a white paint, the desired color having been provided for by the use of opaque pigments, there may then be added to the paint as much inert pigment as can be used without destroying the practical opacity of the paint. In a white paint made of lead and zinc pigments, the amount of inert pigment to be added may probably run as high as 20 per cent., but a more dense paint will be obtained by the use of not over 10 to 15 per cent. In a colored paint, such as an iron oxide red, there may be used as high as 50 per cent. of inert pigments, without detracting from the color of the paint. In a colored paint such as a chrome green, there might be used as high as 80 per cent. of an inert pigment.

The use of barytes in the manufacture of chrome green conspicuously illustrates the function of an inert pigment. Chemically pure green is prepared by simultaneous precipitation of lead chromate and Prussian blue. This pigment is used as a tinting color, and when it is used alone it is apt to produce a dense, dingy green. In order to make a more satisfactory green, the color is precipitated upon a base of barium sulphate in the proportion of 20 parts of color to 80 parts of barium sulphate. Such a color is brilliant and clear, producing bright shades when added to a white base. This statement also holds true with other colors, especially the adjective dyes precipitated on a barium sulphate base. The para-nitraniline reds and other permanent vermilions may be cited as examples of brilliant colors which are now precipitated in the course of manufacture upon a base of barium sulphate or similar inert pigment. The legitimate office, therefore, of barytes as an inert pigment in the manufacture of colors is to diffuse the color by its presence and brighten and make more clear the tints which are prepared from it as a base.

It is a fact, which at first glance appears paradoxical, that the incorporation of small amounts of these transparent, colorless, inert pigments in a paint containing very finely divided opaque pigments, such as zinc oxide, lampblack, etc., increases the opacity of the paint coatings made from them. The reasons, however, are obvious. In the first

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place, the proportion of liquid in the vehicle of the film is reduced, on account of the fact that barytes grinds to a workable base with a very small amount of linseed oil, whereas many other pigments used in the manufacture of paints require a more substantial quantity of oil. It is the amount of oil in a paint which materially affects its hiding power. Moreover, the thickness of a paint coating is to a great extent increased when it contains an inert pigment such as bartyes, and the thicker the coat the more dense will be the paint.

The chemical effect of inert pigment is also a matter of great importance to the painter. This effect varies with the structure of the pigment. In the case of pigments such as bartyes, silica, etc., the effect is to impart "tooth" under the brush. "Tooth" is an expression used to describe the brushing action of a paint, which indicates that the paint is being taken up by the wood or other surface to which it is applied. Paints made solely from finely divided pigments such as zinc oxide or lampblack would be deficient in tooth.

From the foregoing, it is apparent that there should be no technical objection to the use of bartyes in paints. The only question is an economic one, and involves two considerations: The first is that barytes and other inert pigments shall not be sold except under their true names and shall not be used as adulterants, using that term in the sense given it by the American Society for Testing Materials, as follows: "Adulterant—A substance partially substituted for another." The other consideration is that the amount of bartyes shall be such that it will not impair the opacity of the paint under the conditions of application, which implies that three coats shall effectively hide the underlying surface to which it is applied.

*Data on Barytes*

Occurrence: As rock barite and as the gangue in zinc and lead ores.

Principal sources: Missouri, Kentucky, Virginia and Tennessee.

Structure: Natural, crystalline; artificial, amorphous.

Color: White, transparent.

Usual method of production: "Gophering."

Principal use: As a pigment.

Preparation of pigment: Breaking, comminution, washing with acid to remove iron, washing, grinding, water floating or sedimentation.

Specific gravity of pigment: 4.45.

Formula: $BaSO_4$.

Desirable qualities of barytes as a paint pigment: Softness, whiteness, fineness.

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# Quarrying Shale by the Tunnel System

BY DWIGHT T. FARNHAM,* SEATTLE, WASH.

(Pittsburg Meeting, October, 1914)

## *Description of Quarry*

THE shale used at the Renton plant of the Denny-Renton Clay & Coal Co., for the manufacture of vitrified paving brick occurs in a hill rising from 200 to 300 ft. above the level of the valley in which the plant is situated. The shale deposit, which is of unknown thickness pitches about 10° to the southeast. The 150-ft. face worked extends approximately east and west, so that the dip, as shown by the photograph, is to the left and into the hill. The shale is overlain with about 50 ft. of gravel, 70 ft. of clay, and 30 ft. of mixed loam and gravel, making altogether from 125 to 150 ft. of overburden.

## *Disposal of Overburden*

The overburden is removed by sluicing. For this purpose a No. 2 Joshua Hendy giant is used, throwing a 1½-in. stream, and having 60 lb. pressure at the nozzle. The water is drawn through an 8-in. pipe from Cedar river, about 1,000 ft. distant, by means of a Worthington four-stage centrifugal pump, direct connected with a 100-h.p. motor placed just south of the factory, and 1,500 ft. from the nozzle This plant handles the material satisfactorily to a height of about 75 ft. above the level of the top of the shale. When the higher levels are worked, an auxiliary plant consisting of a four-stage Byron Jackson centrifugal pump with a 100-h.p., belt-connected motor, is thrown into the system. This plant is situated on top of the shale and can be discerned at the extreme right of the trestle work in Fig. 1.

The overburden is worked in either two or three benches, depending upon the height of the hill at that particular point. Fifty feet is about

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the extreme height which can be worked at one time with any degree of efficiency with the plant.

The usual method of working is as follows: A cut is made about 50 ft. in height and 100 ft. in depth, extending completely across the face, which is about 800 ft. long at present. The material is shot down, men being lowered over the brink by ropes, when necessary, to do the drilling. The clay, loam and gravel fall in large lumps, some of which need further reduction, depending upon the nature of the material.

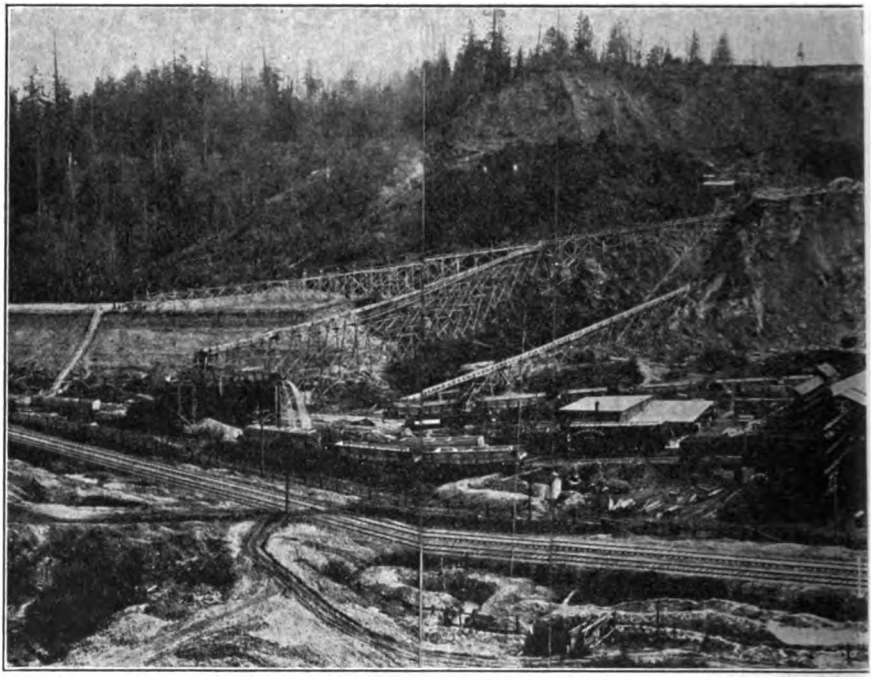

FIG. 1.—SHALE QUARRY AND

The stream from the giant reduces these to a slurry containing lumps up to perhaps 4 in. in diameter. Once a cut is finished, the bank is sloped to an angle of about 45°, and the next cut below started and completed in the same manner until the shale is uncovered.

Fig. 2 shows the stream playing on the 50-ft. gravel strata which overlies the shale.

Sluicing operations are carried on continuously. The men work in three shifts, arc lights, and 5,000-c.p. acetyline headlights being used at night. In this way 19,000 to 20,000 yd. of overburden are moved each month.

The material is removed from the face by means of ground sluices which connect with the boxes supported on the trestles shown in Fig. 1.

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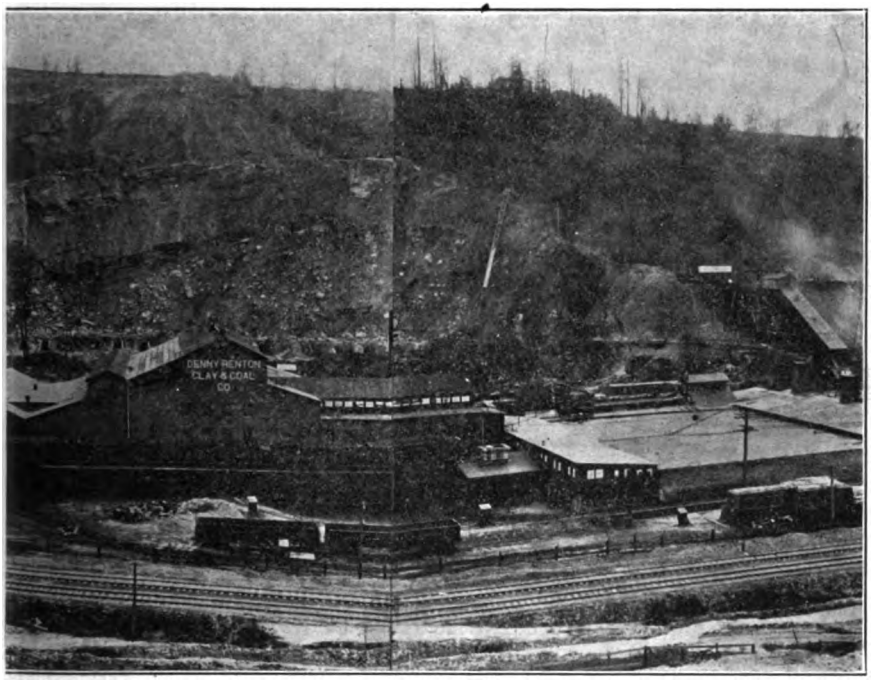
These boxes are 10 by 12 in. inside, and the bottoms are lined with vitrified brick, which outlast any other material. When running gravel, which gives the hardest wear, iron boxes last 90 days, wood blocks six weeks, while the brick remain in service from 10 months to a year.

Gravel which is suitable only for ballast is run directly into railroad cars by means of a sluice box as shown in Fig. 1. To the left of this can be seen the gravel trestle which leads such gravel as is suitable for concrete work to a washing and screening plant. Enough gravel is sold




PAVING BRICK PLANT AT RENTON.

to reduce the cost of removing the overburden 12 per cent. The trestle to the left carries the boxes which remove the clay and loam and other unsalable material. The larger lumps, at the time the photograph was taken, were settled behind the large dike just across the railroad tracks, while the finer material in the form of slurry was run underneath the tracks and settled in a series of basins.

Great care is taken to prevent any of this material from flowing into Cedar river, which is being made a navigable stream.

Large rocks which cannot be sluiced, are shot and allowed to remain on top of the shale until the shale is used, when they are taken out over the floor of the quarry.



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*Former Quarrying Methods*

The quarry floor is perhaps 10 ft. above the level of the factory. The shale, which is used in the manufacture of vitrified paving brick at the rate of 700 tons per day at present, presents a face 700 ft. long and 150 ft. high.

Various methods of working have been tried, but none have proved so successful as the present one. A steam shovel is, of course, out of the question with so great a lift. Only about 30 ft. of this material could be

FIG. 2.—REMOVING OVERBURDEN BY HYDRAULICKING.

worked efficiently by shovel, and the vision of five steam shovels working one above the other is too appalling to contemplate when only about 700 tons of shale a day is to be moved.

We did try, however, a ½-yd. orange-peel bucket and a locomotive crane. With this outfit an attempt was made to pick up the shale, once it had been shot down, and to drop it into cars which were dumped over a grizzly having bars spaced 6 in. apart, which allowed the fine material to fall directly into the conveyor, while the coarse material was led to a crusher. This proved unsatisfactory on account of the large blocks of shale, all of which had to be shot and re-shot. Furthermore, the bank is made up of alternate layers of sandy and pure shale, which require it to be evenly mixed in order to produce the highest grade of brick. This outfit did not allow proper mixing, and on account of the lumps, had a capacity of less than 300 tons per day.

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After this experiment, we reluctantly went back to our old system of shooting the face as widely as necessary, and then gathering the material with shovels and by hand into cars.

*Tunnel Method Now Used*

The present method of quarrying and handling the shale is as follows: The whole face is worked in benches of about 30 ft. Two-inch holes are bored by hand, with a Spry boring machine, about 15 ft. into the shale, and are loaded with 40 per cent. "Giant" powder and shot seven to ten at a time with a battery. The shale falls on the tunnels shown in Fig. 1 just above the factory roof.

These tunnels—at present about 20 in number—approximate the timbering in a mine crosscut. On 8 by 14 in. mud sills are erected 12 by 14 in. legs, 6 ft. 10 in. apart at the collars and 8 ft. 10 in. apart on mud sills—which are connected by 14 by 16 in. collars, 7 ft. above the mud sills, the whole comprising a set. These sets are spaced 5 ft. apart, and are connected by 3 by 12 in. planks on the sides, which act as lagging, and by 6 by 12 in. bars set 8 in. apart on top of the collars which form the grizzly shown in Fig. 3.

Between each set, and underneath this grizzly are constructed two hoppers which together hold a car of shale. These hoppers are closed at the bottom with "needles" similar to those in the bottom of the old-fashioned dump wagons which may be moved easily when the cars are to be filled.

In constructing a tunnel, five sets are set in the solid, and perhaps seven or eight are set up outside. As the shale face of the quarry advances the tunnel is driven to keep five sets ahead of it, the timbers being removed from the outside, and used for driving ahead.

In preparing to shoot on a tunnel, 4 by 12 in. planks are placed crosswise on the grizzly, and these are covered with lumps of shale to a depth of 2 or 3 ft. From 2,000 to 3,000 tons of shale are then shot down on a group of three tunnels, which converts each tunnel, as it were, into a bunker containing from 750 to 1,000 tons of shale. The planks are then removed, a few at a time, from the outermost end of the tunnel and the shale fed through the grizzlies into the hoppers. All lumps are bull-dozed, from one-half to three sticks of dynamite being used, or they are broken up with wedges and hammers; 100-ton lumps are a common occurrence, and the fact that these often fall 125 ft. speaks well for the strength of the construction. The shale between the tunnels, which are placed on 20-ft. centers, is left in place until it is time to remove the outer timbers, when it is loaded into cars by hand. This protects the sides of the tunnels and prevents the timbers from buckling—and together with the 2-ft. cushion of shale placed on top of the tunnels before shooting, closely approximates the conditions prevailing in a crosscut in a mine where the "arch"

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is formed and comparatively weak timbers withstand the weight of millions of tons.

### *Loading and Conveying*

The shale having been shot down covers the tunnels from the quarry face nearly to the outer end of the tunnel. The method of working it down to the grizzly is shown clearly in Fig. 3. The lumps are started

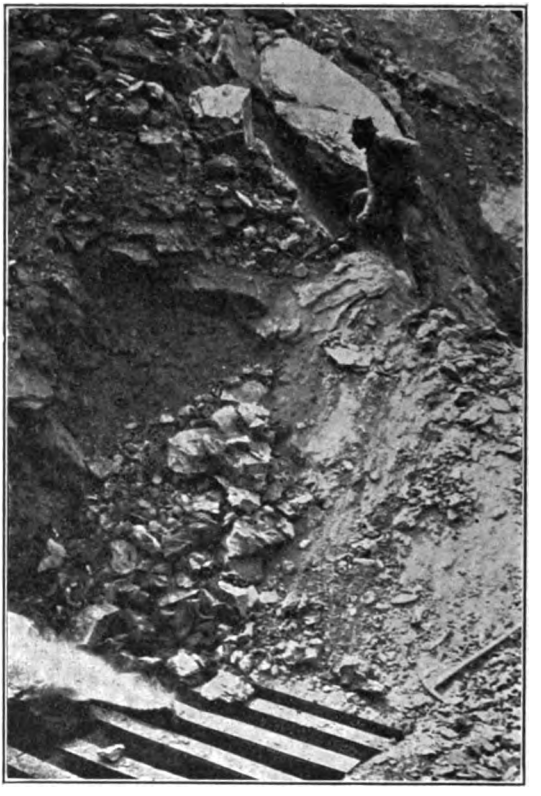

FIG. 3.—WORKING SHALE DOWN TO GRIZZLY OVER TUNNEL.

with a bar and roll down, some falling through and others having to be broken by hand. When the hoppers are full, the cars, which are a simple box 7 by 2½ ft. on the bottom, and flaring to 7 by 3½ ft. on the top, which is 5 ft. above the top of the rail, are filled and run down to the dumping plant on a 3-ft. gauge track. The main lines of the track system are built of 40-lb. rails and all tracks are so graded that the power required to move a full car to the dumping plant is approximately the same as to move an empty back into the tunnel. The trackage and general layout of the tunnels is shown in Fig. 4.

From each direction the main tracks lead into the dumping plant,

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which is equipped with two Link-Belt rotary dumps. These are so arranged that when a trigger is released the whole car rolls over sideways and dumps its contents into a hopper. As soon as the car is empty the center of gravity is thus shifted and the car rolls back into place, too great celerity of action being prevented by a band brake operated by a lever and rope.

The dumping plant is equipped with two hoppers, one for coarse shale equipped with a grizzly and crusher, and the other for fine material.

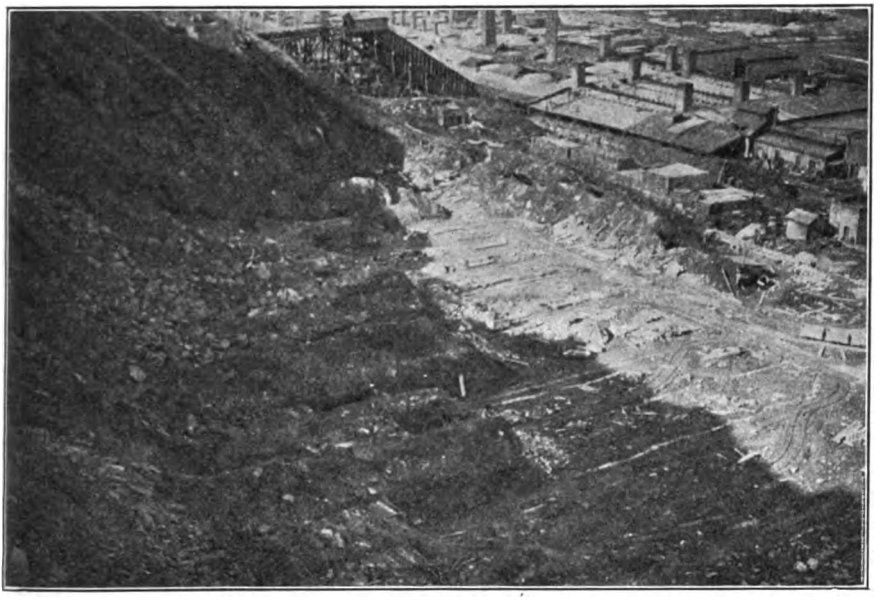

FIG. 4.—GENERAL LAYOUT OF TRACKS AND TUNNELS.

These hoppers empty the material on a 30-in., 6-ply rubber belt about 50 ft. between pulley centers, which in turn dumps the shale on a similar belt which rises at an angle of 22°, and running at about 300 ft. per minute, delivers the shale to the third floor of the factory.

On reaching the factory, the shale is dumped into a Jeffreys chain conveyor, the flights of which, 24 by 10 in., scrape it along the bottom of a steel-lined box to the openings into the bunkers. There are six of these bunkers, each one of which holds shale sufficient for about 40,000 brick. Each bunker feeds a dry pan, or grinding mill, the duty of which is to grind shale sufficient for 30,000 brick in 9 hr.

Each one of the six openings is equipped with a slide, usually left about one-sixth open, which causes all the shale carried along by the chain conveyor to be equally distributed among the six bunkers so that a very even mixture is obtained for each mill. When the bunkers are

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full, or when it is found desirable to fill the storage, the shale still escapes through one or more of the six openings, but is thrown by means of a hinged door into rope and disk conveyors, 150 ft. long, which run at right angles to the chain conveyor, and carry the shale into the storage shed, the capacity of which is 12,000 tons. This shale is stored in the summer and is mixed with the wet shale in winter in order to make a mixture dry enough to work in the mills efficiently. The stored shale is conveyed to the dry pans by means of underground belt conveyors, whose pits are covered with plank which may be taken up as the shale is fed to the conveyors.

### *Efficiency of Tunnel System*

Under the hand-loading system, 2-ton capacity, side-dump cars were used, which a crew of three men operated, loading the shale and pushing the cars on an average 100 ft. to the dumping plant, and dumping their own car. These men worked on contract and averaged 10 cars per man per 10 hr. The output was therefore 2 tons per man per hour.

When the present system was installed the pit was enlarged so that by the time the system was in working order the average haul was 200 ft. The men still worked in crews of three, but each man averaged 1.9, 2.3-ton cars per hour in summer (1.7 in winter), or 4.37 tons per hour per man—more than twice the former output. When the haul reached an average of 300 ft. a mule and driver took over the hauling which increased the tonnage per man per hour to 4.6 tons—enough to pay the cost of the mule and the man. Five men, with the new dumping, conveying and distributing plants handle 700 tons per day as against four men before who handled about 300 tons, so that this cost is just about cut in half by the installation of a modern plant.

The advantages of the new system over the former system may be summed up briefly as follows:

1. The cost of loading, tramming, crushing, conveying and distributing the shale is reduced more than half.

2. A more evenly mixed material is obtained by the system of distribution described and because the shale can be shot down in larger quantities, allowing more choice.

3. The drainage furnished by the tunnels allows the quarry to be worked at times—during the rainy season—when it was formerly impossible, and furnishes a dryer, and hence more easily ground material. This makes it unnecessary to draw from storage so frequently and so reduces this expense.

Altogether, the system works most successfully. The details have required considerable study and minor changes have been made from time to time, but we believe that, on the whole, this method of quarrying and handling the shale meets the difficulties encountered in working so large, and so unusual an undertaking, in a manner which is satisfactory.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Capillary Concentration of Gas and Oil

BY C. W. WASHBURNE, WASHINGTON, D. C.

(Pittsburg Meeting, October, 1914)

FORMER studies of sedimentary strata have been based upon the mineralogical and mechanical characters of the solid components, rather than upon the open spaces between them. For present purposes let us take another point of view, and consider not the rock material, but rather its cavities, the pores and the fissures or cleaved joints which connect with them. Let us think of the geologic column not as a series of rocks, but as a mass of irregular connecting capillaries. Different parts of the column may be distinguished by the absolute size and by the relative amount of its open spaces.

## *Porosity*

Each stratum of the column is characterized by a certain porosity, which would be independent of the size of the rock grains if all were of the same shape. Slichter has shown that the percentage of open space in a pile of equal spheres is practically constant regardless of the diameter of the spheres. This is approximately true of clean, assorted sand or gravel, but it is far from being the case with most natural sediments which have a widely variable porosity, the magnitude of which is determined mainly by the amount of fine and of secondary material between the larger grains. The shape of the grains materially affects porosity, round grains leaving more open space than angular or flat grains. The thin plates of kaolin and the fine plastic matter of clay fit quite perfectly when pressed together, and it is probably this close fit, determined by shape of grain, rather than the small size of a grain, that causes the low porosity of ordinary clay shale.

The best-assorted, least-cemented sediments are the most porous. The porosity of even a very fine-grained rock may exceed 10 per cent. if the grains are fairly uniform and discrete as in the so-called "oil shale" discovered by Woodruff in the Green River formation of western Colorado

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and Utah. The rock has the megascopic appearance of a tough cemented shale, but it is really a consolidated calcareous mud or fine marl, composed of discrete, angular microscopic grains of calcite, with solid and liquid hydrocarbons in the interstices. The pores of this rock probably are not large enough to be charged with oil by the method of capillary concentration described below, which moreover operates only in wet rocks, but they are much more efficient channels for migration than the pores of clay shale. If the rocks were dry, as they now are, at least near the surface, the fine pores of the limey layers would rob oil from fissures by capillary action, and thereby create the so-called oil shale of western Colorado. It is possible, also, that the hydrocarbons represent organic matter that was buried with the calcareous mud, and are therefore analogous in origin to the Scotch oil shale. The latter, however, contains no liquid oil until it has been heated in the retorts, and it occurs in a region where there are no oil seeps. The rarity of oil shales and their general absence from oil fields suggest an origin of their hydrocarbon content different from that of oil, or else that there is some process operating in oil regions which has destroyed pre-existing oil shales and which has concentrated the oil in the sands and fissures. The same reasoning shows that the porosity of a stratum can have no relation to the concentration of oil except in a passive way, being only a measure of the maximum reservoir capacity of the stratum.

### *Capillarity*

Each stratum of the geologic column is characterized not only by a certain porosity but also by a certain average pore diameter, which may differ greatly from the average pore diameters of adjacent strata. The size of the capillaries varies approximately as the grain of the rock, being greatest in conglomerates and very coarse sands, and least in clay shales. Liquids are drawn into these pores by the force of capillary action, which varies directly as the surface tension of the liquid and inversely as the diameter of the pore. Crude oils probably have only about one-third the surface tension of water, and hence only about one-third of the capillary power.

Lord Raleigh[1] found that a water-air surface has a tension of 75.6 dynes per centimeter at 0° C. and of 72.8 dynes at 20° C. Somewhat lower values are given in recent tables, 73.21 dynes at 0° and 70.60 dynes at 20°.[2] A salty water, such as that commonly found in oil fields, having

[1] *Philosophical Magazine*, 1890; similar results by Hall, *Philosophical Magazine*, 1893.

[2] Landolph-Börnstein-Meyerhoffer tables, quoted by Müller-Pouillet, *Lehrbuch der Physik*, 10th ed., vol. iii, pp. 277–308 (1907). These are the same as the values found by Morgan and Stevenson, *Journal of the American Chemical Society*, vol. xxx, pp. 360–375 (March, 1908).

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a density of 1.14 at 20° C., would have a surface tension of about 79 dynes per centimeter.[3]

No determinations of the surface tensions of crude oil have been found in the literature examined. The writer obtained a value of 24.1 dynes per centimeter at 20° C. by the capillary method, using Pennsylvania crude oil (specific gravity, 0.852). Müller and Pouillet[4] quote a value of 26.4 dynes at 25° C. for a (presumably refined) petroleum having a specific gravity of 0.847. Magie found a tension of 25.9 dynes at 20° C. on petroleum (refined?). Quincke[5] using a refined oil of 0.7977 specific gravity obtained a tension of 29.7 dynes on the oil-air surface and 28.9 dynes on the water-oil surface, both determined at 20° C.

It is highly probable that the molecular attractions produce a concentration of the fractions of lowest surface tension in the molecular "film" at an oil-air surface, and hence that the surface tension of crude oils is much less than would be expected from their specific gravities. For the same reason only pure material can be used in determining the surface tension of individual hydrocarbons even approximately. Some of these have been calculated by the writer from the values of the capillary constant $\alpha$ determined by Bartoli and Stracciati[6] and are given below. Evidently the old statement repeated in many text books,[7] that the addition of $CH_2$ does not change the surface tension, is not true for hydrocarbons, although it appears to hold for alcohols.

*Capillary Constants of the Paraffin Series*

| Substance (Normal) | Temperature, Centigrade | $\alpha$ (a) | Surface Tension (b) | Substance (Normal) | Temperature, Centigrade | $\alpha$ (a) | Surface Tension (b) |
|---|---|---|---|---|---|---|---|
| $C_6H_{12}$ | 11.0 | 1.635 | 16.0 | $C_{11}H_{24}$ | 14.0 | 2.688 | 26.4 |
| $C_8H_{14}$ | 11.0 | 2.038 | 20.0 | $C_{12}H_{26}$ | 12.8 | 2.781 | 27.2 |
| $C_7H_{16}$ | 12.0 | 2.399 | 23.5 | $C_{13}H_{28}$ | 14.0 | 2.848 | 27.9 |
| $C_7H_{16}$ | 12.0 | 2.387 | 23.4 | $C_{14}H_{30}$ | 13.0 | 2.915 | 28.7 |
| $C_8H_{18}$ | 11.0 | 2.477 | 24.3 | $C_{15}H_{32}$ | 13.3 | 2.997 | 29.4 |
| $C_9H_{20}$ | 14.0 | 2.538 | 24.9 | $C_{16}H_{34}$ | 14.0 | 3.041 | 29.8 |
| $C_{10}H_{22}$ | 13.0 | 2.629 | 25.8 | | | | |

(a) Milligrams per millimeter.
(b) Dynes per centimeter, calculated from (a).

[3] Approximate calculation of the density of solutions may be made by the use of the *Smithsonian Physical Tables*, 4th ed., p. 91, and with this the surface tension may be estimated from Volkmann's table, *Wiedemann Annalen*, vol. xvii, p. 353: *Smithsonian Physical Tables*, No. 142, p. 128.

[4] *Loc. cit.*

[5] *Poggendorff Annalen*, vol. cxxxix, and *Philosophical Magazine*, 1871, as corrected by Worthington, *Philosophical Magazine*, vol. xx (1885). See Gray, *Smithsonian Physical Tables*, 4th ed., p. 129 (1908). Worthington regards Quincke's values as too high.

[6] Engler, *Das Erdöl*, vol. i, table 22, p. 91.

[7] Wilhelmy, *Poggendorff Annalen*, vol. cxxi, p. 44 (1864). Jones, *Elements of Physical Chemistry*, p. 139 (1902).

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At the end of this list there is an unfilled gap up to melted paraffin which has a surface tension of 30.56 dynes at 54° C.[8] The surface tension of solid hydrocarbons, paraffin, asphalt, etc., must be very much higher, but small bodies of these, colloidal or other, would not affect the surface tension of crude oil. The unsaturated hydrocarbons appear to have a higher surface tension than the corresponding members of the methane series,[9] but if there is any of the latter in a crude oil it probably will determine the surface tension of the mixture. In general it follows from the Gibbs-Thompson law, that the hydrocarbons of lowest surface tension will tend to be concentrated at an oil-air surface, sufficiently to lower greatly its tension. Hence it is probable that all crude oils have low surface tension, probably in the neighborhood of 25 dynes, except only the oils that have lost all of their light constituents, which are of no consequence in problems of deep migration. It is unfortunate that these deductions must be based on theory. The need of accurate experimental work is very great.

### *Capillary Concentration*

Since water has about three times the surface tension of crude oil, capillary action must exert about three times as much pull upon it. The amount of the capillary pull varies inversely as the diameter of a pore. Hence the constant tendency of capillarity is to draw water, rather than oil, into the finest openings, displacing any gas or oil in the latter. Gas itself is not drawn into capillaries by the action of surface tension, and it can leave the fine pores without any capillary resistance. It is therefore the most quickly and the most completely gathered in the largest spaces available. Moreover, capillarity resists any movement of water from fine toward large pores more than it resists the movement of oil or gas in that direction. In short, water enters fine capillaries about three times as readily as oil, and it encounters about three times as much capillary resistance in leaving them.

The final result of this action must be the concentration of nearly all the gas and oil in the openings having least capillary power, namely, in fissures if present, and in the larger rock pores. This appears to be a general rule in oil fields, where the pores of the coarser-grained rocks contain practically all of the oil, while the pores of the adjacent fine-grained rocks contain practically no oil. Any slow flow between shale and sand would bring about the concentration of oil and gas in the sand,

[8] Landolph-Börnstein-Meyerhoffer tables, Müller-Pouillet, *Lehrbuch der Physik*, vol. iii, p. 277 (10th ed.).

[9] Hexane has a surface tension of 20 dynes at 11° C. (Schiff); benzene, 28.3 at 11° C. (average of various authorities, except Morgan and Stephenson, who found a tension of 29.38 dynes at 22.5° C. or 30.5 at 11° C.); benzole has a surface tension of 29.86 dynes at 12.5° C. (L.-B.-M. tables) or 30 dynes at 11° C. .

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and in the absence of such independent movements it is possible that capillarity alone would drag enough water into the shales to displace most of the gas and oil. However there is no need to place such a heavy burden upon capillarity, since independent flows would almost certainly be created by the differences of pressure or head arising during the deformation and subsequent erosion of the strata, besides the general upward slow flow of the rock fluids that is indicated by the fact that the observed downward increase in pressure is greater than the corresponding hydrostatic head. Any of these movements of the rock fluid would concentrate gas and oil in the sands, regardless as to whether the movements are continuous across the sands or oscillating, because capillarity pulls water out of sand into shale with greater force than it exerts on gas and oil, and because capillarity offers greater resistance to the passage of water from shale into sand than it offers to gas and oil. The recognition of this principle is the first step in the clarification of the local lithologic distribution of gas and oil.

In dry rocks there could be no capillary concentration of oil. In the absence of water, oil would be drawn into the finest pores and would be widely diffused. It is probable, however, that the pores of all clay shales contain abundant free water, in addition to the adsorbed moisture, except in dry regions near the ground surface. A rock like the so-called oil-shale of western Colorado and Utah might conceivably be formed by the capillary impregnation of layers of fine porous limestone by oil which ascended through fissures into the dry rocks near the surface. Since the pores of the limey layers are as fine as those in shale the bituminous layers could not be formed by capillary concentration in wet rocks. On the other hand productive deposits of oil and of gas, which are limited strictly to coarse-grained rocks and fissures, cannot be formed in the complete absence of water, nor if formed could they long exist in dry rocks, because they would be destroyed by capillary diffusion into the finer pores.

There is both an upper and a lower limit to the diameters of pores in which capillary action can take place. The capillary action of pure water is nil in tubes over 0.508 mm. diameter, and in smooth fissures over 0.254 mm. wide. Openings larger than this exert no attraction on water, and when present they are the most favored loci of the capillary concentration of oil and gas, because water is the most effectively removed from them. Capillary action of an average crude oil is nil in tubes over 0.2 mm. diameter and in fissures over 0.1 mm. wide, yet openings of this class are favored in the concentration of oil, because water is the controlling medium, and it is drawn into the finer openings with about three times the capillary force of oil, which tends to be left behind in the larger openings. The water tends to displace the gas and oil in the smaller openings, shoving them into the bigger spaces.

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The minimum diameter of capillary openings is somewhat uncertain. It is usually placed at 0.0002 mm. for tubes and at 0.0001 mm. for plane fissures. Owing to the great friction in such small openings, the rate of flow is almost negligible,[10] and there is little need to consider their capillary action. According to theory, capillary action must cease when a pore becomes so small that there is not room for two or more molecules of water to pass through it together, that is, in the case of a fissure when its width becomes less than twice the radius of the sphere of molecular action. Quincke[11] finds that the range of molecular attraction of water is about 0.00005 mm. This indicates that the minimum diameter of capillary openings is 0.0001 mm. for fissures, and probably twice as much or 0.0002 mm. for tubes.

### *Capillary Pressure*

The maximum theoretical pressures developed by capillary action are those developed in the finest tubes in which capillary action is possible. Johnston and Adams[12] have calculated these capillary pressures for the temperatures prevailing at various depths, but they probably err in extending their calculations to tubes as fine as 0.00001 mm. (0.01 micron), which gives 20 times the theoretical maximum pressures. The work of Quincke and of Lord Raleigh shows that there can be no capillary action in such small tubes. Accepting the assumptions of Johnston and Adams, and the minimum diameter of 0.0002 mm. for tubes, the maximum capillary pressures developed by pure water will have the values shown in the following table. The capillary pressures of a salt water such as that common in oil fields, would exceed these values by about 5 per cent.

*Approximate Maximum Capillary Pressures*

Diameter of Pores, 0.0002 mm. (0.2 $\mu$).

Temperature Gradient, 1° C. per 30 m.

| Depth, Meters | Pressure, Atmospheres | Depth, Meters | Pressure, Atmospheres |
|---|---|---|---|
| 100 | 15.0 | 2,000 | 12.1 |
| 200 | 14.6 | 5,000 | 7.8 |
| 500 | 14.1 | 10,000 | 1.0 |
| 1,000 | 13.6 | 20,000 | 0.0 |

Assuming a temperature gradient of 1° C. per 30 m., as in the preceding table, it is evident that capillary action loses about half of its force at a depth of 5,000 m. Since the heat increases more rapidly down-

[10] See calculations by Johnston and Adams, *Journal of Geology*, vol. xxii, pp. 1–15 (1914).

[11] *Poggendorff Annalen*, cxxxvii, p. 402.

[12] *Loc. cit.*

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ward in many oil fields, it is probable that the capillary force decreases half at depths of 3,000 or 4,000 meters, below which it becomes practically ineffective. Moreover the surface tensions of all but the lightest hydrocarbons decrease much less rapidly than that of water for each increment of temperature, so that the surface tension of water does not have such great excess over that of oil at these depths. Hence it is probable that the capillary concentration of oil and gas must all be effected within 4,000 or 5,000 meters of the ground surface. Oil in deeper strata must remain diffused in the shales, if that were its original distribution, unless it was concentrated in the sands at some former period when the strata concerned were closer to the surface.

It will be observed from the preceding table that even the maximum capillary pressures are wholly inadequate to account for the high pressures in oil and gas fields. Nor is it probable that capillarity contributes anything to the total pressure on a body of oil or gas, since the capillary pressure operates in the opposite direction, *i.e.*, toward the finer pores and is really a pull from the larger pores, or a negative pressure. It would decrease the pressure of the collected oil in the sand were it not for the return of fresh oil and gas displaced from the shale, which nearly counterbalances the partial vacuum that capillarity tends to make. Therefore it is necessary to abandon previous assumptions that capillary action may contribute to the high pressures in oil fields, the mysterious source of which must be sought elsewhere.

Possibly the high pressure is an effect of the upward migration of the rock fluids (mostly water, nitrogen and methane) which are subjected to outward pressures of deep-seated origin. If we accept the plausible assumption that the rock pores and fine fissures are continuous, and that they extend downward to the zone of rock flowage, it is apparent that this deep connection would cause the abyssal gases and liquids to press upon those above, raising their pressure above the normal hydrostatic pressure, and tending to crowd them toward the surface. An imperceptible outward translation of the rock fluids *en masse* would be sufficient to explain the excess pressures in oil fields. The capillary penetration of the abyssal fluids need not approach the base of the sedimentary strata to account for the increased pressure in sands far above them. However there are some phenomena such as the excess temperature in oil sands,[13] the excess of chlorine in oil-field waters,[14] and the great amount of helium and argon in natural gas,[15] which suggest that the upward migration in oil regions may be more extensive than the excess pressure

[13] Höfer, Temperature in Oil Regions, *Economic Geology*, vol. vii, No. 6, pp. 536–541 (September, 1912).

[14] Washburne, Chlorides in Oil-Field Waters, *Bulletin* No. 87, pp. 375–381 (March, 1914).

[15] Cady and McFarland, *University Geological Survey of Kansas*, vol. ix (1908).

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would seem to require, and that possibly some fluids of abyssal origin may be penetrating far through the sedimentary strata.[16]

### *Time*

In tubes approaching the minimum capillary diameter the rate of flow from capillary action would be negligible. But in pores of 0.001 mm. diameter, which is of the same order of magnitude as the intergranular pores in shale, the capillary movement of water would be about $15 \times 10^{-6}$ c.c. per year.[17] In a rock containing 10 per cent. of pore space, this would result in a flow of about 15 c.c. across each square centimeter per year. Probably half of this amount, or about 7 c.c. per square centimeter per year, corresponds more closely to the factors of flow imposed by the porosity of ordinary clay shale. This amount of flow seems quite adequate to produce capillary concentration in a geological period, but not in short time. In 10,000 years the movement, if continuous, would amount to 700 m. However the movement would not be continuous, but would vary in direction, with the changes in pore diameter, etc., and only small distances could be traversed directly by capillary pull. The actual amount of capillary movement required rarely would exceed a few meters, since most shales are cut by fine fissures, in which the oil would first be collected by capillary exchange for water. In big shale formations the fine capillary fissures probably gather the oil from its original disseminated distribution in the shale, and pass it on to the coarser joints and pores of the sandstones. The fine fissures of shale offer much less resistance to the flow of oil than the rock itself, through which the flow need be only that required to reach a fissure. Under the principles here advanced, the capillary flow of oil in wet rocks should tend always toward the larger openings.

Moreover, it is not necessary to assume that all of the movement in the pores is due to capillarity. Doubtless there are other slow motions of the underground fluids arising from various causes. Any motions of the rock fluids, whether continuous or intermittent, direct or reversed, or oscillating, would assist in the concentration of oil and gas in the larger openings, because water always would enter the finer capillaries with greater ease than oil or gas and would leave them with greater difficulty. The more to-and-fro movement of these fluids between shale and sandstone, or between shale and fissures, the more perfect will be the capillary concentration of oil and gas in the sands and fissures.

A factor which tends to increase the perfection of this concentration

[16] Moreau and Lepage argue the abyssal origin of the nitrogen and associated rare gases in coal mines and in thermal springs. *Academy of Science*, Paris (Mar. 23, 1914); abstract in *Revue Scientifique*, 52e Année, p. 442 (1914).

[17] Johnston and Adams, *loc. cit.*, p. 14.

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is found in the behavior of water vapor, which is evaporated from surfaces of large curvature at the same time that it is condensed upon surfaces of sharp curvature. In this way water is passed from the coarse pores of sandstone, and from fissures, to the fine pores of shale. The vapor transfer will begin as soon as capillary concentration has partly filled the sand with gas, and thereafter it will supplement that process. This may explain the dryness of natural gas.

### *Extended Application. Dry Sands*

Capillary concentration must operate on any of the gases present in the rock pores. These consist of nitrogen (which commonly is distinguished from atmospheric nitrogen by its relatively high content of argon and helium), of methane and of carbon dioxide. Oxygen probably does not occur in the deep gases except in rare traces, and samples containing it probably are contaminated with air. The capillary concentration of nitrogen in sands is exemplified by the great nitrogen well of North Dakota, and by the gas of the western fields of Kansas, which in places contains over 80 per cent. of nitrogen that is notably rich in argon and helium. Gases rich in carbon dioxide are rare, outside of California, and the disappearance of the great quantities of this gas which would be formed by the decomposition of organic matter, is an unexplained difficulty of the organic theories. All of these principal rock gases are concentrated in the coarser pores by capillary action.

Capillary concentration does not of itself produce pressure. Many of the deep sands of the Appalachian and other regions are apparently dry, yet they do not contain enough gas under pressure to cause much escape when they are penetrated by the drill. Doubtless the sands were once full of the water in which they were deposited, but now they absorb some of the water used in drilling. Their dryness may be explained by the process of capillary concentration, as well as in other ways.

The process should operate effectively in all sands in which there is no very active supply of water, but not in sands like the Dakota of most western localities, where there is an artesian circulation that would overcome the capillary withdrawal of water. In sands of this type the process would be effective only in places which are protected from rapid water circulation by structural peculiarities or other causes.

### *Weakening of Water "Films"*

There are two main elements of possible weakness in the preceding hypothesis: First, the weakening of the capillary force of water by contamination with oil, and second, the high resistance of bubbles to capillary flow. A thin film of oil or other greasy substance, amounting to

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only an invisible trace, greatly lowers the tension of a water-gas surface, reducing it almost to that of an oil-gas surface. Moreover, there is considerable tension on the water-oil surface,[18] which in the case of light oils, is nearly equal to the tension on an oil-gas surface. In a capillary tube the water-oil surface is concave toward the oil. In other words, a water-oil surface exhibits the same capillary phenomena as a water-air surface, the difference being only one of degree. The capillary force of a water-oil surface is decreased very little by the slight angle of contact of the surface with the pore walls, and it amounts approximately to one-third that of a water-air surface. Hence the capillary behavior of water and oil is nearly the same as that of water and air, and there is a constant tendency for water to be pulled into fine capillaries previously occupied by oil. This tendency is demonstrated experimentally by the familiar damage to a lamp flame produced by a little water at the bottom of the lamp's reservoir. If the wick touches the water, the latter rises through the capillaries previously filled with oil, makes the flame splutter, and often extinguishes the light. In the same way water will pass from the coarse spaces of sand or from fissures into the fine capillaries of shale, displacing the oil, which is thereby forced into the sand through neighboring pores. Thus, the weakening of the oil-gas surfaces by contamination with oil, is counterbalanced in part by the capillary action of the water-oil surfaces.

### *Effect of Capillary Gas Bubbles*

The second element of possible weakness in the hypothesis lies in the high resistance to liquid flow offered by bubbles of gas and by drops of an immiscible liquid. If a fine capillary tube be filled with water containing bubbles of air, so that the bubbles are strung along the tube in alternation with drops of water, a pressure difference of over one atmosphere is required to produce flow in the tube. The various water-air films appear to cling rather tenaceously to the walls, and each seems to take up a part of the external pressure by a slight change of curvature. The action of the pressure is analogous to a vertical pull or push on a horizontal weighted cable supported at the ends. A certain amount of energy must be expended in lifting or depressing the central part of the cable, and in changing its form of curvature, before its terminal supports are affected. A series of drops of oil or of gas in water, is divided by twice as many catenoidal "films," stretched across the capillary, among which the applied pressure is distributed, so that the total resistance of a long series of such drops in a fine tube is very high.

[18] Quincke found a tension of 28.9 dynes per centimeter on a surfac separating water and petroleum. *Poggendorff Annalen*, vol. cxxxix, as corrected by Worthington, *Philosophical Magazine*, vol. xx (1885).

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Rock pores, however, are not single tubes, but the most irregular passages between the grains of rock, having countless lateral and longitudinal connections by means of which many of the little water bodies communicate with each other. They become single bodies as regards their tensional forces, which tend so to draw them together as to reduce the total area of their surfaces. The minute bodies of oil and of gas have a similar tendency when connected in any way, however devious. All the drops and bubbles of the three fluids must grow into larger, more compact, bodies. Thus a condition of many small alternating bubbles of gas and drops of liquid could not exist very long in the rock pores, and they could offer only temporary resistance to capillary flow.

Geologists sometimes appeal to hypothetical emulsions as a means for the migration of oil. This is wrong. For the reasons stated above minute bodies of oil can have only temporary existence when compared with the long time required for geological processes. Surface tension will soon destroy an emulsion of water and petroleum, and moreover the high viscosity of emulsions would prevent their migration through fine pores. There is no reason for believing in the existence of any natural oil emulsions before the final rush of the oil and water out of the rock pores into an open well.

### *The Anticlinal Theory*

The rôle of capillarity in producing the anticlinal distribution of oil requires consideration.

Whenever light oil touches a water-air surface, it is immediately spread over the surface and retained by surface tension. A heavy oil is retained with nearly equal strength, but after spreading a molecular film of its lightest elements over the surface of the water, it gathers itself into a drop, or remains as an irregular thick mass. If a bubble of gas rising through water touches a globule of oil the two immediately unite, and the oil forms a thin continuous shell about the gas bubble. Johnson has suggested that this process may explain the accumulation of oil at the top of water-soaked anticlines, the oil having been lifted bit by bit on bubbles of gas.

The theory is plausible and better than the old assumption that the difference in density between water and oil caused particles of oil to rise to the highest saturated parts of sands. It should be noted, however, that the volume of gas required is many times (probably 100 times) the volume of the oil which it could lift in this way; also that small masses such as bubbles or drops, do not have sufficient defect in weight to be forced upward through fine pores. Before gravity can move them it is necessary for the globules of oil and gas to be united into larger masses by surface tension as described above. Not until then can the lifting force on a body of gas or oil be sufficient to cause its ascent.



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It is probable that very large bodies of oil must be gathered before they displace sufficient water to receive a pressure competent to force them upward against the friction of irregular rock pores. A very simple demonstration of this fact may be made. Take a basin of water, and with a pipette or fountain-pen filler squeeze drops of oil against the bottom and sides of the vessel. It will be observed that most of the drops cling persistently to the vessel and do not rise through the water. The force holding them is not surface tension because the same surface energy prevails when the drops are free. Gravity does not dislodge the drops, which can be effected only by stirring the water or by tapping the vessel. The force holding the drops is merely the friction of contact, yet if lone of the same drops was spread out in the pores of an ordinary oi sand, the area of solid contact and therefore the external friction would be from 50 to 200 times greater. The gravitative lifting force on each drop would be the same as in the open vessel, and conclusively it could not move the drop of oil against the greater friction in the pores of a sandstone. This confirms the contention of Munn that gravity is not competent to cause the accumulation of oil above water in anticlines, etc.

The following working hypothesis is suggested to explain the common anticlinal distribution of oil.[19] First, the oil and gas must be gathered in the sands by capillary concentration. Second, the oil and gas must be segregated by the effects of surface tension into bodies large enough to be lifted by gravity, *i.e.*, into bodies so large that the difference between their weight and that of the displaced water is sufficient to drive them upward through the pores of the sand. Whether the volumes of such bodies would be a few liters or many cubic meters is a point requiring further study, but they are certainly of much higher order of magnitude than the drops of oil of the preceding experiment.

It is improbable that surface tension can gather oil into bodies large enough for the preceding requirement. Therefore a third process or stage is now postulated. Gas rises through water-saturated sand probably 300 to 500 times as readily as water.[20] This may account for the fact that the tendency of gas fields toward anticlinal distribution is much stronger than that of oil fields. It is therefore much easier to assume that gravity can collect gas at the top of anticlines than it is to assume the less probable condition that it can lift oil through the rock pores to a similar position.

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[19] Other working hypotheses are under consideration.

[20] This approximation is based on the fact that the lifting force of water displacement applied on a given volume of gas would be about 7 times the lifting force on the same volume of oil, combined with the moderate assumption that oil would suffer from 40 to 70 times as much frictional resistance as gas. Data for exact calculation are not at hand.

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After a small reservoir of gas has been started at the crest of an anticline, then whenever circulatory or migratory movements of the water body bring a particle or mass of gas or oil in touch with the water-gas surface, the gas or oil can never descend. They will be held on the water-gas surface by its tension. The oil will first make a film along the top of the water, and any oil which is brought in contact with this film must join it. In this way a deposit of oil will be formed between the water and the gas, and neither the ascent of the oil nor its larger lateral collecting movement in the sand need be the result of its low specific gravity. Any movement of the water in the sand, which would in time bring the various parts of the water in contact with its upper surface, would carry bodies of oil of all sizes to the water-gas surface, where surface tension would retain them permanently.

As an example, imagine a slow lateral migration of the water in a gently folded sand. Such slow migration has probably occurred in many oil sands, because of the commonly evident differences of head. Gravity would cause the accumulation of gas in the sealed arches, and these accumulations would not be destroyed if the sand be thick and if the flow of water be very slow. On the contrary, the transported accessions of gas would increase the deposit. Because of surface tension, the transported masses of oil would cling to the water-gas surface whenever they touched it, and later contributions would cling to the water-oil surface. Thus an anticlinal deposit of oil would be formed without the intervention of gravity except in the initial collection of an anticlinal body of gas.

These principles hold generally under any theory of the origin of oil. The process of capillary concentration is especially required by the organic theory, under which it is necessary to postulate an originally disseminated distribution of the oil in shale or limestone. Supporters of the organic theory may claim that the present absence of oil in the pores of shale adjacent to the oil sands is due to its removal by capillary action. Under the inorganic hypothesis it is not necessary, as a rule, to assume that the oil ever has entered the fine pores of shale. Its entire migration may have been in fissures and sands, and it may never have been widely disseminated. All of these possibilities must be considered until someone shall find satisfactory evidence concerning the organic or inorganic origin of oil.

### *Conclusion*

The final condition of capillary stability under which the rock fluids come to rest is not an equilibrium of pressure, but rather a condition of minimum surface energy. Balanced pressures would be secured by the occupation of the finest openings by gas and oil and of the coarsest by water. Capillarity, however, does not operate in this way. It seeks only to reduce the surface energy to a minimum by the shortest route,

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which consists in the destruction of emulsions, the union of bubbles and of drops of like substances, and in the movement of the substance of highest surface tension (water) into the finest capillaries.

Capillary concentration gathers gas and oil in the coarsest spaces available: In fissures and super-capillary spaces when present; in conglomerates or coarse layers in sandstone; in sandstone layers in shale; in dolomitic and cavernous parts of limestone. It cannot operate in dry rocks.

Surface tension collects the small bodies of gas and oil in a sand into larger bodies, which in the case of gas are capable of gravitative displacement by water, and are lifted to the highest position they can reach. When circulatory or other movements of the water bring occluded masses of oil in contact with the water-gas surface, the oil is held firmly on the surface by its tension. In this way a deposit of oil is gathered at the top of the water-bearing part of the sand. In the absence of these conditions, oil could accumulate below water, as it does in some fields.

The capillary concentration of oil and gas in sands and fissures is an essential requirement of the organic theories, which postulate that oil was originally disseminated in shales and limestones. It is not essential to the inorganic theories, under which the phenomena of concentration are capable of independent explanation, but the principles stated are thought to hold true for all oils in porous rock, regardless of their origin.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Origin, Mining and Preparation of Phosphate Rock

BY E. H. SELLARDS,* TALLAHASSEE, FLA.

(Pittsburg Meeting, October, 1914)

PHOSPHATE rock like most other mineral substances is found in nature in varying degrees of purity. Of the impurities that are present some are constituents of the rock itself; others are inclusions of a foreign substance within the rock; while still others represent merely associated materials or minerals, either clinging to the rock or found in cavities and natural depressions, and hence largely removed in mining. Some of these impurities are distinctly deleterious to the processes of manufacture for which the phosphate is mined, while others, although neutral in action or nearly so, yet by their presence reduce the average grade of the rock and thus add useless bulk to the shipment.

It is the object of refined processes of mining to bring the product, as delivered from the mine, to the highest possible grade consistent with the market requirements and demands. This, however, is not accomplished without actual loss in the form of discarded phosphate. It is evident, therefore, that the devising of methods for reducing this loss in mining, and yet maintaining the grade of the rock which the market requires, is one of the serious problems that the producers have to face.

## MINERALS OF PHOSPHATE ROCK

The minerals included under the term "phosphate rock" are the calcium phosphates. Of these, apatite is perhaps the most definite and constant in composition, although of this mineral two varieties are recognized, namely, fluorapatite, $Ca_5(PO_4)_3F$, and chlorapatite, $Ca_5(PO_4)_3Cl$. Moreover, the calcium of this mineral may be partly replaced by manganese, forming yet another mineral, manganapatite; or the mineral may become hydrated forming hydroapatite, which is found as mammillary deposits often not unlike chalcedony in appearance. The term "phosphorite" has been applied to the massive amorphous deposits

* Non-member.

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of phosphate, which may be compact, earthy or concretionary.[1] Among other varieties of apatite may be mentioned staffelite, which contains a small percentage of both iron and aluminum. It is of interest to note also that this variety is believed to result from the action of carbonated waters on phosphorite, and hence is likely to occur incrusting ordinary phosphate rock acted upon by carbonated waters. Another variety, pseudoapatite, contains both sulphur and carbon dioxide. Of the many other calcium phosphate minerals some closely approximate apatite while others grade into compounds so variable and indefinite in composition as scarcely to be classed as minerals. The deposits of phosphate found in nature evidently contain a number of calcium phosphate minerals, the constituent impurities of which affect the market value of the rock. The aluminum phosphate, wavellite, should also be mentioned since it is mined to some extent as a source of phosphorous.

## Associated Minerals

Various other minerals in nature are found associated with the calcium phosphates. This association is sometimes due to actual relationship between the minerals. On the other hand, the association of minerals may be purely accidental, or incidental to the manner of formation of the deposits. With regard to the related minerals, it is apparent that where the calcium phosphates are abundant, other phosphates are likely also to occur. In fact it is scarcely to be expected that extensive calcium phosphate deposits will be found without the presence of at least a limited amount of other phosphate minerals. This is particularly true of iron and aluminum phosphates. These two bases are widely disseminated in nature, and moreover they combine readily with phosphoric acid to form phosphates. Of the iron phosphates, the mineral vivianite, although occurring in relatively small quantities, is widely distributed in nature, and may occur in limited quantities in phosphate deposits, usually as an incrustation, or as an alteration product of other minerals. The iron minerals frequently form in bogs, and it is an observed fact that the phosphate deposits in such localities not infrequently contain more iron than do the same deposits when found on the uplands. In such cases the iron is doubtless a comparatively recent infiltration, and may include phosphates of iron as well as oxides and other iron minerals. Of the aluminum phosphates a large number are known, one of which, wavellite, as already stated, is mined as a source of phosphorus. This mineral and others of

[1] Dr. Austin F. Rogers, who is investigating phosphate minerals, states that phosphorite or phosphate rock seems to be a mixture of two minerals, amorphous collophanite, largely a solid solution of calcium carbonate in calcium phosphate, and crystalline dahllite, a calcium carbonophosphate with the formula $3Ca_3(PO_4)_2.CaCO_3$, analogous to fluorapatite. The amorphous collophanite gradually changes to the crystalline dahllite. (Personal letter of May 23, 1914.)

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the aluminum phosphates are likely to occur in association with calcium phosphate.

Some of the large phosphate deposits have been formed by the replacement of an original rock by calcium phosphate. In this process parts of the original rock not infrequently remain unchanged or incompletely phosphatized. Since the phosphatizing processes proceed from the surface, the imperfectly phosphatized remnant is likely to lie within the rock, thus giving rise to included impurities that are difficult to eliminate. Moreover small amounts of clay and silica are usually found in the limestone, and as these substances do not readily phosphatize, if not leached out, they remain as impurities in the rock.

Aside from these related minerals, the materials associated with the phosphate rock are varied in character. They include clay, fragments of limestone, flints, gravel, silica in the form of sand, and other resistant materials, the character of which is determined by the manner of formation of the deposits. The associated materials of this nature make up the matrix in which the phosphate rock is imbedded. It is scarcely possible in mining to remove entirely all associated minerals, and the purity of the rock as delivered to the market is affected, without doubt, by the presence of more or less of these minerals, as well as by the constituent impurities of the rock.

### Objectionable Impurities

Of the impurities contained in or associated with phosphate rock, the most objectionable in the processes of manufacture of acid phosphate for fertilizers, for which purpose the phosphate rock is almost wholly used, are iron and aluminum. For this reason practically all phosphate mined is sold under a guarantee that the combined iron and aluminum expressed as oxides, do not exceed a given small percentage of the whole, from 2 to 4 per cent. being allowable. Iron when present in excess of about 2 per cent. brings about reactions which result in the formation of a gelatinous substance injurious to the mechanical condition of the mixture, occasioning also a loss of soluble phosphoric acid. A first step in the reaction with the iron is probably as follows: $2FePO_4 + 3H_2SO_4 = 2H_3PO_4 + Fe(SO_4)_3$. Of the sulphate of iron thus formed, a part, according to Fritsch,[2] reacts on acid phosphate of lime thus forming the objectionable gelatinous precipitate. Owing to the demand of calcium sulphate for water, hydrated iron phosphate, which is a product of these reactions, may subsequently become dehydrated and insoluble, thus causing the loss of available phosphoric acid.

Aluminum existing as a silicate in phosphate rock is likely to be injurious, since according to Fritsch, if not decomposed by the acid, it

[2] J. Fritsch: *The Manufacture of Chemical Manures*, p. 79 (1911).

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may cause a part of the phosphoric acid to retrograde. However, when existing in the rock in small amounts as a phosphate, the aluminum is supposed not to occasion a loss of phosphoric acid, both the hydrated and the non-hydrated phosphate being soluble in the precipitated condition in phosphoric acid.

Carbonates of calcium when existing in small quantities in phosphate rock, are beneficial rather than injurious. When the ground rock is treated with acid the carbonate is the first of the ingredients to be attacked, and the heat thus engendered promotes subsequent reactions among the other constituents. Moreover the carbon-dioxide gas given off from the carbonate, lightens the mixture and facilitates drying. Phosphate rock low in, or lacking carbonate develops little heat in mixing, and reacts slowly. In such cases this constituent must be added. It is true that the presence of the carbonate necessitates the use of an increased amount of acid, which in turn results in the formation of an increased amount of calcium sulphate or gypsum. The amount of carbonate that is desirable is sometimes given as 5 per cent., but the limits are not strict, and the manufacturers do not as a rule find it necessary to specify directly the amount of the carbonate that the rock must or must not contain. Indirectly, however, an excess of the carbonate is guarded against by other requirements as to the grade of the rock. If it is true as elsewhere stated in this paper that the principal mineral of the massive phosphate deposits is a calcium carbono-phosphate, this fact will afford an explanation of the presence of the desired amount of carbonate in all phosphate deposits of this class.

The fluorine found in phosphate rock, upon being attacked by the acid, forms hydrofluoric acid gas which passes into the atmosphere, it being estimated that as much as from 50 to 66⅔ per cent. of the fluorine present is eliminated in this way. Although a small amount of acid is used up in this reaction and a proportionate amount of calcium sulphate formed, yet it is seldom, if ever, necessary to specify against the fluorine content of the rock, the amount present being negligible.

Among the numerous other impurities that may be present in phosphate rock, silica and clay are perhaps the most common. Here also should be mentioned moisture, which when present not only adds bulk to the shipment but also interferes with the processes of manufacture. The excess of moisture must, therefore, be removed by drying, not more than 3 or 4 per cent. being allowable in the rock as shipped.

## The Origin of Phosphate Deposits

The origin of phosphate deposits is such that the presence of associated minerals as well as constituent impurities is almost invariable. The original source of phosphorus, the constituent for which phosphate rock

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is mined, is in igneous and crystalline rocks, where it exists in combination with other elements forming phosphate minerals. These minerals, as indeed is true of all minerals, are soluble in water, the degree of solubility, however, varying with the different minerals, and with the diverse conditions to which they are subjected. Indeed some very interesting and suggestive observations have been made on the relative solubility of phosphates under varying conditions. Thus it has been shown that the solubility of the phosphate minerals is increased by the presence of decaying organic matter in water. They have also been found to be appreciably soluble in carbonated waters. In this connection Reese[3] has made the very important observation that while the phosphate dissolves freely in waters containing decaying organic matter and in carbonated waters, yet when allowed to stand over calcium carbonate, the phosphate is redeposited. In summing up his observations Reese says: "This experiment shows that phosphates may be transported in hard waters, but on standing on calcareous beds would tend to be given up." In speaking of hard waters the author evidently has in mind, waters containing, among other things, carbon dioxide in solution, and his conclusion is that such waters will or may drop the calcium phosphate from solution when they stand over limestone. These observations, if true, have two important corollaries, one of which has been noted by Clarke,[4] namely, that in the presence of the carbonate, the phosphate would probably not be dissolved, while the carbonate could pass into solution, thus leaving the enriched residue of phosphate. The second corollary is that calcium phosphate taken into solution by the soil waters at and near the surface may be thrown out of solution in case the water stands for a time at a lower level in the earth on or over limestones. That this process may have been and probably was a factor in the formation of large phosphate deposits resting upon limestone will be shown in the subsequent pages of this paper.

The rain water in passing through the soil and surface materials receives organic acids from the decay of vegetable and animal matter. It also receives carbonic acid which is held in solution, the water thus becoming carbonated, and hence more efficient as a solvent. The original rocks and the soils derived from them contain particles of the phosphate minerals, which when acted upon by the ground waters pass slowly into solution. It is through the solution of the mineral, its removal and subsequent redeposition that workable phosphate deposits are formed. When it is remembered that the phosphorus in the igneous rocks amounted to merely a fractional part of 1 per cent. of the whole,[5] and was without

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[3] *American Journal of Science*, 3d Series, vol. xliii, p. 402 (1892).

[4] *Bulletin No.* 330, *U. S. Geological Survey*, p. 443 (1908).

[5] According to Clarke, *Bulletin* 330, *U. S. Geological Survey*, p. 32 (1908) the amount of phosphorus in the lithosphere is 0.11 per cent.

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doubt originally widely disseminated, the importance of the processes of concentration of the mineral by ground water and the extent to which they have operated becomes evident.

The removal of the mineral from the original rocks and its concentration in later rocks is by no means a simple process. There are as a rule many intermediate stages, the load being taken up, dropped again for a time, only to be once more started on its journey. Among the primary results of concentration and enrichment may be mentioned the phosphate deposits of the crystalline rocks, where the mineral is found in veins, being more or less perfectly crystallized as the mineral apatite. Of deposits of this class some, including those of Canada, are of economic importance, and would be more extensively worked were it not that other and more cheaply mined phosphate deposits are available.

Of the phosphate taken into solution by the ground water a part is taken up from the soils through the roots of plants, and thus becomes a constituent of the plant life of the earth. From the plants the phosphorus passes to plant-eating animals, and through them to carnivorous animals. Phosphorus thus becomes a constituent of the organic life of the earth. The bones of the vertebrate animals in particular contain an appreciable amount of calcium phosphate. It seems well established also that certain of the important phosphate deposits, as well as the guano deposits, are derived from excrement and remains of gregarious animals, particularly birds. It is also true that a part of the phosphate taken into solution by the ground waters is again thrown out owing to changed chemical conditions, and in this way important phosphate deposits are formed. In either case, however, the phosphate may be regarded as only temporarily delayed in its round of circulation. Ultimately phosphate is carried in solution in the ground waters through springs and rivers to the ocean. While the amount in solution at any one time is relatively small, yet, through the continued operation of this agency over long periods of time, a large amount has reached the ocean.

The phosphate carried into the ocean is again removed from solution through the agency of organic life, or, owing to changed chemical conditions, is precipitated. Of the animals that utilize phosphorus taken from the sea water in the construction of a shell covering or skeleton, the best known perhaps is the brachiopod, *lingula*, the shell of a recent species of which has been found to contain 85.79 per cent. of calcium phosphate. The tests of the crustacea, although less distinctly phosphatic than the shell of *lingula*, contain an appreciable amount of phosphate. Thus the shell of a recent lobster was found to contain 3.26 per cent. of calcium phosphate, while the lobster as a whole contained 0.76 per cent.[6] The aquatic plants also utilize some of the phosphorus in

[6] *Bulletin* 330, *U. S. Geological Survey*, p. 448 (1908).

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solution in the water and through them the phosphorus passes to the skeleton of vegetable-eating aquatic animals, and through them in turn to the carnivorous animals.

The phosphorus taken from solution by chemical action is evidently considerable, since nodules of chemical origin, high in phosphates, are found somewhat abundantly in the bed of the ocean. Some of these nodules, reported by Clarke,[7] contain 19.96 to 23.54 per cent. of phosphoric acid.

The amount of phosphate that finds its way into the sedimentary formations through organic and chemical agencies is thus undoubtedly considerable, resulting in the enrichment of certain deposits, which, if not workable, at least serve as an important source of phosphate from which by further concentration workable phosphate deposits are formed.

In this respect the deposition of calcium phosphate is analogous to that of the related mineral calcium carbonate, although of the carbonate much more extensive deposits accumulate than of the phosphate.

Silica ($SiO_2$) in its round of circulation in the earth presents some interesting analogies and yet strong contrasts to both calcium phosphate and calcium carbonate.

While the round of circulation of phosphate minerals is capable of demonstration as a normal process comparable to that of other common minerals, yet the actual processes of the accumulation of large workable deposits of phosphate rock are in many ways complicated.

### *Florida Phosphate Deposits*

The complexity of origin of the phosphate rock and the manner in which impurities are included in the formation is well illustrated by the Florida phosphate deposits. Of these there are two distinct types, known respectively as the hard-rock and the land-pebble phosphates. These differ materially in their location, origin and manner of occurrence. The hard-rock phosphates lie in a belt along the Gulf side of the peninsula, extending in a general north and south direction roughly paralleling the Gulf coast for a distance of about 100 miles. The land-pebble phosphate deposits are found farther south, lying chiefly in Polk and Hillsboro counties.

The hard-rock phosphate deposits rest upon a thick and very pure, light-colored, porous and cavernous limestone known as the Vicksburg formation, which is of Lower Oligocene age. At present, in that section of the State no formation other than the phosphate itself lies on top of this limestone. It has, however, been demonstrated by the combined observations of several geologists that certain formations of which only a residue remains formerly extended across the area that now holds the

[7] *Bulletin* 330, *U. S. Geological Survey*, p. 105 (1908).

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hard-rock phosphate deposits. The formations referred to are Chattahoochee limestone and Alum Bluff sands, both of which are of Upper Oligocene age. These formations are now found bordering the hard-rock phosphate area. The proof of their former extent has been given elsewhere and need not be repeated at this time.[8] The hard-rock phosphate deposits are made up largely of the residue of these formations which have disintegrated *in situ*, and accordingly consist of a mixture of materials of the most diverse character including sands, clays, limestone fragments, pebbles and water-worn flints, vertebrate, invertebrate and plant fossils; in fact a heterogeneous mixture of the relatively insoluble and resistant elements of the earlier formations.

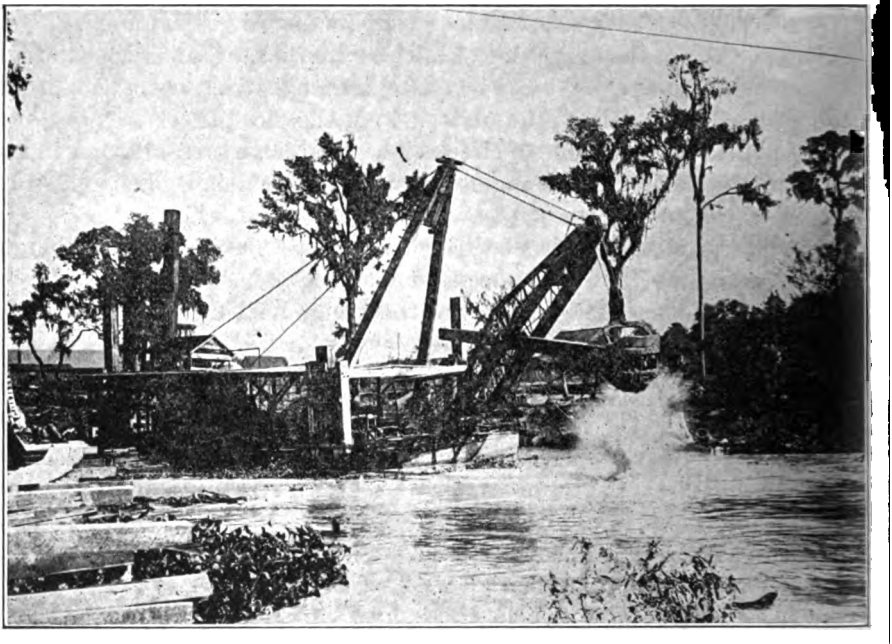

FIG. 1.—MINING PHOSPHATE ROCK BY DREDGE IN FLORIDA.

The phosphate itself is derived from the Alum Bluff sands, the later and thicker of the two formations that have disintegrated. This formation, the Alum Bluff, in some places reaches a thickness of several hundred feet, and has a large areal extent reaching from west Florida, through northern and central Florida, into southern Florida. Throughout its entire thickness, and throughout its whole areal extent this formation is distinctly phosphatic, although in no instance is the phosphate in this formation sufficiently concentrated to form workable deposits.

[8] Origin of the Hard Rock Phosphate Deposits of Florida, by E. H. Sellards, *Fifth Annual Report, Florida State Geological Survey*, pp. 23–80 (1913).

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While these formations, the Chattahoochee and the Alum Bluff, were disintegrating in the area that is now the hard-rock phosphate region, the calcium phosphate from the Alum Bluff formation was gradually being taken into solution by ground water and was being redeposited at a lower level in the earth, thus forming the workable hard-rock phosphate deposits. In this process the replacement of the original limestone by calcium phosphate was an important factor, and these deposits afford excellent illustration of the formation of phosphate rock by the replacement process, the shells of the original limestone in many instances retaining their form, although changed chemically to calcium phosphate. In addition to replacement, other processes are observed, prominent among which is the formation of the phosphate by precipitation from solution in

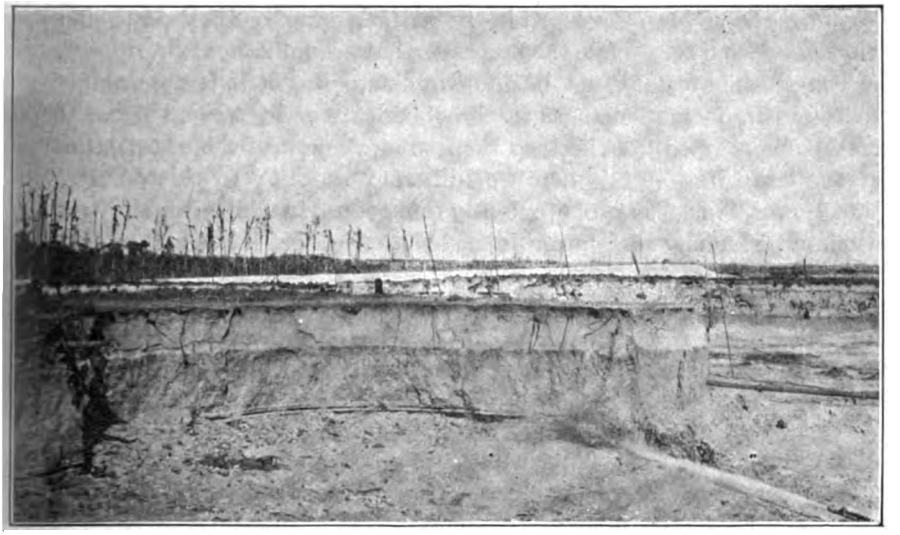

FIG. 2.—REMOVING OVERBURDEN FROM PHOSPHATE ROCK IN FLORIDA BY HYDRAULICKING.

a manner similar to the formation of calcium carbonate deposits in caves. This process is evidently secondary, and being now operative is to be observed in the phosphate boulders themselves, in which all existing cavities are being gradually filled by the accumulation of calcium phosphate. By this process pinnacles are formed hanging from the roof of the cavities, while successive layers of phosphate are spread out over the floor of the cavities. This method of the formation of phosphate deposits has given rise to very high-grade phosphate rock, the Florida hard rock grading, under present methods of mining, 77 to 80 per cent. tricalcium phosphate, while individual specimens contain 84 to 85 per cent.

While the origin of the hard-rock phosphate in its present form is thus

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clearly evident, there yet remains a large field of investigation to determine the chemical processes by which the phosphate is first taken into solution, and is subsequently redeposited. Some of these processes, however, are well understood. That normal phosphate is to some extent soluble in soil waters is well established and fully recognized. In the hard-rock phosphate section of Florida there is practically no surface drainage, the rain water passing directly into the earth. At a lower level the circulation of the water is interfered with and the water may become stationary or nearly so. The check in circulation is due in some instances to masses and beds of clay residual from disintegrating formations. In any case the movement of the water is checked upon reaching the water line. The relation of the phosphate deposits to the ground-water level, and also the evident and probable changes of the water level during geologic time have been discussed in the writer's paper on these deposits previously referred to. While the processes that brought about the deposition of the phosphate may not be fully understood, yet it is apparent that there are important changes in the chemical conditions. Among these may be mentioned the check to the free movement of the water, and the evident mingling at this depth of different waters. In this connection the observations of Reese, previously referred to, in which it is shown that calcium phosphate in solution in carbonated waters is precipitated when the water stands over limestone, are particularly suggestive. As shown in my earlier papers the hard-rock phosphates of Florida are invariably formed directly upon limestone, and need not be sought for elsewhere. Moreover they are thrown out of solution from carbonated waters which pass over and through these limestones, the manner of their formation apparently being entirely in accord with Reese's experiments.

The land-pebble phosphate deposits of Florida are probably derived, like the hard-rock deposits, from the Alum Bluff formation. The processes by which they have accumulated in their present form are, however, strikingly different. The hard-rock phosphates, as has been stated, represent chemical precipitates or replacement deposits, the phosphate having been transported to its present location in solution. The land-pebble deposits, on the other hand, appear to represent materials which are residual from erosion of the parent formation. The hard-rock phosphates occur in sections where the parent formation has entirely disintegrated over limestones. The land-pebble deposits on the contrary are found as a blanket deposit resting upon, and representing a concentration from, the parent formation. The matrix of the land-pebble deposits is, therefore, not strikingly different from that of the hard-rock deposits except as chemical action has modified the residue, particularly by the formation in many instances of siliceous boulders in the hard-rock deposits.

The grade of rock produced from the land-pebble deposits under the

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present methods of mining varies from 66 to 74 per cent. tricalcium phosphate, while individual samples contain 77 or 78 per cent.

### *Tennessee Phosphate Deposits*

As further evidence of the complexity of the origin of phosphates in workable deposits, and of the diversity of ways in which they accumulate, may be mentioned the phosphates of Tennessee. Several distinct types of phosphate are found in Tennessee, only two of which, however, are being mined at present, namely, the blue rock and the brown rock. The blue phosphate occurs as a bedded deposit within, but forming the lowest member of the Chattanooga formation of Devonian age, which rests directly upon limestones of Ordovician age. The beds vary in thickness from zero to 50 in., although the bed of high-grade rock rarely exceeds 20 in. in thickness. In grade the blue rock varies from 30 to 85 per cent. The iron and aluminum in the better grade aggregate less than 3 per cent. The brown-rock phosphates, which at present are being more extensively mined than the blue, occur in irregular deposits lying near the surface and resting upon limestones. The brown rock is largely a shelly rock which breaks up into pieces of varying size. In addition, however, there is a considerable proportion of small pebbles which in mass have a dark brown color. The underlying limestone has an irregular top surface, and not infrequently projects into the phosphate stratum. The limestone is dense and has a bluish cast. It is more or less phosphatic. The age of this underlying limestone is Ordovician. The overburden consists of clay, some phosphate and a limited amount of sand. It is reddish in color, and in thickness is entirely variable, averaging perhaps 8 or 10 ft.

The brown phosphates of Tennessee are very evidently formed *in situ* from phosphatic limestone. Hayes and Ulrich[9] find that at least four limestone horizons have given rise to brown phosphates in Tennessee. The calcium carbonate from the limestone is more or less completely leached out, and is replaced in part at least by calcium phosphate.[10] The rock is thus enriched and becomes a workable phosphate. The leaching of the rock usually begins along jointing planes and unchanged masses of the original limestone in this type of deposit frequently remain as "horses." Two types of deposits are recognized, which are described as blanket and collar deposits. The blanket deposits are those which extend over a considerable area. The collar deposits are formed where the phosphatic limestone comes to the surface around the slope of a hill. The

[9] *Columbia Folio, U. S. Geological Survey*, p. 5 (1903).

[10] Some of the brown phosphate of Tennessee is formed, according to Dr. A. F. Rogers, by replacement of crinoidal limestone by calcium phosphate. (Personal letter, April 1, 1914.)

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collar deposits are necessarily limited in extent, while the blanket deposits may cover considerable areas. The brown phosphates, from their manner of origin, have necessarily accumulated in comparatively recent times.

The blue phosphates, on the other hand, are much older than the brown, having accumulated in their present form during Devonian time It is believed by Hayes and Ulrich that the blue phosphates were originally formed as residual material from, and resting upon, the Ordovician phosphatic limestones, much as the brown phosphates of the present time are formed. After this residual material accumulated the area was depressed allowing the sea to cover the limestone. By the action of the waves in shallow water the residual mass was pretty thoroughly washed, the soil and clay material being sifted out and carried away while the phosphatic material was left to form the phosphate rock as found at present. The sea subsequently deepened, so that shales and other formations were deposited upon the phosphate.

The richest of the blue-rock deposits is believed to have been formed from the Ordovician limestone, known as the Leipers formation. This formation is full of the same minute spiral and other shells that occur abundantly in the immediately overlying phosphate rock. Phosphates were formed from Ordovician limestone other than the Leipers formation, but they are of a lower grade and at present are not workable.

It would thus seem that the blue rock of Tennessee was formed during the interval between the Ordovician and the Devonian, washed during the Devonian, and in this condition was preserved for modern mining operations.

### *Western Phosphate Deposits*

The extensive phosphate deposits of the western United States are interbedded with sedimentary formations, and to this extent resemble the blue-rock phosphates of Tennessee. The rock in these deposits is described as prevailingly oolitic, although an exceptional occurrence is recorded by Richard and Mansfield[11] in which high-grade rock was found to consist of shell fragments regarded by Girty as broken shells of pelecypods. The source of the phosphoric acid and the history of its accumulation in the form in which it is now found in these deposits, if at present obscure, will perhaps, upon further investigation, become apparent.

### *Phosphate Deposits from Guano*

The phosphate deposits of Navassa, a small island in the West Indies, may be mentioned as an illustration of those which are believed to have been formed from guano. In the case of phosphate deposits formed from guano, the phosphate is taken in solution by rain water, and after being

[11] *Bulletin* 470, *U. S. Geological Survey*, p. 376 (1911).

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carried to a lower level is redeposited, replacing the carbonate of the limestone. The rapidity with which this process may be carried on is illustrated by an instance cited by Dr. Albert R. Ledoux[12] in which limestone on one of the south Pacific islands was observed to have been changed to phosphate to a depth of several feet within a period of 20 years, the phosphoric acid in this instance being leached by rain water from recently deposited guano.

## Mining

Phosphate rock is mined either by open-pit or by underground mining. Those deposits having a removable overburden are mined by the open-pit method, underground mining being resorted to only for deposits interstratified with other formations, so that the overburden cannot be removed.

### *Underground Mining*

The deposits mined in America by underground mining include the blue rock of Tennessee, the Arkansas deposits, and for the most part the extensive deposits of the western United States, which are as yet but little developed. In underground mining, ordinarily, operations begin at the surface outcrop of the phosphate stratum, the first rock being uncovered by stripping off the overburden. When the overburden can no longer be removed economically, drifts are run into the bank and the phosphate rock removed, support being given to the roof, when necessary, after the phosphate is taken out. This method of mining is similar to that used in mining coal seams. In the Arkansas and Tennessee mines the phosphate rock is first drilled and blasted. It is then broken up by pick and loaded into tram cars to be drawn from the mine.

### *Open-Pit Mining*

By far the greater part of the phosphate rock produced in America is obtained at present by open-pit mining, in which the overburden is first removed from the rock. The purity of the rock, however, is not materially affected by the methods employed in removing the overburden, and hence it is not necessary to describe these methods in detail. It may be noted, however, that diverse methods prevail, depending upon the thickness and character of the overburden, the magnitude of the operations and the facilities available. Examples may still be found of removal of overburden by pick and shovel, team and scraper, or team and dump cart. It is, however, only a shallow overburden that can be so removed profitably. In the larger mining operations the overburden is removed

[12] *Transactions of the N. Y. Academy of Science*, vol. ix, p. 85 (1890).



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by steam shovel, by means of which the material is loaded into cars, which are then drawn to the overburden dump; or the overburden is removed by the hydraulic method, the material being pumped through pipe lines to the overburden dump. Whatever method is employed a limited amount of overburden remains with the phosphate rock which must be separated in subsequent treatment.

It is not necessary in this connection to describe in detail the methods of removing phosphate rock from the pit, since the purity of the rock is but incidentally affected thereby. It may be said, however, that the phosphate rock is either loaded by pick and shovel on wagons or tram cars to be drawn from the pit, or it is taken up by dredge or by hydraulicking (Figs. 1 and 2). If taken up by the hydraulic method the rock is forced through pipe lines to the washer plant. The character of the deposit determines the methods of removal that may be employed. Where the phosphate is loaded by pick and shovel, such objectionable impurities as siliceous boulders, limestone rock and clay balls are rejected at the pit, and in some mines only the coarse rock is taken, leaving the finer phosphate, clay and sand. As a rule, however, there is no attempt to separate the phosphate from the matrix before removal from the pit.

## Preparation of Phosphate for Market

The phosphate rock mined by the open-pit method must be washed and dried. The methods of treatment described in this paper are those followed in America and particularly in the Florida mines.

### *Washing*

The hard-rock phosphate of Florida when brought from the pit is dumped on to grizzlies with 2- or 2½-in. openings. The fine materials of the matrix pass through while the coarse materials, including phosphate, flint, and limestone boulders, and clay balls, are lodged on the grating. The phosphate boulders are then thrown by hand into a rock crusher near by, while the flint and limestone boulders and clay balls are discarded. That part of the matrix which passes the grizzly together with the rock from the crusher is dropped into a log washer beneath. The practice in the land-pebble mines is somewhat different from that followed in the hard-rock section, the matrix, as pumped from the pit, being thrown as a rule into a large revolving tube, known as a separator, punched "hit and miss" with holes 1 or 2 in. in diameter. As the separator revolves the phosphate pebble, as well as the finer materials of the matrix, fall through the openings and lodge on a screen beneath, while the coarser materials, including sand, rock and clay balls remain in the separator from which they are carried to the waste dump. From the

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screen beneath the separator the phosphate rock passes into the log washer. While this is the usual arrangement in the land-pebble phosphate mines, yet in some of the newer plants it has been found practicable to omit the separator altogether, the rock from the dump being allowed to enter the log washer after passing over a screen of about $\frac{1}{16}$-in. mesh. When the separator is omitted practically all the matrix from the pit passes through the log washers, and it has usually been found necessary in these plants, to install a crusher, which is then placed between the two logs. The larger pieces of bone and phosphate rock, as well as the clay balls, if not disintegrated by the washer, are broken up in the crusher, and the phosphate which they contain is saved (Fig. 3).

The log washer, through which the phosphate rock is passed, consists

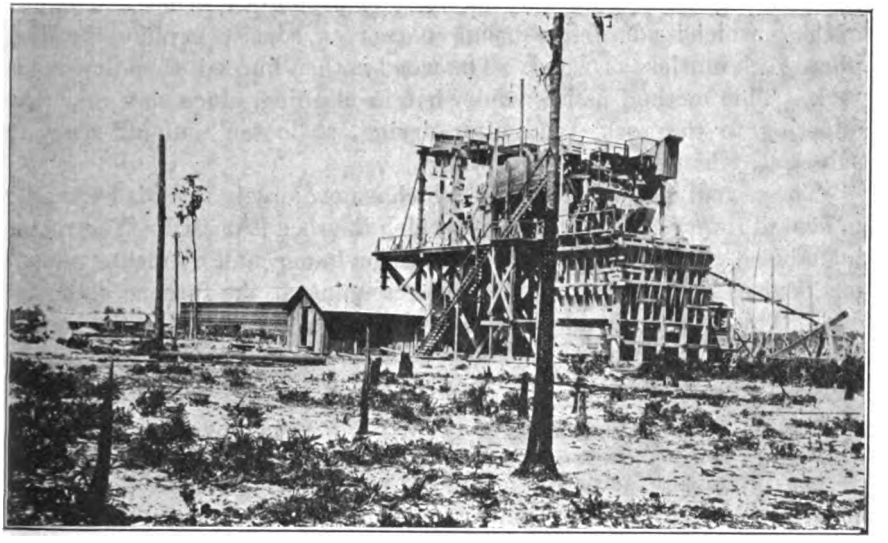

FIG. 3.—WASHER USED IN REMOVING IMPURITIES FROM PHOSPHATE IN FLORIDA.

of two cylinders or logs placed side by side in a box or trough. A series of blades arranged in a spiral is fastened to each cylinder. The trough is niclined, the phosphate being run in at the lower end, and as the logs are made to revolve in opposite directions, the phosphate rock is pushed forward by the blades, meeting as it goes a constant stream of water. By this means the rock is pretty thoroughly washed, the water carrying all the finer materials of the matrix escaping at the lower end or in the newer washers through an opening at the side of the trough. Frequently the phosphate rock is passed through a second log of the same type as the first, and in all cases receives a final rinsing while passing over screens.

In the hard-rock phosphate mines of Florida the coarse phosphate after leaving the rinser is made to pass over a picker belt which is usually,

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made in the form of a large revolving table. The phosphate rock remains on the picker belt during one complete revolution of the table being carefully inspected by men and boys stationed around the table. The inferior rock, clay balls, flint and limestone fragments, so far as recognized, are picked out and discarded at this time, thus bringing up the grade of the shipment. In the land-pebble mines the phosphate rock from the last washer falls on jig screens, the finer of which are $\frac{3}{16}$- or $\frac{1}{32}$-in. mesh. From these screens the rock is elevated by endless cup chains to the loading bin.

### *Drying*

After being taken from the pit the phosphate rock must be dried before being marketed. Two methods of drying are in use. The first of these, which is adapted to drying coarse rock, consists in piling the phosphate rock on ricks of wood. The wood is then burned, thus drying the rock. This method assists somewhat in cleaning, since clay and sand adhering to the rock, tend, after drying, to loosen and fall away in subsequent handling.

The second method of drying, which is now largely used, is by the use of heated rotary cylinders through which the rock is passed. The rock is introduced usually at the cool end of the cylinder, and by means of various devices is made to pass through, escaping at the furnace or heated end. This method is adapted to drying small pebble rock; the coarser rock must be crushed before being dried by this method.

## Improvements in Mining Methods

An important factor in the cost of producing phosphate rock is the necessity of discarding the low-grade rock, as well as that which cannot be properly cleaned or separated from the minerals with which it is associated. It is encouraging, however, to find that through improved methods of mining the amount of phosphate thus discarded is being considerably reduced, while the grade of rock produced is being maintained or advanced. In the modern phosphate-mining plants of Florida, practically no phosphate rock reaches the dump except that which will pass a $\frac{1}{16}$-, $\frac{3}{16}$- or $\frac{1}{32}$-in. mesh screen, or is carried out with the overflow from the side opening in the log washer. In Tennessee where a very considerable part of the brown-rock phosphate is in the form of minute pebbles, settling tanks for saving the small pebble have been introduced in late years. From the log washer the fine material passes through these tanks, and much phosphate that was originally thrown into the dump is thus cleaned and saved, and in fact some of the abandoned dumps in Tennessee are now being rewashed by this method.

However, notwithstanding these improvements in mining methods a

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very considerable amount of phosphoric acid in the form of very fine pebble and soft phosphate still finds its way into the waste dump. It is true that a large part of the phosphate that is thus lost is too high in iron and aluminum to be used in the manufacture of fertilizers under existing processes. It is, however, none the less desirable that the phosphoric acid be saved. To this end it is to be hoped that devices for recovering the very fine pebble phosphate may be further perfected, and that a new process of manufacture may be developed in which the grade of rock that can be used is not so strictly limited.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Tennessee Phosphate Practice

BY JAMES A. BARR, MT. PLEASANT, TENN.

(Pittsburg Meeting, October, 1914)

## *Geology and Mineralogy*[1]

TENNESSEE phosphates are commercially divided into three varieties: Brown, blue and white. The first two only are now of commercial importance. The white phosphates of Perry county are not being mined at present owing to their pockety occurrence. The white phosphate is

FIG. 1.—DEPOSITS OF 75 PER CENT. BROWN PLATE ROCK UNDERLAIN BY SAND.

the highest grade of the three varieties, running often more than 90 per cent. tricalcium phosphate.

The brown rock is found in three varieties, grading from the dense almost vitreous gray rock, 76 to 82 per cent. tricalcium phosphate, down

[1] For a more complete discussion of the geology of Tennessee phosphates, refer to *Resources of Tennessee*, vol. 4, No. 2, Article by J. S. Hook.

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through the ordinary cellular brownish gray rock, looking somewhat like a sandstone, 72 to 76 per cent., to the lowest grade of soft mucky blackish brown rock, 72 per cent. and lower. The solid rock occurs in layers much broken up by joints filled with a corresponding grade of phosphatic sand, clay being the chief impurity.

The blue phosphate looks like a granular bluish limestone. It also varies as to color, the grayish variety being highest in grade, averaging 72 per cent.; the darker blue runs 72 per cent. and lower.

The brown phosphates of the Mt. Pleasant district occur in Bigby limestone of the Ordovician period. In Giles and Hickman counties the brown varieties occur in the Leipers limestone of the same period. Both limestones are a bluish crystalline variety often showing a banded structure upon weathering. These limestones are phosphate bearing often to the extent of 35 to 40 per cent., and are occasionally used as a low-grade fertilizer or for a filler for the manufactured fertilizer.

The principal deposits of blue rock, which are of commercial importance, occur between the Chattanooga shale and the Ordovician limestone, also phosphatic and of the same character as those just described.

### *Prospecting and Valuation*

*Brown Rock.*—The extent, depth and quality of the phosphate deposits are commonly determined by boring with earth augurs and post-hole diggers 4 in. in diameter. Test pits are also employed in conjunction to give a better idea of the character of the strata.

In blanket deposits the holes are sunk 200 ft. apart in squares while in hat-brim formations (ring deposits on a hill side), the holes are spaced 200 ft. on the long axis of the deposit and 50 to 200 ft. on the short axis, the latter spacing depending upon the width between upper and lower boundaries.

Since the overburden is not excessive, rarely being over 30 ft., the light and cheap 4-in. earth auger gives good results with two operators; ¾-in. pipe in convenient joints is used for rods. The post-hole digger is used for penetrating hard substances like flint or plate phosphate rock. It is merely a form of hollow drill, with a 4 by 12 in. barrel of ½-in. steel. When penetrating phosphate a sample or disintegrated core is brought up by the tools and must be removed each time when full, necessitating frequent removals of the diggers from the hole.

About every 400 ft. a pit is sunk to obtain a sample of rock for recovery tests. This will be representative of the rock as mined by hand and sent to the washers, and will also give the true ratio of sand to lump rock with their respective analyses, which is of considerable importance.

The samples from prospecting are prepared for analysis by hand washing in an ordinary wash tub, aving the sand by decantation and

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separating the lump with the aid of a hose and a ¼-in. slotted screen. The recovered products are dried, weighed and analyzed separately. The percentage of recovery is calculated from weights in the usual manner. The weight of a cubic foot of phosphate in the ground should be determined so that the percentage of recovery as found may be converted into such convenient terms as 1 acre 1 ft. deep (43,560 cu. ft.).

*Cost.*—The cost of prospecting will average 7 to 7½c. per foot of hole with augurs or $1.50 to $2 per acre for average conditions.

Tonnages are usually calculated on the basis of a recovery of 1,000 tons[2] dry rock per acre-foot for high-grade deposits, but for the average

*A*. With 4-in. Post-Hole Digger.

*B*. With 4-in. Earth Auger.

Fig. 2.—Prospecting for Brown Rock.

grades this will be more nearly 850 tons per acre-foot. For more accurate work the percentages of recovery should be used as determined in the laboratory, allowing 15 to 20 per cent. in actual mining and treatment.

Especial care is necessary in determining the average thickness of rock for unit areas when the deposits are known to lie in cutters. The phosphate will constitute from 30 to 50 per cent. of the total volume below the surface of the lime table.

The percentage of iron and aluminum oxides remaining in the washed product has an important bearing on the valuation of the rock. The usual

[2] A ton of dry phosphate is 2,240 lb.

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guarantees are given below. For each per cent. in excess of guarantee, 2 per cent. of *BPL* is deducted.[3]

| *BPL* Per Cent. | $Fe_2O_3 + AL_2O$ Per Cent. |
|---|---|
| 70 | 6.5 |
| 72 | 5.5 |
| 75 | 5.0 |
| 76 | 4.5 |
| 78 to 80 | 4.0 |

The total final cost of dry rock f.o.b. cars may be estimated as $2.50, but with a good deposit and modern plant a cost of $2 should be obtained. The resulting profit as determined from the valuation of the rock less working costs should not be less than 20 per cent. per annum on the total investment which will be: Property; this cost will average 25c. per ton of rock in the ground, or if on a royalty basis, 50c. Plant; capacity 75,000 tons per year, $150,000. Mining equipment, $30,000.

*Blue Rock.*—Since blue rock in the known commercial deposits lies mostly under a stratum of shale, the prospecting is necessarily done by diamond drills or drills of the Calyx type. Both give satisfactory results. The outcrops are usually exposed by shallow test pits. Since these deposits are continuous only in one direction, to any great extent, the long and short axes of the deposits are determined at the outset, by tracing the outcrops in gullies. This information materially aids in keeping down the cost of prospecting.

The minimum thickness of the vein for profitable mining at present prices is 24 in.

The sales price of blue rock f.o.b. cars Mt. Pleasant, Tenn., is: 68 to 72 per cent. *BPL* with 2 to 2½ per cent. iron and alumina, $3.50 to $3.75. The lower grades of 62 to 68 per cent. *BPL* are usually sold ground 80 to 90 per cent. through 100 mesh at $3.50 to $4, of which 50c. represents the charge for grinding.

The cost of equipment and dead development work will be practically the same as given under cost of plants for brown rock, for a plant having a capacity of 300 tons in 12 hr.

The cost of blue rock f.o.b. cars with a modern plant averages $1.80 to $2 per ton, to which must be added with some deposits now being worked 25c. a ton to bring the price as quoted f.o.b. cars at Mt. Pleasant, because of extra freight rates.

### *Brown Rock Mining and Treatment*

*Removal of Overburden.*—Occasionally narrow benches are stripped by hand methods. The dirt is more or less undermined by the removal of

[3] *BPL* is a local term for bone phosphate of lime or tri-calcium phosphate, $Ca_3P_2O_8$.

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the underlying rock, and the bank caved over into the previously mined out bench by prying with a long rod from above.

Scattered deposits are stripped by scraper outfits such as are used in railroad work. The outfit usually consists of 5-wheeled scrapers with a hook team extra and a plough team. The work is contracted at 14½c. per cubic yard.

The major portion of the stripping is now done by Class 14 Bucyrus drag-line excavators mounting a 70-ft. boom with a 1½-yd. bucket. This machine is adapted to the removal of overburden that does not average more than 15 ft. With more than this depth the pits become too narrow at the bottom.

In Hickman county where the overburden averages 30 ft. a Class 24 drag-line machine is being installed. This has a 100-ft. boom and a 3½-yd. bucket.

The cost of operating the Class 14 machines is:

| | Per Shift |
|---|---|
| 1 Runner, $150 per month | $6.00 |
| 1 Fireman | 2.50 |
| 1 Teamster | 1.75 |
| 1 Ground man | 1.50 |
| 1 Foreman | 3.00 |
| 1 Team (owned by company) | 3.00 |
| Coal, 2 tons @ $2.40 | 4.80 |
| Cable wear | 5.00 |
| Repairs, oil, and supplies | 3.00 |
| 6 per cent. interest on $13,000. 250 shifts | 3.20 |
| 10 per cent. depreciation | 5.20 |
| Total | $38.95 |

The above is for a machine having caterpillar traction for moving. If a timber and roller machine is used an extra ground man is required.

The capacity averages 1,000 cu. yd. per 10-hr. shift, although often 1,300 to 1,400 yd. are dug under favorable conditions.

While the larger-size machines average more in yardage the operating costs also increase so that the cost per yard is nearly the same.

Two mines use hydraulic stripping because the depth of the overburden is excessive in one and the mining conditions require it in the other. The cost averages 7c. per cubic yard.

The bank is cut down by a hydraulic monitor using a 2 or 2½-in. tip. The water pressure usually needed is 150 to 175 lb. per square inch. The water flows back from the face carrying from 10 to 20 per cent. solids, into a pump well. From the sump the water and overburden are pumped to the waste ponds by an 8-in. direct-connected motor-driven centrifugal pump. The pump requires from 1,500 to 2,000 gal. of water per minute for full capacity, with a motor power of 75 h.p.

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*Mining.*—In the Mt. Pleasant field, the rock is mined by hand since the rock occurs partly below the top of the lime stratum, in cutters of irregular shape. Steam-shovel mining has been tried but is not favored because it does not permit hand sorting of clay and flint in the mines.

The rock, being soft and well jointed, is broken with an ordinary hand pick, and shovelled into 3-yd., 36-in. gauge tram cars. The miners use mostly a 10-tined fork where the cast is short but prefer a long handled shovel when throwing up from the bottom of cutters.

The work is done chiefly by contract, the price being 25c. per tram when one handling only is required. Where two casts are required, 35 to 40c. per tram is paid. The miners keep the track up to the mining face.

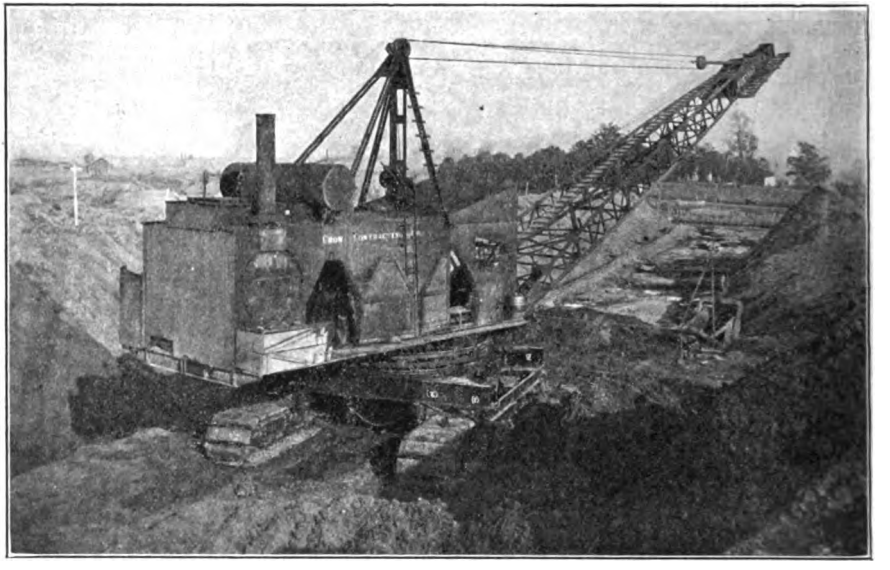

FIG. 3.—STRIPPING WITH STEAM CATERPILLAR DRAG LINE.

In Giles county some deposits are 20 to 30 ft. in depth without interfering lime boulders or cutters. These have been mined by steam shovel but results were not satisfactory because of impossibility of sorting out clay and flint at the mines, which resulted in low grade of rock at the washer. The grade of deposits, now being worked on the lower outcrop, should improve further back in the hill as is the case with most Tennessee rock. Hydraulic-mining methods seem more suitable than steam-shovel operations, in this case.

In Hickman county, because of the depth of the cutters, from 20 to 40 ft., hydraulic mining is successfully employed.

The same equipment is employed as described under hydraulic stripping.

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FIG. 4A.—MINING PHOSPHATE IN CUTTERS.

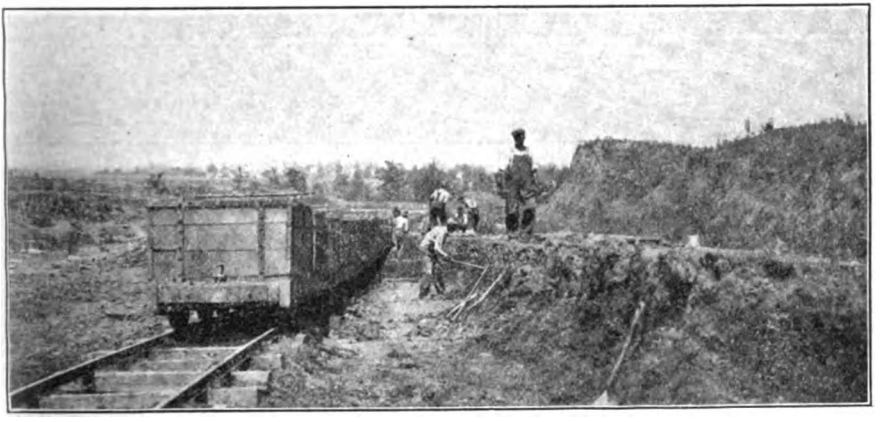

FIG. 4B.—FACE MINING.

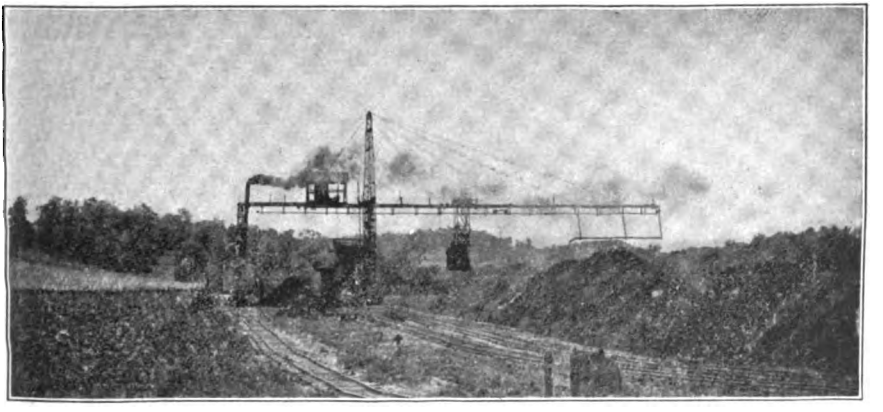

FIG. 4C.—MINING DEEP CUTTERS WITH CANTILEVER.

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The centrifugal pumps used are of very rugged construction, being the same as employed in the Florida pebble district. The shell is of

FIG. 5A.—MINING DEEP CUTTERS, HICKMAN COUNTY.

heavy cast iron, and when worn out it is rejected this being found more practical than the use of liners. The other wearing parts are the end plates which are easily replaced. The shaft is protected from grit, which

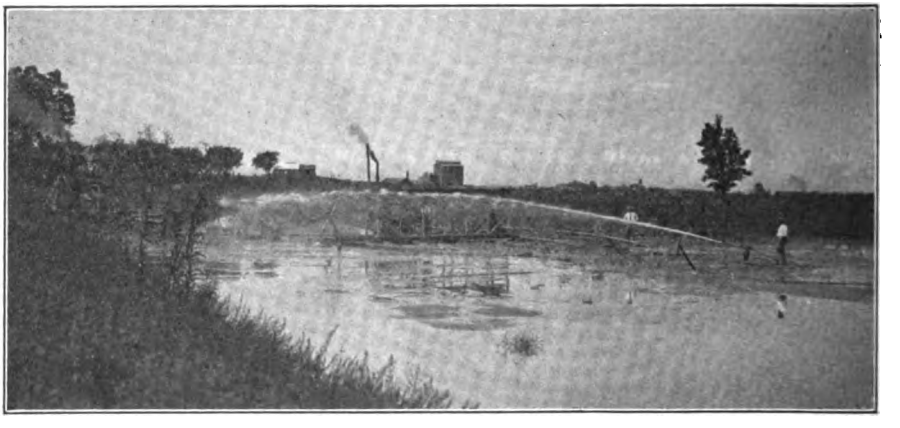

FIG. 5B.—RECLAIMING PHOSPHATE SAND TAILING WITH 4-IN. STEAM-DRIVEN CENTRIFUGAL PUMPING OUTFIT.

gets into the stuffing box, by a stream of pressure water. The stuffing box is submerged in a water seal to prevent air leakage. A bearing of

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ample proportions, with ring oilers and oil reservoir, takes the weight of the shaft and overhanging impeller. The motor is connected to the pump shaft through a flexible coupling, the entire unit being mounted on one base plate. The motor is usually a three-phase, 60-cycle, 2,200-volt, wound-rotor type, operating at 500 rev. per minute with drum controller and grid resistances for continuous duty from half to full speed.

The pump is connected to the suction and discharge lines through heavy rubber tubing called flexibles. Steel tubing with peaned flanges is commonly used for all lines.

*Transportation.*—In the case of hand mining the loaded trams are hauled to the plant by saddle-tank locomotives usually of the 17-ton class for main-line hauls and 12 ton for yard and mine switching. The

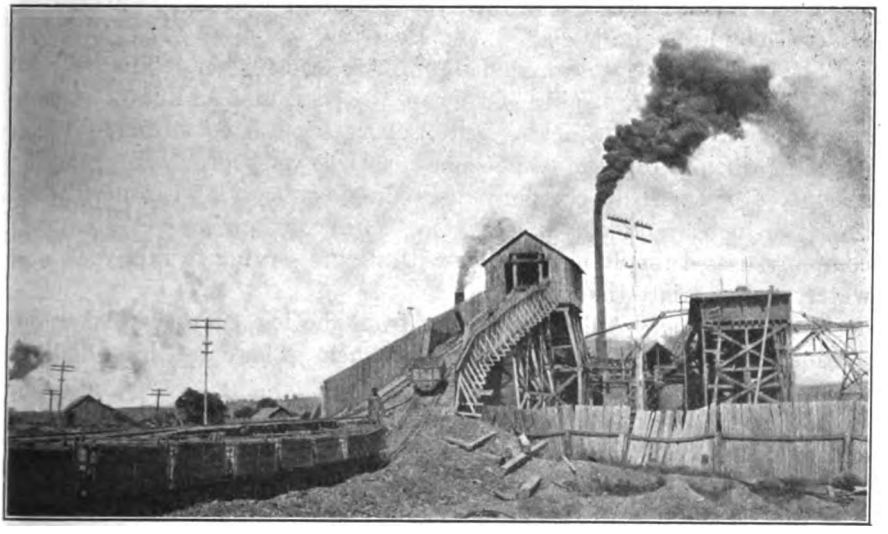

FIG. 6.—PHOSPHATE WASHER, SHOWING SETTLING TANKS FOR FINE SAND ON THE RIGHT.

17-ton dinkey can haul 30 cars on the average mine-track while the 12-ton can handle 20 cars.

On both main line and mine sidings 40-lb. rails are mainly used, but 30-lb. rails are more often used in the mines.

*Washing.*—The tram cars are hauled up an incline by a belt-driven hoist to a tipple where the muck is discharged, with the aid of water, if necessary, into a crusher hopper.

The crusher is of the single-roll type, being more of a disintegrator and feeder. Enough water is sprayed in at this point to enable the crusher to pass a tram load through every 2 min.

The crusher discharges into the low end of a log washer, 25 ft. long, set on a slope of 1 in. per foot. The solids are carried forward and up the

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slope by the paddles of the log which are set on an angle of 45°, at the same time being subjected to a very thorough mixing so as to emulsify the clay, some of which is carried back with the water and phosphate sand to the tail of the log where it overflows an adjustable tail gate, into a launder leading to the sand washer.

Log No. 1 discharges into a second log washer, where fresh water is added.

Log No. 2 discharges into a conical trommel, 30 and 60 in. by 17 ft., having a ½-in. metal jacket with 1½-in. round perforations, which merely acts as a shell for the screen, and support for the inner lining jacket of ⅛-in. metal having ⅛ by ¾-in. slots. A 4-in. pipe extends through the center supplying water under pressure through ¼-in. holes; the water jets are directed against the rock, and effectually remove the undersize and remaining clay.

The undersize of the trommel goes to the sand washer. The oversize is discharged on to a picking belt where the clay, flint and limestone are removed by boys. The picking belt discharges to the wet storage system.

The principal tonnage of phosphate rock is washed according to the above methods. One company uses drag conveyors with perforated bottoms, instead of logs, water being sprayed from above. Another company uses a revolving cylinder with flights serving to mix rock and water, and elutriate the clay.

*Sand Washing.*—All the undersize from the log- or rock-washer department is treated in a separate sand washer. There are three distinct systems used:

1. The sand is washed in a drag classifier of the "El Oro" type locally called the riffle trough. It consists mainly of a double-strand chain flight conveyor with 1¼ by 30 in. flights traveling up a slightly inclined trough, about 40 ft. per minute. The sand pulp is fed near the middle and carried up past sprays of water. This system is not as efficient a washer as could be desired, has high repair costs and a low percentage of recovery.

2. Tank System. (*a*) The sand pulp from the rock washer is run into sand tanks about 10 ft. square and 30 ft. deep. When the tank is full the settled sand is washed out through a slatted opening into the sump of a steam pulsometer pump, which elevates the pulp to the next bin. This operation is repeated five times.

(*b*) Same as (*a*), but the operation is made continuous by hydraulic ejectors or jets, as they are locally called, to pump the sand, as it settles into the succeeding bin. This system avoids the use of expensive pulsometer pumps but requires more water, using 700 gal. per minute to pump 80 to 100 tons of sand in 11 hr.

3. The pulp is fed into the high end of an inclined revolving cylinder provided with internal flights for agitation of the charge. A counter cur-

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rent of wash water is supplied to carry off the elutriated clay. The sand is discharged at the lower end by a spiral and further dewatered in an intermittently discharged settling box. The overflow from the settling

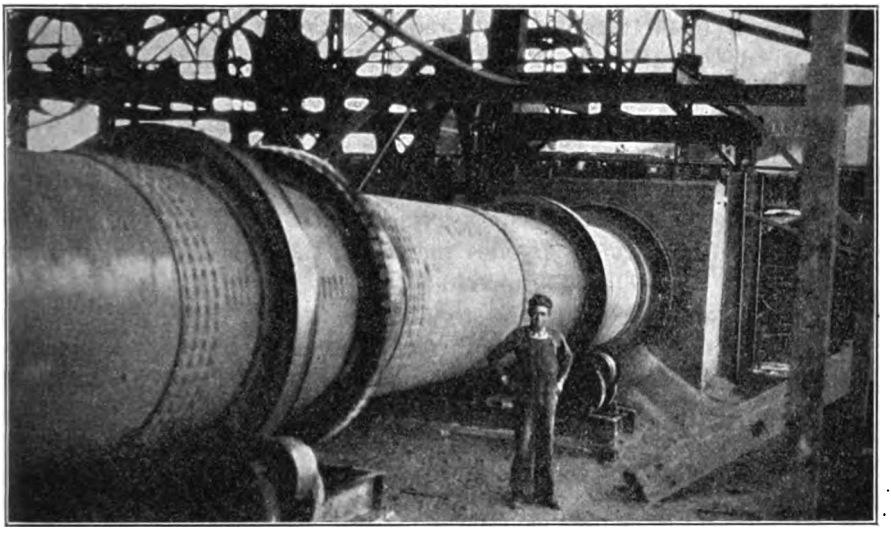

FIG. 7A.—MODERN ROTARY DRIER, 5 BY 40 FT.

box and cylinder washer is further settled in a tank with transverse baffles. The sand caught in this tank is washed out through holes in the side to a drainage pond.

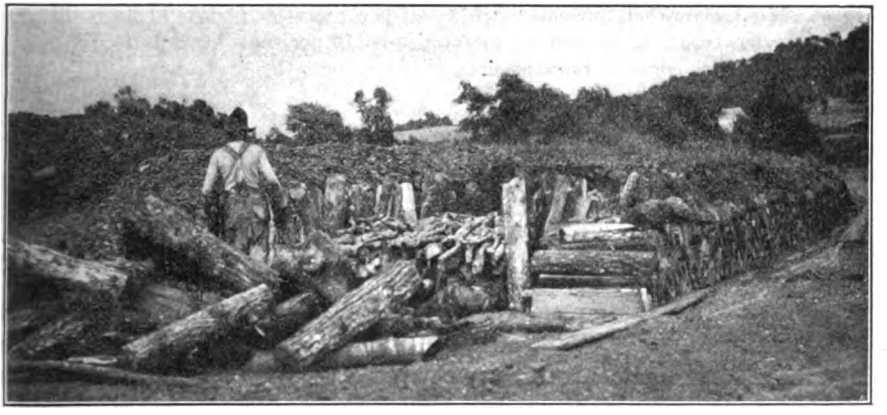

FIG. 7B.—OLD METHOD OF DRYING LUMP ROCK ON PILES OF WOOD.

A new plant is now being designed that will use a dewatering tank the underflow of which will be jetted into a series of Dorr classifiers, fresh water being added to the dewatered sand before feeding into the succeed-



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ing classifier, the final dewatered product going to the wet storage system. Two of the rakes near the head end of each classifier will be replaced by spray pipes. This system has been tested out with excellent results and low water consumption. The overflow from all the above will be settled in a large 20 by 60 ft. tank. The product settled in these latter tanks will be washed separately either in Dorr classifiers or in Dorr thickeners with sprays on the arms.

*Wet Storage and Drainage.*—Since the material comes from the washers with 25 to 30 per cent. moisture, it is very important to provide a drainage system. By storing from two to three weeks the moisture content is reduced to 20 per cent., saving from 2 to 2½c. per ton of rock in coal alone for drying, besides increasing the capacity of a 60 in. by 40 ft. dryer from 90 to 150 tons in 11 hr.

In the older style plants the storage is done in tanks or by building up in dumps with tram cars. The former is necessarily limited in capacity and the latter rendered expensive on account of the amount of labor used. In the modern plants, electric cranes operating clam-shell buckets are used.

Below is given a comparison of costs of the different types of cranes.

| | |
|---|---|
| One 10-ton electric traveling crane 60-ft. span (with 2½-yd. clam-shell bucket) | $10,000 |
| Steel runway 40 ft. to top rail, 400 ft. long | 10,000 |
| Generator capacity in power house, 40 kw. @ $55 | 2,200 |
| Total | $22,200 |
| One 30-ton, 8-wheeled locomotive crane, standard gauge, 40-ft. boom, 2½-yd. clam-shell bucket, | $14,000 |
| Wooden trestle 12 by 400 ft.; material used 75 per cent. heart L. L. Y. pine, with concrete foundations | 4,000 |
| Generator capacity 40 kw. @ $55 | 2,200 |
| Total | $20,200 |
| One 30-ton steam crane same as above | $10,000 |
| Trestle | 4,000 |
| Total | $14,000 |

With current at 1½c. per kw-hr. and coal at $2 to $2.50 per ton the steam crane can be operated as cheaply as the electric, where alternating current is used, but with direct current the cost is more in favor of the electric unit.

*Drying System.*—The next step after the storage of the material in the drainage piles is the reclamation of the drained phosphate and the drying of it in rotary driers or kilns.

The driers first used were light in construction, giving trouble from break downs and rapid wear. They were from 52 to 56 in. in diameter

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and 32 ft. long. The drive was by friction only through the trunnions. Power consumed was high, averaging 30 h.p. per unit including fan. The material was fed into the hot end of the shell. The coal consumption averaged 90 to 100 lb. per ton of rock dried to 2 per cent. moisture, with a capacity of 80 tons per drier.

Modern driers, which are fed at the cold end, are characterized by heavy construction such as is used in rotary cement kilns. The size of the unit is usually 60 in. by 40 ft., and the power for driving 15 to 20 h.p., including fan for forced draft. A view of a modern rotary drier is shown in Fig. 7.

These driers reduce 135 to 165 tons of phosphate from 20 per cent. moisture down to 2 per cent., for the lower capacity given, and down to 3 per cent. for the higher capacity. The coal consumption averages 75 lb. West Kentucky coal per ton of rock dried.

The dried phosphate is elevated to a sizing screen and storage hopper, by an elevator which is usually a single strand of No. 111 combination chain with buckets every fourth link. A traction head pulley and plain sprockets and idlers are used since the use of sprockets with teeth would cut down the life of the chain 75 per cent. Such an elevator lasts three years in ordinary service.

The phosphate is either chuted directly from the storage bin to railroad cars or else carried out to storage sheds by an electric-driven larry. The older plants use cable-driven cars or push cars.

By storing the rock, advantage can be taken of the residual heat to remove at least 1 per cent. of moisture. Cars are sampled at the works so that no credit is allowed for drying out in transit.

The rock is stored in a ground-level shed and loaded into cars by contract labor. This arrangement appears to be crude, but special conditions seem to warrant it.

| | |
|---|---|
| Overhead reinforced concrete storage 10,000 tons capacity, | $80,000 |
| | Per Ton |
| Cost of loading from above bin with car loaders | 1.8c. |
| Interest 6 per cent. and amortization 6.7 per cent | 10.2 |
| Total | 12.0c. |

The above is based on a 15-year mine life with a capacity of 100,000 tons per annum.

| | |
|---|---|
| Electric traveling crane | $10,000 |
| Steel shed 10,000 tons capacity with crane way | 25,000 |
| Total | $35,000 |
| | Per Ton |
| Cost of loading with crane and car loaders | 3.9c. |
| Interest and amortization | 4.4 |
| Total | 8.3c. |

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| | |
|---|---|
| 1 Steel shed, 10,000 ton capacity | $12,000 |
| Or wood construction | Per Ton $5,500 |
| Cost hand loading, contract price | 7.5c. |
| Interest and amortization on $12,000 | 1.5 |
| Total | 9.0c. |

The hand loading has the important advantage of being flexible and selective. The loader does not load wet streaks of rock; he throws out any foreign substance, spots and sweeps out his own car. He loads the car to the required capacity or is fined.

The automatic scraper and belt conveyor type of loading equipment would not be permissible on account of not being selective.

*Production Costs.*—The following were taken from the average costs of a mine during six months of good mining weather:

| | Per Ton, Cents |
|---|---|
| Mining | 0.64 |
| Transportation and team expense | 0.23 |
| Washing | 0.46 |
| Drying | 0.47 |
| Shipping and track expense | 0 09 |
| Total, per dry ton 2,240 lb. | $1.89 |

### *Mining and Treatment of Blue Rock*

The outcrop is usually stripped and mined by hand, back to where the overburden does not exceed over 12 ft. on an average. For extensive outcrop mining a steam shovel is used for stripping.

The underground work is done on the room-and-pillar system, the pillars being recovered on the final round by drawing back. Main and cross entries are driven according to the topography, the rooms being led off from the cross entries alternately every 50 ft. The ordinary dimensions of the room are 25 by 200 ft. The long axis of the room is parallel with the rock jointing so as to obtain the maximum effect of the powder in blasting.

Advantage is taken of an overlying layer of low-grade kidney formation between the blue rock and the shale to make an overcut with air drills and to blast to this face. The skull is thrown back into the gob, acting as a support to the roof. Short timbers are used at the face to protect the workers from the shale roof which tends to slab off.

The rock is loaded and mined by the contract system, into cars holding about 3,000 lb. The loaded trams are pushed to the room entry by hand where they are hauled to the mouth of the tunnel by mules. Here they are made up into trains and hauled to the plant by a 17-ton 36-in. gauge locomotive.

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The surface plant is principally a crusher and drier. The blue rock is not subjected to any washing process.

The order of machinery, etc., in the plant is: Gyratory crusher; 60 in. by 60 ft. rotary drier fed at cold end; crushing rolls set to ½ in.; sizing screen; cable-driven storage larry.

The cost per ton for a modern plant producing 300 tons per 10 hr. is:

| | Per Ton |
|---|---|
| Rock at tipple | $1.30 |
| Drying | 0.10 |
| Crushing, screening and stocking | 0.40 |
| Total | $1.80 |

### *Fine Dry Grinding, Both Blue and Brown Rock*

A considerable tonnage is now being ground for use as a fertilizer direct, or for shipment to fertilizer factories for acidulation.

For phosphate containing lumps ½ in. or more in size primary coarse crushers are used. Following the coarse crushers a fine-grinding mill of the ring roll or Fuller-Lehigh type is used. One company uses a Hardinge mill. The discharge from the fine-grinding mill is elevated to sizing screens where the final size is taken out and the undersize returned to the mill.

Two companies now use tube mills after the fine-grinding mill as described above, to reduce the phosphate to the final size.

The raw rock, for direct application as a fertilizer, is ground to 90 or 100 per cent. through a 100 mesh; for acidulation through a 60-mesh screen.

### *Manufacture of Acid Phosphate*

*Chemistry.*—The final step in the treatment of phosphate rock for the fertilizer trade is its conversion from the insoluble tri-calcium phosphate to the more soluble forms, by direct treatment with sulphuric acid.

The following are the basic reactions of the process:

(1) $Ca_3P_2O_8 + 2H_2SO_4 + 6H_2O = (CaH_4P_2O_8 + 2H_2O) + 2(CaSO_4.2H_2O)$.

(2) $Ca_3P_2O_8 + 3H_2SO_4 + 6H_2O = 2H_3PO_4 + 3(CaSO_4.2H_2O)$.

(3) $Ca_3P_2O_8 + H_2SO_4 + 6H_2O = (Ca_2H_2P_2O_8 + 4H_2O) + (CaSO_4.2H_2O)$.

Reactions (1) and (2) are the ones desired in the manufacturing process. With too little sulphuric acid present in the mix reaction (3) takes place, forming more or less di-calcium phosphate, insoluble in water. With an excess of acid, too much phosphoric acid is formed as in (2), resulting in a moist and lumpy finished product.

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In determining the amount of acid required, account must be taken of the consumption of the principal impurities as noted below.

Carbonate of lime and magnesia are desirable to the extent of 5 per cent. because of the heat furnished in the reaction with the acid, the leavening effect of the $CO_2$ released and the drying effect on the final product in storage.

Fluorite is broken up by the acid, the resulting fluorine combining with the silica and passing off in part as the volatile $H_2SiF_6$. When troublesome, this must be properly handled and washed in scrubbers.

Iron and aluminum oxides are the chief constituents that must be carefully handled and regulated. They not only consume acid but result in a moist lumpy acid phosphate, if in excess of 4 to 4½ per cent.

When the acid phosphate does not dry out well, its quality and per cent. of available $P_2O_5$ may be materially increased by adding raw, fine-ground phosphate rock before storage preferably, but also directly before mixing. Some advocate the use of ground limestone but this will cause some of the soluble phosphate compounds to revert to the insoluble, often to the extent of 2 to 3 per cent.

The manufacture of double superphosahates is done by filter pressing out the soluble portions of acid phosphate, and evaporating the filtrate in pans to 45 per cent. $P_2O_5$. The concentrated phosphoric acid is used to treat a fresh charge of raw phosphate rock:

$$Ca_3P_2O_8 + 4H_3PO_4 + 6H_2O = 3(CaH_4P_2O_8) + 2H_2O.$$

This is a highly concentrated product used mainly for export trade.

*Manufacture.*—The lump and sand phosphate is unloaded from the railroad cars by hand and wheeled in a barrow to the storage pile or to the coarse crusher. From the coarse crusher the phosphate is elevated into feed hoppers of the fine-grinding mills. The mills and the process used are the same as described previously under "Fine Grinding." A mill that is much used, with satisfaction, not previously mentioned, is the three-roll Griffin mill.

The ground phosphate is next treated in a mixing pan, in batches of 1 to 2 tons, the ratio of acid to rock being 1,000 lb. rock to 1,000 to 1,008 lb. 50° Bé. acid. One type of mixing pan is a round cast-iron pan revolving slowly, with one or two sets of stirring arms or plows, also revolving but at a higher rate.

When completely mixed, the contents of the pan are discharged through bottom doors into a square concrete or brick chamber below, called a den. Here the reactions continue aided by the heat.

After remaining in the den for about 48 hr., or until the mass becomes cool, the contents are again discharged through bottom doors, by manual labor, into an elevating and conveying system that carries the acid phosphate to large storage piles.

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The manufacturing process is carried on the year around, but shipment is made principally during three months in the spring and two months in the fall.

Before shipping to the consumer it is usual to add enough ammoniates and potassium salts to make up a complete fertilizer.

During shipping seasons the acid phosphate is reclaimed from the storage piles by hand labor, wheeled to the mixing and bagging system. This consists mainly of a disintegrator which breaks up the lumps of the acid phosphate; a screen takes the discharge from the disintegrator and returns the oversize, the undersize being sent to the mixing machine, where it is mixed with the proper amounts of ammoniates, potash salts and filler. The mixed product is bagged in a bagging machine, weighed and delivered to cars by hand.

It will be noted that there is a large amount of hand labor included in the scheme of operation that might be eliminated by properly designed machinery, and such is beginning to be included in modern plants.

The costs vary widely with the different freight rates on acid and rock. To the costs of raw material the additional charges for acidulation, mixing, and shipping are from $1 to $1.25 per ton. There is a loss of about 7 per cent. in the manufacture of the acid phosphate that must also be taken into account.

A plant with a capacity of 125 tons of acid phosphate per day will cost erected complete $100,000. However, the principal outlay in connection with the fertilizer business is the selling of goods on credit to farmers, which may amount in a business of usual volume to as much as $400,000, for a plant of the capacity mentioned above.

The present selling price of bagged acid phosphate f.o.b. cars Mt. Pleasant, Tenn., is $14 per ton for 14 per cent. $P_2O_5$ material, with $1 added for each unit above 14 per cent. The price of the complete fertilizer varies according to analysis, the base prices being $22.50 and $18.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at a meeting of the Institute. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1914.

# Rock Disturbances Theory of Petroleum Emanations vs. the Anticlinal or Structural Theory of Petroleum Accumulations

BY EUGENE COSTE, CALGARY, ALBERTA, CANADA

(New York Meeting, February, 1914)

ALTHOUGH some of the observers who first paid especial attention to the occurrences of oil and gas in the strata (such as Hunt in 1859, Andrews in 1861, Winchell in 1865, Mendelejeff in 1876, Höfer in 1876, Minshall in 1881, and I. C. White in 1883) seem to have been impressed with the fact that oil and gas deposits were connected with the disturbances in the rocks caused by their upheaval,[1] yet, with the exception of Mendelejeff, and possibly Andrews, these observers do not appear to have understood what is really the one essential factor in the occurrence of oil and gas in the strata: viz., the faulting, uplifting, fracturing, fissuring, and jointing always accompanying even slight rock disturbances, and in certain districts (petroliferous provinces)[2] allowing solfataric hydrocarbon emanations to force their way up from the interior and to reach and impregnate the porous portions of sediments subjected to the disturbances. Instead of this simple explanation to account for the observed connection between petroleum deposits and rock disturbances, most of the observers mentioned above, and many others who have followed them on the same lines since, have entirely reversed the problem. To them, instead of the dissemination of gases from fissures into the sediments, it presented the accumulation and concentration, according to anticlinal or other geologic structures, of oil and gas originally present throughout the whole mass of the strata where they were produced by the decomposition or distillation of organic remains.

Whether this important problem involves accumulation from sources within the oil-bearing strata themselves, or infiltration and dissemination from outside sources, may be easily determined if all preconceived ideas are put aside and the established facts only are considered.

## *Physical Facts*

Sedimentary strata are composed of alternate beds of shales, sandstones, and limestones, most of which are sufficiently close-grained to

[1] West Virginia Geological Survey, vol. 1A, pp. 49 to 53 (1904).

[2] Petroleums and Coals, *Journal of the Canadian Mining Institute*, vol. xii, pp. 273 to 301 (1909).

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have retained from the time of their deposition enough water, held in their minute pores by capillary action, to fill all the spaces between their grains and render them entirely impervious to other fluids. Shales or consolidated clays form the bulk of the sediments and are especially fine-grained saturated impervious rocks, but many beds of sandstone or limestone are also quite impervious. Carll, Lesley, White, and Orton have often referred in their reports[3] to the wonderful impermeability of the sedimentary strata of the oil regions. A sufficient proof is the very strong natural gas pressures always recorded in the gas and oil deposits, and the great differences in the pressure of the gas of different sands tapped at various depths in the same well, or in nearby wells in the same pool or field. These strong pressures and wide differences in pressure have been noted in thousands of wells drilled in the 10,000 or 12,000 ft. of sedimentary strata along the Appalachian oil belt, for instance, from New York State through Pennsylvania, West Virginia, Ohio, and Kentucky to Tennessee.

In order to explain the retention of this gas under strong pressures (up to 1,500 lb. per square inch in some cases) in so many thousands of separate spots and at depths sometimes as shallow as 100 ft. and varying from that to 4,000 ft., we must certainly conclude that the sedimentaries forming the substrata of this vast region and ranging through many geological ages are remarkably impervious. It is for this reason alone that the petroleum deposits have not escaped to the surface, and have been preserved in a multitude of separate pockets, pools, or fields, with different pressures, though often quite close to one another and always comparatively near the surface. Yet without an iota of evidence we are asked to admit the reverse of what we see everywhere to be the case; and we are told by the advocates of the anticlinal theory that these impervious rocks which confine the gas and oil in their present reservoirs are themselves the source from which these substances were obtained. If it is impossible for the natural gas, notwithstanding its strong pressure, to traverse the 100 ft.—or the 3,000 or 4,000 ft. at the most—which separate it from the surface, it is still more clearly impossible that it could have accumulated, according to the anticlinal theory, by traveling miles through the same surrounding rocks to its present reservoirs, from supposed minute organic sources distributed throughout the very same impervious sediments.

Again, if, as required by the anticlinal theory of oil and gas accumulation from within the sediments themselves, these products could circulate freely through the minute pores and mass of the shales, such a movement could and would have commenced immediately after the deposition of the sediments, while they were still lying flat and undisturbed. It is entirely

[3] Pennsylvania Geological Survey Reports, 1885, p. 665; 1890, pp. 12, 13. West Virginia Geological Survey, vol. 1A, p. 63 (1904). *Bulletin of the Geological Society of America*, vol. ix, pp. 95, 96 (1897).

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superfluous to assert that the upward movements of the petroleums in the sediments were started by a slight folding of the rocks, either in the form of an anticline or any other form, "by which the oil, gas and water may have been separated out and the oil concentrated in one locality."[4] The supposed organic petroleum products if they could move freely through the sediments, whether from the action of gravity, hydraulic pressure, or any other cause, would certainly start up toward the surface as soon as formed; they would never stop until reaching it; and therefore there would be no oil or gas at all found in the sediments to-day. In the Californian fields, Madill, Wheeler and other fields in Oklahoma, the Athabasca region of Canada, and many others, the productive sands are in unconformable formations, and sometimes right above the unconformities. To suppose that in these cases the petroleum in the lower bed of the Cretaceous, for instance (Dakota sandstone in Canada, or Trinity sandstone at Madill), is derived from the unconformable Paleozoic rocks underneath, is to ignore the long lapse of time between Paleozoic and Cretaceous times, which was certainly sufficient to exhaust out of the Paleozoic into the air, long before the deposition of the Cretaceous beds, all petroleum products formed from the decomposition of Paleozoic organic remains. If several unconformable formations (and in California there are at least three or four marked unconformities between different productive sands) contain petroleums in the same district and none at all in other districts with the very same sequence of formations, it must be due to some source for the petroleums outside of all these sediments, and to some infiltration, at a period more recent than the youngest productive formation, from a source beneath the oldest.

In all cases, therefore, the anticlinal theory of petroleum accumulations from sources within the sediments themselves fails to explain how the petroleums could possibly enter the porous portions of the sands, and remain there, and not continue their migration to the surface. On the contrary, solfataric petroleum emanations, through the agency of rock disturbances and fissuring, may enter and be retained in a patch of porous sands entirely surrounded by impervious rocks and there separate their component hydrocarbons, and associated gases and vapors more or less according to gravity, the gas working its way to the higher parts of the porous sands, the water, if any, remaining in the lower parts, and the oil floating on the water, between it and the gas. In dry sands, such as the deep sands of Pennsylvania and West Virginia, the oil will naturally work down more or less to the lower part of the porous portions impregnated and will often be found in synclines. That part of the anticlinal theory which provides for a certain amount of separation of the water and of the different petroleums once they have reached a porous

[4] F. G. Clapp: *Economic Geology*, vol. vii, No. 4, p. 380 (June, 1912).

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reservoir, is, of course, true. But even this has been much exaggerated; since the sand reservoirs in the oil and gas fields are very irregularly porous, and far from forming ideal tanks like a bottle or a room. Many impervious streaks or patches of various forms are found in the very heart of their porous portions, and they are seldom continuously porous over large areas. During the periods of disturbance there was also much fissuring and jointing; and, under the strong pressures of the gas always present in the petroleum emanations, these irregular tanks could not be filled up in the theoretical manner mentioned above. Every day, in the drilling of wells this theoretical arrangement of gas first, in the higher portions of the reservoirs, then oil and then water, is entirely reversed. Every day dry holes or oil wells, or salt-water wells are "drilled in" on the top of the anticlines while large gas wells are obtained away down on the slopes or at the bottom of synclines. On the other hand, many anticlines are barren of petroleums, although these anticlines are developed in sedimentary formations where every requisite condition demanded by the anticlinal theory is absolutely fulfilled: namely, fossiliferous strata; porous arched reservoirs; impervious covers; and water in the porous rocks; but where the essential factor is missing: namely, the rock disturbance producing the necessary fissure through which the solfataric hydrocarbon emanations could force their way up to the porous reservoirs.

The structure of many an oil or gas field has no resemblance to an anticlinal structure. A. Beeby Thompson in a paper[5] read before the Institution of Mining and Metallurgy of London, England, in which he reviews the relationship of structure to the occurrence of petroleum, graphically illustrates by many good sections the great diversity of oil and gas field structures and thus plainly demonstrates the reverse of what he advances in the text: namely, that oil and gas fields are generally connected with anticlines and that "an anticlinal structure favors the accumulation of oil"[6] and "played a most important part in the formation of oil fields"[7] Indeed, a number of prolific and prominent oil fields are shown by Mr. Thompson's diagrams to exist in strata presenting structural conditions entirely different from anticlines. Other instances and examples to show that oil and gas fields are found under all sorts of structural conditions have been often furnished by other writers in their studies of the different oil fields of the world, especially of America. It has been found necessary really to transform the anticlinal theory by expanding it into a structural theory including all sorts of other forms. This structural theory was elaborated by F. G.

[5] *Transactions of the Institution of Mining and Metallurgy*, vol. xx, pp. 215 to 237 (1910–11).

[6] *Bulletin* No. 77, *Institution of Mining and Metallurgy*, p. 2 (Feb. 15, 1911).

[7] *Idem*, No. 78, p. 4 (Mar. 15, 1911).

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Clapp in his papers in *Economic Geology* (vol. v, No. 6, *Sept.*, 1910, pp. 503 to 521, and vol. vii, No. 4, June, 1912, pp. 364 to 381). What the author principally proves from his classification of oil and gas fields is really petroleum deposits are not dependent on or controlled by any kind of structure whatever. Such deposits are found, according to this classification: (1) on strong anticlines standing alone; (2) on well-defined alternating anticlines and synclines; (3) on monoclines with change in rate of dip; (4) on structural terraces; (5) on broad geanticlinal folds; (6) on bulged anticlinals; (7) in saline domes; (8) around volcanic rocks; (9) along sealed faults; (10) sealed in by asphaltic deposits; (11) at contact of sedimentary and crystalline rocks; (12) in joint cracks of sedimentary rocks; and (13) in crystalline rocks. To these classes of deposits may be added the following: (14) on gentle slopes or monoclines without any change in the rate of dip, as the Welland field, Ontario, the Madill field in Oklahoma, etc.; (15) in vertical veins cutting across the strata such as the gilsonite veins in Utah, the albertite vein in New Brunswick, Canada, and the grahamite vein near Cairo, West Va.; (16) in quicksilver and other metallic veins; (17) in and along volcanic or igneous dikes; (18) in meteorites;[8] (19) in the volcanic emanations of to-day; and (20) in synclines.

With so many different classes of petroleum deposits, it is clear that the structure in itself is not the controlling factor and that too much weight has been attached to the form or folding of the sediments surrounding the petroleum deposits. In order to make the anticlinal theory fit everywhere unwarranted new names have been given and supernatural properties have been attributed to certain structures (such as "arrested anticlines" and "quaquaversal domes") which in no possible way could of themselves affect the oil or gas accumulations.

Even along the Appalachian oil belt, which is supposed to give many typical examples of anticlinal structures, it is well known that the oil and gas fields are really in the bottom of a deep geo-syncline between the Cincinnati anticline and the Appalachian uplift. These so-called anticlines, on which the oil and gas fields have been developed in that region, are mere wrinkles of small amplitude in the bottom of that deep geo-syncline. The height of each wrinkle is only a few hundred feet at the most, and therefore (if the sands were continuously porous, and the strata in general were as permeable as the anticlinal theory requires to explain the accumulation of the large quantities of petroleums obtained), the oil and the gas would not have stopped on or near the summit of arch of these wrinkles of porous sands, but, if the covers of these sands were impervious, would have traveled along the sands from one arch to the other and gradually up the western or the northern slope of the geo-

[8] *Bulletin*, No.—, U. S. Geological Survey, p. 632.

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syncline until reaching the surface at the outcrops of the sands in Ohio, or northern Pennsylvania and New York State. Many differences in pressure have been noted between the gas found on one of the wrinkles and that in the same sand on the adjoining one. The few hundred feet of water in the syncline between these two wrinkles could not possibly prevent the gas from traveling from one to the other and equalizing that pressure, and, as I have said above, could not possibly prevent the gas from getting out up the slopes of the geo-syncline to the outcrops.

In the Gulf oil fields of Louisiana and Texas, it is still more clearly evident that the particular shape or structure of the salines or mounds was not a factor in the accumulation of oil from the surrounding sediments. These salines cover indeed such small areas that it is only a stretch of the imagination to suppose that the enormous quantities of oil obtained from under them could possibly be derived from the strata affected by what has been called the "quaquaversal doming" under the salines. Moreover, this doming does not extend to the Tertiary or Cretaceous clays and sands surrounding the salines or mounds; and therefore this particular structure could not be a determining factor of oil accumulations from the surrounding sediments. On the contrary, it is well established that the sediments under the salines have been replaced by masses of salt, limestone, dolomite, silica, sulphur, and gypsum, of secondary origin from solutions moving vertically and carrying hot oils, hot water, and hydrocarbon and sulphur gases, and that these products did not extend horizontally into the impervious sediments surrounding the chimney-like channels under the salines; also, that these salines are distributed along lines of deep faults which were evidently the first channels for emanations of the hydrocarbons and other vapors and waters, before they finally came out to, or near, the surface through the chimneys under the salines. What has structure to do with occurrences of oils in deposits of this kind, which are clearly due to the rock disturbance under the salines, and to hot gas-aqueous emanations coming up from below? This explains the fact that oil in large quantities is "struck" at many different levels under the salines, and none is struck, outside of the salines, in the surrounding sediments.

That these are solfataric volcanic emanations is made plain by the occurrences of oil a little further south, along the same coastal plain, in Mexico. There one can actually observe numerous volcanic necks of olivine basalt, scattered at wide intervals, and also distributed along fault lines similar to the lines connecting the salines of Texas and Louisiana; and these volcanic necks are surrounded by large seepages of asphalt and the very prolific oil fields of this region are all developed in close proximity, around these volcanic necks. F. G. Clapp in a paper in *Economic Geology* (vol. vii, No. 4, June, 1912) says with regard to this class of oil deposits in Mexico:

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"In close proximity to the basaltic upheavals, them Tamasopo limestone and overlying formations have been domed upward, forming pockets or places of change in rate of dip at the base of the upheavals and surrounding them, where large deposits of oil have accumulated.

It is known, however, that these Mexican lava cores form really vertical pipes, which may be said to have drilled themselves upward through undisturbed and almost horizontal strata of shales, limestones, and sandstones.[9] and Mr. Clapp himself admits that in his hypothetical cross-section of one of the Mexican oil fields.[10] The supposed action of the doming referred to by Mr. Clapp would therefore have been so infinitesimal as to be entirely negligible in considering the accumulations of oil out of the surrounding sediments. According to Mr. Clapp's theory the Cretaceous sediments surrounding the lava cones of Mexico tenaciously held back in their pores, during long ages, enormous quantities of petroleum products, until such time as the volcanic peaks and needles pierced through them in late Tertiary times. In doing so the volcanic cores tilted only very slightly the sediments immediately surrounding them. It would be impossible for this slight tilting of very small portions of the whole mass of the strata of this region to effect the immediate release from these sediments of the enormous quantities of oil supposed to have been so firmly held by them during long periods; and it can only be concluded that the effect of structure in this case at least has evidently been much overdrawn. Why should so much be demanded from poor impotent structure alone, when it is manifest that these Mexican petroleum deposits are directly connected with vulcanism, and due to solfataric volcanic emanations accompanying the upheavals of the basaltic cones?

Many other instances to show that structure is not a determining factor in the migration and accumulation of oil in the California fields may be found in the reports of Arnold, Anderson and Johnson. These authors summed up their conclusions as follows:[11]

"This migratory faculty may be ascribed entirely to the presence of the associated gas which would cause the oil to fill every crevice offering a point of escape or a point of lodgment."

"The condition of the rocks is the chief factor that controls the matter of where the oil is stored most abundantly."

In California, "many of the 'oil sands' so called, are not true sands, but zones of fractured shale or flint offering interspaces in which the oil can gather."

"Large accumulations in anticlines may be accounted for primarily by the cavities offered by the strata along upward folds, and secondarily by the presence of less pervious beds arching over such folds and affording favorable conditions for the confinement of oil and gas tending to escape."

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[9] *Mining and Scientific Review*, Aug. 24, 1907, pp. 247, 248. *Journal of the Canadian Mining Institute*, vol. xii, p. 281 (1909).

[10] *Economic Geology*, vol. vii, No. 4, Fig. 58, p. 378 (June, 1912).

[11] M. R. Campbell: *Economic Geology*, vol. vi, No. 4, pp. 387, 388 (June, 1911).

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If to these conclusions is added the consideration of the many strong faults and disturbances always plainly in evidence in the fields of California, and also the consideration of the fact that in these fields a number of unconformable formations, from and including the crystalline gneisses to the Quaternary, are productive of oil, it will be readily understood that the migration of oil is effected vertically along fissures and from a source beneath the crystalline gneisses instead of horizontally and more or less under the determining influence of the structure.

The same conclusion has been reached by Washburne in the Florence field of Colorado: viz., that geologic structure has had little or no influence on the movement or the accumulation of oil which is present mostly in open fissures in a zone of shales 2,500 ft. thick, at the bottom of a syncline.

### *Geologic Facts*

Simple and well-known geological conditions and considerations prove just as conclusively that the occurrences of oil and gas in the strata can only be attributed to a process of infiltration, dissemination, and impregnation of the porous portions of the sediments from deep solfataric volcanic sources. As I have already on several other occasions[12] presented in detail a number of facts and arguments in support of these views, I will here only recapitulate the principal points as briefly as possible:

1. In the solfataric volcanic phenomena enormous quantities of hydrocarbon gases and vapors are constantly thrown out in the air in all the volcanic districts of the world. See my previous papers and Dr. G. F. Becker, *Bulletin No.* 401, *U. S. Geological Survey*, for short reviews of this evidence, which, however, will be found at length in the writings of the eminent geologists and chemists who have made special original studies on this subject, notably, Elie de Beaumont, Humboldt, De Lapparent, Suess, Fouqué, Le Blanc, Silvestri, Stocklassa, Brun, Charles Sainte-Claire Deville, Lacroix, Gauthier, Tschermak, Janssen, Libbey and Sokolow. A. Gauthier, for instance, has shown not only that hydrocarbons and other combustible gases are to be found in the lavas or volcanic rocks of to-day, or associated with these rocks in their eruptions, but that enormous quantities of these combustible gases are contained also in ancient igneous rocks, and that by simply warming these rocks to a moderate red heat he could extract from them at least 100 times their volume of mixture of combustible gases and water vapor.

[12] Natural Gas in Ontario, *Journal of the Canadian Mining Institute*, vol. iii, pp. 68 to 89 (1900). The Volcanic Origin of Natural Gas and Petroleum, *Journal of the Canadian Mining Institute*, vol. vi, pp. 73 to 123 (1903). Petroleum and Coals, *Journal of the Canadian Mining Institute*, vol. xii, pp. 273 to 301 (1909). The Volcanic Origin of Oil, Trans., xxxv, 288 to 297 (1904). Fallacies in the Theory of the Organic Origin of Petroleum, *Transactions of the Institution of Mining and Metallurgy*, vol. xxi, pp. 91 to 192 (1911–12).

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After giving the details of his experiments and analyses he says[13]

"Putting aside the reactions pertaining in the melted portion of the interior of the globe, if we consider what happens when certain already crystallized masses of the crust are reheated to red heat on account of their sinking internally, or because lateral pressures of different parts of the crust cause an uplift of the melted rocks from the interior along the points of minimum resistance. When these rocks already solidified once are thus reheated by the incandescent masses it will be seen that they will be bound to give off by all possible vents the gases and vapors produced in the experiments just cited and recorded. From these, as we have seen, one liter of granite gave (at 1,000° and calculated only for this temperature) about 20 liters of various gases and 89 liters of water vapor, that is to say, more than one hundred times its volume of gases. One can understand from this the great explosive force due to these reactions, and that it is not at all necessary to admit the hypothesis of the penetration of superficial or meteoric waters down to the igneous masses as a necessary condition to the production of volcanic phenomena. . . . One will understand that to explain the origin of the water in volcanoes, the nature of the gases emanated and the violence of the eruptive phenomena, that it is sufficient for the deep crystalline strata to be reheated only a few hundred degrees, and the emanation of volcanic gases with their composition (combustible gases) and their formidable pressures, will be the necessary result of this reheating."

Dr. Albert Brun in his *Researches sur l'Exhalaison Volcanique* gives many interesting analyses of the gases found in volcanic rocks, and given off at volcanic or explosive temperatures, that is to say, above 600° or 800° C., which he recognizes as the only true volcanic gases. These analyses show that hydrocarbons are frequent in volcanic rocks all over the earth and that they are sometimes quite abundant. For instance, in an obsidian from the Plomb du Cantal, France:

"The analysis showed that the vapor which is distilled is slightly ammoniacal and carries much bitumen. Heated in the vacuum its fused, vitreous residue is perfectly black with carbon. Moreover, there is enough bitumen present to form on the cold parts of the apparatus oily striations. These oils are soluble in chloroform, with a brown fluorescence, and are combustible with a clear flame."

Many other analyses of lava, obsidian, and other volcanic rocks are stated by Dr. Brun to have given from traces of hydrocarbon to quantities even more abundant than in the instance cited above, and out of 67 analyses 38, or 57 per cent., gave hydrocarbons.[14] The volcanic rocks containing the petroleums were from widely distributed localities, such as Mt. Erebus (antarctic), Armenia, Abyssinia, Java, Japan, Peru, Tamanfaya, Canaries, Iceland, Milo, Vesuvius, Stromboli, Etna, Arran Islands, Scotland, Sweden, Germany, Hungary, Italy and France.

In view of these and other evidences, no geologist can to-day deny that petroleums are now proved to be abundantly produced in the phenomena of vulcanism, whether recent or ancient.

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[13] *Comptes Rendus de l'Académie des Sciences*, 1901 to 1903.

[14] *Economic Geology*, vol. vii, No. 1, pp. 7, 8, 9 (Jan., 1912).



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2. There are no geological phenomena of to-day or of ages past in which, through organic agencies, petroleums—that is to say, the mixtures of hydrocarbons known as petroleums—are being produced, or are known to have been produced. The gradual decomposition of entombed vegetable organic matter in nature to-day, and during the past geological ages, is and has been accomplished in the sedimentary strata at low temperatures and has resulted in the formation of peat, lignite, coal, and anthracite; that is to say, in the formation of oxygenated fixed carbon compounds very different from petroleums. If these "coals" had been distilled it might be argued that "petroleums" were thus formed, but it is, of course, well known that as a general rule they have not been distilled: the necessary heat to bring about distillation was never attained in the unaltered sedimentary strata. At any rate, it is a matter of geological record that the "coals" are found everywhere in the sediments in the undistilled state and not as coke, and that coal and petroleum deposits have no genetic connection of any sort one with the other.

It is equally a matter of geological record that the soft tissues of animal organic matter were not finally entombed in the sediments, where they have left absolutely no trace of their former existence, as demonstrated by the billions of fossils in our paleontological museums and collections, and in the impervious rocks everywhere, without the slightest trace of any carbon compound to be found in them. The former animal organic matter of what is to-day the fossil animal world evidently decomposed fully, or otherwise disappeared entirely, before the final entombment of the hard part of the animal; and therefore no petroleums could possibly be formed from it.

It is insufficient to argue, against such evidence to the contrary, that petroleums must have an organic derivation, because the large petroleum deposits are always found "associated" with sedimentary strata and fossiliferous rocks. This word "associated" is very vague. It takes more than the presence of water in a cave or in a porous rock to prove that that water originated there. Are not volcanic rocks themselves thus "associated" with sedimentary strata—often in successive horizontal flows or laccolitic intrusions interbedded with sediments, or in veins and masses cutting across sediments? Are not the solid petroleums thus found in irregular veins or masses cutting across the sediments? Are not the liquid or gaseous petroleum deposits themselves local and irregular impregnated spots of the porous sediments, not at all in sheets like coal beds? In some districts (petroliferous provinces) they occur in many sands throughout all the sequence of the sediments from the earliest Cambrian to the latest Quaternary, entirely irrespective of the fossiliferous beds, and mostly in sharp sands without fossils—sometimes above great thicknesses of shale strata, sometimes below all shales, as, for instance, the petroleums in the Trenton limestone of Ohio and Indiana,

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while in other districts the same entire series of sediments is absolutely barren of petroleums.

It is also insufficient to say that because volcanic rocks are not everywhere to be seen in the petroliferous districts a solfataric volcanic origin cannot be attributed to the petroleums. As I have already quoted several times in other papers, De Launay in his *Science of Geology* remarks:

> "The dislocations of the earth are more and more observed to have taken place not alone in mountainous regions but even in regions of plains." . . .
>
> "All the regions of the earth, probably without exception, have been subjected to dynamic movements to which are connected igneous manifestations of internal origin."

These disturbances furnished the necessary fissuring of certain belts of strata, whether much disturbed and uplifted, as in California, Roumania, and Galicia, or sometimes lying still comparatively flat and apparently undisturbed, as along the Appalachian belt in Pennsylvania and West Virginia. In all cases, however, the imperviousness of the bulk of the strata, especially the shales, and the fact that these readily caved in and sealed the fissures, prevented the petroleums from entirely escaping to the surface. Through this fissuring of the strata the pent-up solfataric hydrocarbon emanations came up from the interior, losing some of their pressure as they forced their way up through the fine minute fissuring and the imperfectly porous sands. Hence the differences of pressures recorded in the different sands of the same field and the higher pressures recorded in all fields as the petroleums are found in deeper and deeper sands. Hence the fact that in every field or district, not one but a number of different sands, belonging to a number of different geological formations, sometimes unconformable, are impregnated with the petroleums in irregular spots, here and there, along the structural lines of disturbances, while much larger areas of the same series of sediments in the neighboring districts are absolutely barren.

The distribution of the Mexican and of the Texas and Louisiana oil fields along lines of fault has already been referred to, as well as the evident connection of the oil of these fields with vulcanism.

Similarly the oil fields of California belong to much disturbed and fractured belts bordering the Coast Range on each side for many miles. There also the migratory ascent of the hydrocarbons through faults and fissures is most plainly attested by numerous asphalt veins, and by tar and gas springs. That these faults and disturbances, constantly referred to in the reports of Arnold and Anderson, have brought up the hydrocarbon emanations from below the crystalline gneisses and granite is evidenced by the fact that in Placerita canyon, 5 miles east of Newhall, Los Angeles county, a very light oil (between 50° and 60° B.) is produced

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from crystalline gneisses which overlie the San Gabriel granite.[15] Above these crystalline rocks, the oil is found in California to be stored in the porous reservoir-rocks, or in the seams and joints of any and all the strata affected by these profound disturbances in a geological column of some 26,000 ft. of Cretaceous, Tertiary, Fernando, and Quaternary sediments, those of each period lying unconformably on the next lower, and the lowest unconformably on the crystalline rocks. It cannot be imagined that the petroleums contained in any one of these rock series can have originated from organic remains in the underlying unconformable series, since they would have been lost at the surface, if they had migrated at all, during the long intervals marked by the unconformities, unless a still more improbable process be imagined: namely, that in each case the organic remains accommodatingly waited until the end of the long period marked by the uncomformity, before beginning to be decomposed and transformed into petroleums, and to migrate upward. There is only one possible explanation: solfataric volcanic emanations of hydrocarbons coming up from below the crystalline rocks along the fault lines and in the zones of disturbance, at repeated periods of dynamic movements of the Coast Range. Some of these movements must have been very recent to explain the oil in the gravels of the Quaternary and the large seepages often found at the surface.

If the other oil districts of America are considered broadly it will be seen that the Appalachian fields, the Northwestern Ohio and Western Ontario fields, the Illinois fields and some of the Indiana fields, and the Mid-Continent fields, all show linear distribution of their petroleum deposits more or less parallel to the Appalachian uplift, except some of the Oklahoma pools, which follow the direction of the Arbuckle or Wichita Mountain uplifts. Along these oil belts the numerous petroleum-bearing rocks do not represent fixed geological horizons—the oil and gas rising from various beds of different ages; and outside of these oil belts, structurally favorable folds are barren of petroleums in the same geological horizons.

Following the Appalachian oil belt, for instance, from Tennessee and West Virginia through Pennsylvania to New York State, the petroleums are found in lower and lower rocks in the geological scale, until they are found right on the top of the Archæan in the Potsdam sandstone. The Archæan, therefore, or the igneous magma below it, must be the final source. It would be puerile indeed to assume one hundred different sources in the sediments between the one hundred different productive sands of this oil belt, since it must then be assumed also that every member of the sedimentary series separating two oil-bearing sands is the im-

[15] *Bulletin No. 309, U. S. Geological Survey*, pp. 100, 101 (1907).

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pervious cover, and cannot therefore be considered as the source, of the petroleums in the sand below; and since it has to be admitted, after all, that the Archæan rocks, or the igneous magma below the Potsdam, is one at least of the sources.

This peculiar occurrence of the petroleum deposits in the sedimentary strata of all ages, yet in certain districts only, along zones of tectonic structural disturbances of these strata, where they align themselves in "petroliferous provinces," as do the metals in "metallogenetic provinces," is exemplified not only in North America but all over the earth. In Russia the petroleum deposits follow the Caucasus uplift from the Tamansk peninsula in the northwest to the Apcheron peninsula in the southeast for a distance of 750 miles. In Galicia and Roumania they follow the Carpathian range and turn with it in a grand sweep of more than 500 miles from a northwest to southeast direction in Galicia to an east-and-west one in Roumania. The same association of naphtha to a dislocated zone along the Kurdestan chain in Persia, has been well described by J. de Morgan.[16] The Tertiary tectonic and igneous "girdles" around the Pacific and the Caribbean sea are followed by an immense Tertiary "petroliferous province," of which the oil fields of California, Alaska, Japan, Borneo, Sumatra, Java, New Zealand, Peru, Colombia, Venezuela, Trinidad, Mexico, and Texas are salient points.

The constant recurrence of hydrocarbons in volcanic and igneous rocks, volcanic emanations, metallic and other veins, meteorites, comets and other stellar bodies, clearly demonstrates that petroleums are inorganic. The hydrocarbon emanations sometimes follow closely the volcanic lava, as in the Mexican and other oil fields, but they need not be, and often are not, accompanied to the surface by volcanic rocks. They must, notwithstanding, everywhere be held to be of a solfataric volcanic nature, as shown by their peculiar position along the tectonic structural disturbances, and because, moreover, they could not originate in the mass of the impervious sediments, and could only travel through these by means of faults or fissures from deep sources. Besides, with their associated salt and sulphur they are identical with solfataric volcanic vapors emanating from all the volcanic districts of the earth, and are absolutely unlike anything else in nature known to be in the active process of formation at the present time. Like volcanic vapors these hydrocarbons are always found under high pressures, often giving rise to violent explosions and sudden outbursts in enormous quantities, after earthquake shocks, and along disturbed dynamic lines; and, like volcanic vapor, they are also found at times to be still hot and associated with hot waters.

[16] *Annales des Mines*, 9th ser., vol. i, 227 to 238 (1892).

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## Discussion

Hans von Höfer, Vienna, Austria (communication to the Secretary*). —I will confine my remarks at first to the question which I raised in my paper as to the volcanic origin of petroleum, following thus the order observed by Mr. Coste in his reply, which now lies before me.

1. Methane is always the same chemically, in whatever manner it may have originated.

2. To the geology of petroleum the age of the crystalline schists, whether gneiss or phyllite, is of the highest significance. The occurrence of oil as a primary deposit in a crystalline schist of Archæan age, would warrant the conclusion that it could not have an organic origin, since in that age no organisms existed. But this inference would not hold good as to oil from a crystalline schist of Jurassic age, like that of Newhall, Los Angeles county, Cal., since in that period abundant organic life inhabited the earth, and therefore such oil might have an organic origin. Hence the essential point in the present discussion is not the crystalline schist, but its geological age.

What the Newhall crystalline schists were originally, *i.e.*, immediately after their deposition, we do not know; but it is not impossible that they are the product of the metamorphosis of sand, or of sandstone, as is, for instance, very probable for many gneisses.

3. Concerning the source of bitumen in the quicksilver deposits of California, I spoke with reserve in my paper, leaving the question to my American professional colleagues. I am now fortunately able to cite a recent American authority on that subject, namely, J. Allen Veatch, who says, in his very interesting paper on the The Genesis of the Mercury Deposits of the Pacific Coast:[17]

> "The theory that this oil originated from organic matter precipitated into the open fissure is supported by the fact that the Sierra dikes do not contain oil."

Mr. Coste's assumption that only such solutions as circulated exclusively in veins and fissures impregnated the wall rocks, and that the opposite course would be impossible, surprises me greatly; since we mining geologists have for more than half a century recognized this opposite course as lateral secretion.

4. On the strength of the personal observations of many years, as well as of the literature accessible to me, I can only repeat the declaration that gaseous, liquid, and solid bitumens occur in ore-bearing veins and eruptive dikes very seldom, and always in small quantity. I believe that I was one of the first to collect and publish the data concerning

* Received July 2, 1914.

[17] *Bulletin*, No. 86, February, 1914, p. 225.

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the occurrences of bitumen in ore deposits.[18] In that paper Mr. Coste will find more examples than he appears to be acquainted with. He assumes without proof that the bitumen found in eruptive dikes is of volcanic origin. I assume that the bitumen contained in the intrusive magma has come from bituminous or carboniferous strata, through breaks in which it ascended. The two hypotheses face each other, both lacking direct, positive proof. My explanation has been adopted very recently by W. W. Arschinom[19] for the occurrence of anthraxolite in the eruptives of the Crimea, and by Falkenberg,[20] for the bitumen found in the pyrite deposit of Lilleboe, Norway. As I have already observed, Mr. Veatch confirms it for the California quicksilver deposits.

The etymological argument based by Mr. Coste upon the relation between "distilling" and "driving away" escapes my apprehension.[21]

5. The significance of the volcanic necks of Mexico has long been clear. The primary oil deposits were elevated by these eruptions, and acquired thereby a dome-like form, which (as I demonstrated at the Petroleum Congress of 1903 in Bucharest) is peculiarly favorable to the formation of oil. The latest publication concerning the Mexican oil fields—that of the distinguished American expert, I. C. White,[22] confirming Villarello's view, establishes the organic origin of the oil, and shows also that primary deposits of it occur in the Cretaceous strata. Mr. White says:

> "There can be practically no doubt that the mother rock or main reservoir of petroleum along the Gulf coast of Mexico is the Tamasopa limestone of the Mexican Cretaceous terranes."

It is thus impossible to maintain that the occurrence of oil in Mexico furnishes "the clearest evidence in the world of the solfataric volcanic origin of oil in enormous quantities," as Mr. Coste declares. On the contrary, the organic origin of the Mexican oil is beyond doubt.

6. I object to Mr. Coste's volcanic hypothesis, that if it were true, then, as a logical consequence, oil ought always to be found in the neighborhood of volcanoes and deep faults and fissures, which is not the case. He answers this objection with a speculation concerning the Carpathian oil deposits, which seems to ignore the main point—namely, the occurrence of the volcanic rocks in the inner, oil-poor Carpathian

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[18] In my Erdolstudien (Studies of Petroleum): See *Sitzungsberichte der k. k. Akademie der Wissenschaften, math. naturw. Klasse*, vol. iii, part 1.

[19] *Lithogaea*, G. 50, p. 15.

[20] *Zeitschrift für praktische Geologie*, vol. xxii, No. 3 pp. 141, 152 (Mar., 1914).

[21] TRANSLATOR'S NOTE.—Dr. Höfer apparently refers here to the actual derivation of "distil" from the Latin "stilla," a drop, and its primary meaning, "to fall, or (transitively) to let fall, in drops." The word, therefore, does not etymologically convey the notion of "driving away."—R. W. R.

[22] *Bulletin of the Geological Society of America*, vol. xxiv, No. 2, p. 253 (June, 1913).

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circle—and to pursue only the notion of a possible difference in the formation of fissures on the two sides of the range. My knowledge of the Carpathians for 30 years past enables me to say positively that anticlines and fissures are found occurring in the same manner on both sides.

With regard to the conditions in Pennsylvania, and the occurrence of volcanoes far distant from any oil deposits, I would simply refer to what I said in my former contribution.

Let me now briefly consider the objections of Mr. Coste to the organic origin of petroleum. I number my paragraphs to correspond with his.

I. To the question, "Why is not this process in active operation in the world to-day?" I reply that during the last century immense accumulations of the dead bodies of marine or fresh-water animals have been repeatedly observed. In my book on Petroleum[23] I have given a brief summary, and in my Petroleum-Studies, already cited, a more detailed discussion of these accumulations. I may mention as an example the masses of dead fish in Florida, described by the celebrated American biologist, Agassiz, who said of them that such accumulations of corpses might very well form oil-bearing deposits. The actual development of petroleum from the soft parts of mollusks was observed by Bertels[24] in the Guilaja Balka, on the north side of the Caucasus. Where are oil deposits formed to-day by active volcanoes? I would be grateful to Mr. Coste for a few instances.

II and III. In the formation of petroleum, as is usual in processes of decay, carbonic acid is generated, and when prevented from escaping, effects the solution of the calcareous molluscan shells, which accounts for the absence of such shells from oil deposits. Grzybowski studied the foraminiferæ accompanying the Carpathian oil deposits, and found the siliceous shells to occur frequently and in good preservation, while the calcareous shells were either altogether gone or present in scanty amount and corroded condition.

IV. There are in the world so many "tectonic and orogenic disturbances," like those in the oil regions, and yet without any oil. There must be first a primary deposit, and only afterward can such disturbances specially concentrate the oil in some places, excluding it from others. Mr. Coste apparently fails to make sufficient distinction between formation and accumulation.

V. The conditions under which organic material is preserved in bays are by no means vague. Not alone I, but also most petroleum geologists, have reached the conclusion that a petroleum deposit, if it is to be productive, must have clay, clay-slate, or similar material as a hanging

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[23] *Das Erdöl und seine Verwandten*, 3d ed., pp. 256 to 260.

[24] *Erdöl und seine Verwendung*, p. 246.

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wall; that is to say, the buried organism must have been excluded as early as possible from the air, so that the change which takes place shall be, not destruction by oxidation, but that process of decay which plays a leading part in the formation of mineral oil. The air-tight roof hinders also the escape of the oil after it has been formed.

VI. That in chemical processes longer periods of time may take the place and produce the results of higher temperatures, is beyond doubt. I may be permitted to remark, in passing, that in the transformation of fatty organisms, a heat of reaction is developed, which still betrays itself in the small geothermic scale of the oil fields. The formation of coal likewise liberates heat; but that it proceeds differently from that of petroleum need cause no surprise, since its original material—plants, and particularly cellulose—is entirely different from the fats out of which petroleum is formed.

In defending the theory of the organic origin of petroleum, I have been necessarily brief. If Mr. Coste finds my statement inadequate, I beg him to study without prejudice the many pages of my books, in which I have given proofs of this theory, based upon nearly 40 years of study of the subject.

With this suggestion, I regard the present discussion as closed, believing that I have sufficiently shown Mr. Coste's position in favor of the volcanic origin of petroleum to be entirely untenable.

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# Finishing Temperatures and Properties of Rails

BY GEORGE K. BURGESS, J. J. CROWE, H. S. RAWDON, AND R. W. WALTENBERG,
U. S. BUREAU OF STANDARDS, WASHINGTON, D. C.

(Pittsburg Meeting, October, 1914)

NOTE BY THE EDITOR.—*This résumé of a Technologic Paper which is published in full by the U. S. Bureau of Standards, is brought before the membership of the Institute with the object of affording an opportunity for the open discussion of results of an investigation by the authors named above, of some phases of the manufacture of steel rails which are of great importance to manufacturer and consumer alike. This résumé will be presented at the Pittsburg Meeting of the Institute. Discussion is invited, by either members or non-members of the Institute. The discussion may be sent in writing to the Editor, in advance of the meeting, or should preferably be presented in person at the meeting. To those who desire to discuss, copy of the full Technologic Paper (No. 38), as published by the Bureau of Stanaards, will be sent upon request addressed either to the Bureau of Standards, Washington, D. C., or to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y.*

## *Summary of Researches and Conclusions*

The main objects of this investigation were to determine, from measurements taken at representative rail mills, the present American practice regarding the temperatures at which rails are rolled; to demonstrate the ease and accuracy with which such temperatures may be measured, without interfering in any way with the manufacture of the product; to find out what the "shrinkage clause" in rail specifications really means; and, finally, to determine some of the physical properties of rail steels, particularly those of interest in manufacture, some of which seem to be insufficiently known as yet.

1. *Ingot Temperatures.*—Measurements made in four mills show that ingots for rails are rolled at temperatures ranging from 1,075° to 1,150° C. (1,965° to 2,100° F.), and that, in a series of 20 to 40 ingots, the variation in temperature from one ingot to another is only 10° to 20° C.

2. *Finishing Temperatures.*—Rails are finished at temperatures ranging broadly from 880° to 1,050° C. (1,615 to 1,922° F.), but usually ranging within 50° C. of 935° C., and occasionally above 1,100° C.

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3. *Finishing Temperatures of Different Rail Sections.*—With uniform mill practice, the rails of 100-lb. section will be finished at a temperature about 10 to 20° C. hotter than rails of 90-lb. section, and about 50° C. hotter than those of 75-lb. section.

4. *Uniformity of Finishing Temperatures.*—It is possible to roll rails so that all rails from the same original position in the ingot—as *A* or *B*, etc.—will be finished within ± 10° C. of the same temperature.

5. *Temperature Measurements.*—The finishing temperatures of *each* rail may be measured at the hot saws, by means of suitable optical or radiation pyrometers, without interfering in any way with the manufacturing operations.

6. *"Shrinkage Clause" in Rail Specifications.*—A comparison of the shrinkage clause in American rail specifications (for example, those of the A. S. T. M.) with the expansion of rail steel shows that this clause permits rails to be finished at temperatures as high as 1,120° C. (2,045° F.), which is 450° C. (840° F.) above the critical range of rail steel, and which is also above the temperature at which many ingots for rails are actually rolled in practice. Such a shrinkage clause, therefore, does not serve the avowed purpose of limiting the finishing temperature to a value slightly above the critical range.

7. *Critical Ranges.*—For the 10 samples of open-hearth and Bessemer steels examined, the maximum point of the critical range on heating was determined to be within 7° C. of 732° C. (1,350° F.); the critical range on cooling was determined to be between the limits of 680° and 650° C. (1,256° and 1,202° F.).

8. *Melting Range.*—The melting, or freezing, range for rail steels extends from about 1,470° C. (2,680° F.) to nearly the melting point of iron, located at 1,530° C. (2,786° F.).

9. *Expansion.*—The expansion for open-hearth and Bessemer steels is not the same. Above 800° C. (1,470° F.) the expansion of both increases linearly with temperature. The linear coefficient per degree centigrade has the following mean value between 0° and 1,000° C.:

For Bessemer steel (carbon 0.40 to 0.50 per cent.), $\alpha = 0.0000146$.

For open-hearth steel (carbon 0.65 to 0.70 per cent.), $\alpha = 0.0000156$.

Shrinkage measurements on open-hearth rails (C = 0.71 per cent.) gave $\alpha = 0.0000161$.

10. *Rate of Cooling.*—A rail of 100-lb. section, cooling freely in air from a uniform temperature of 1,070° C. (1,960° F.), reaches its recalescence point (670° C. = 1,232° F.) in about 8 min. 30 sec. The maximum difference in temperature between center and outside of head during this cooling is about 85° C. at 1,000° C. (1,832° F.), becoming 0° again at 670° C.

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11. *Other Observations of Properties.*—Chemical, microscopic, and mechanical tests were made on a number of samples of rails whose temperatures of rolling had been observed, but there appears to be an insufficient degree of correlation to warrant associating very specifically any of the characteristics defined by these three methods of examination, either with the temperatures of rolling here observed, or with each other.

### *Observations on the Temperatures of Rolling*

The results of the observations on the temperatures of rolling at the four different mills is summarized in Table I. The only considerable element of non-uniformity of the measurements at the several mills lies in the fact that the distance and time from the finishing pass to the hot saws, at which latter point the observations were made, is not the same. The time for mill *D* is about one-half of that for mills *A*, *B*, and *C*, while the ratio of distances is roughly *A*, 4; *B*, 5; *C*, 3; and *D*, 1.

### *Importance and Possibility of Temperature Control of Rail Rolling*

The undesirable effects of rolling rails too cold and too hot were very early recognized, and have been stated many times by competent authorities, although it should be pointed out that there is not complete agreement, or conclusive evidence, regarding the effects of finishing rails too hot. The argument is that rolling below the critical range distorts and weakens the crystalline structure, while rolling and finishing at too high temperature above the critical range produces a coarse-grained steel, which is weaker than the finer-grained structure obtained by doing work continuously down to the critical range. Indeed, the researches of Wickhorst[1] showed that, with increase of finishing temperature, there was a slight decrease in ductility and a notable increase in the size of the grain.

The introduction of temperature control in rail mills presents no unusual difficulties from the pyrometric point of view. What is desired is a measure of the actual temperature of each of the rail bars as they emerge from the finishing pass, or, better, of the blooms as they enter the rail mill. The present shrinkage clause in rail specifications has no significance whatever, since it is placed so high as practically not to limit the upper rolling temperature at all. It is furthermore an unsatisfactory and inaccurate gauge, and does not permit the exercise of control of the operation, because it can only be used after the rails are completed and cold. This last objection also applies to the determination of the temperature of rolling by means of microscopic examination.

[1] M. H. Wickhorst, Influence of Rolling Temperatures on the Properties of Bessemer Rails, *Report No.* 21, *to Rail Committee, American Railway Engineering Association*, (November, 1911).

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TABLE I.—*Summary of Ingot-Rolling and Rail-Finishing Temperatures in Centigrade Degrees*

| | Mill A[a] | | | Mill B[a] | Mill C[a] | Mill D[a] | |
|---|---|---|---|---|---|---|---|
| Average temperature rolling ingots | 1140[a] | 1082[b] | | 1102[c] | 1087[d] | 118[e] | |
| Average time in blooming mill, secs | 120 | 121 | | 91 | 71 | 37 | |
| Average temperature of finishing rail | 924[f] | 976[f] | 988[k] | 939[n] | 911 | 1047[n] | 992[n] |
| | 918[g] | 964[g] | 962[m] | 928[p] | 928[f] | | |
| | | | | | 901 | | |
| | | | | | 920[g] | | |
| | 906[i] | 942[i] | | | 883 | | |
| | | | | | 904[i] | | |
| Number of rails observed | 120 | 70 | 80 | 312 | 171 | 186 | |
| Weight of rails, lb.-section | 75 | 90 | 100 | 72 | 90 | 100 | |
| Type of rail | Bess. | O.H. | O.H. | O.H. | O.H. | Bess. | O.H. |
| Location of point of observation | Hot saw | | | Hot saw | Cooling Bed No.1 | Hot saw | |
| Distance from last pass to point of observation in arbitrary units | 4 | | | 5 | 3 | 1 | |
| | Blooms not reheated | | | Blooms not reheated | Blooms reheated | Blooms reheated | |

[a] Observations taken on 20 ingots; maximum variation from average ± 10°
[b] Observations taken on 19 ingots; maximum variation from average ± 16°
[c] Observations taken on 30 ingots; maximum variation from average ± 12°
[d] Observations taken on 29 ingots; maximum variation from average ± 17°
[e] Observations taken on 43 ingots; maximum variation from average ± 15°
[f] Rails located in ingot at *A* and D
[g] Rails located in ingot at *B* and *E*
[i] Rails located in ingot at *C* and *F*
[k] Rails located in ingot at *A* and *C*
[m] Rails located in ingot at *B* and *D*
[n] Rails located in ingot at mean of *ABCD*
[p] Rails located in ingot at mean of *EFGH*
[q] Blooms not reheated before rolling into rails
[r] Blooms reheated before rolling into rails.

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There are many types of optical and radiation pyrometers available to solve the problem of direct temperature control. These pyrometers have given excellent service on other technical fields, and with very small cost of installation and maintenance. In the researches made by the authors of this paper all observations were made without interference with the regular operation of the mills, and no special arrangements or in-installations were found necessary for manipulating the pyrometric apparatus.

### *Observations on the Microstructure of the Rails*

In nearly all the cases the size of the pearlite grains shows that the rolling was discontinued at a temperature well above the critical range, but the results here obtained do not bear out the contention that the finishing temperature is equivalent, so far as grain size is concerned, to the highest temperature of objects heated but not rolled or forged. On the contrary, the grain size is smaller than would have been produced by heating the same steel to the temperature at which rolling was discontinued, and then allowing it to cool without mechanical work. It would appear probable then that the restoration of the crystalline arrangement, which is so badly shattered by the squeezing action of the rolls, is not so rapid as is generally supposed, and, although the restoring process begins immediately after the rolling ceases, it is not complete until the steel has cooled somewhat, which, in fact, produces an "effective finishing temperature" somewhat below the observed one.

From the data in the possession of these authors it would appear that the grain size of the steel is determined as much by the total amount of reduction in the rolls as it is by the finishing temperature, or any other factor.

For the determination of any numerical relation between the size of grain and the mechanical properties of the steel a study of the same steel under varying conditions is essential.

The effect of manganese in retarding the condition of equilibrium during cooling has an important effect on the structure and properties of steel.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Manganese Steel and the Allotropic Theory

BY ALBERT SAUVEUR, CAMBRIDGE, MASS.

(Pittsburg Meeting, October, 1914)

AT the New York meeting of the Institute, February, 1914, Professor Hopkinson and Sir Robert Hadfield presented an important paper entitled Research with Regard to the Non-Magnetic and Magnetic Conditions of Manganese Steel.[1]

Any contribution on manganese steel by its discoverer necessarily carries with it much weight and is entitled to serious and close consideration. The momentous discovery of that alloy by Sir Robert Hadfield some 30 years ago was the signal for great activity in the field of alloy steels; and the remarkable results obtained in the last two decades probably constitute the most important modern advance in the metallurgy of steel.

The authors of the paper referred to above are unable to reconcile with the allotropic theory the interesting results obtained by them in heat treating manganese steel. When they speak of the allotropic theory, however, I understand that they have more especially in mind the existence of beta iron. It is not my understanding that they question the occurrence of iron in the two allotropic varieties known as gamma and alpha. If I am wrong, I trust they will correct me.

A careful consideration of the experimental results reported and discussed by the authors leads me to believe that on the contrary they can be very satisfactorily explained in terms of allotropy and that, in the light of our present knowledge, they cannot be accounted for in any other way.

The physical properties of non-magnetic manganese steel described by the authors are those pertaining to iron-carbon alloys in which iron is present in the gamma condition, or, in other words, to austenite. The metal is then characterized by great tenacity and ductility and low elastic limit, while its hardness, mineralogically speaking, is not excessive, but of a special kind termed "tough hardness" by the authors. Even when subjected to the restraining influence of a large amount of manganese and of considerable carbon, however, relatively rapid cooling

[1] *Bulletin*, No. 87, pp. 513–530 (March, 1914).



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is necessary to produce austenitic manganese steel free from beta and alpha iron. On very slow cooling from a high temperature, as indicated by the authors, manganese steel becomes, to some extent, magnetic, because in accordance with the allotropic theory such slow cooling permits the formation of some beta and alpha iron. The mineralogical hardness of the metal is then, as it should be, somewhat greater. Non-magnetic, and, therefore, austenitic, manganese steel, while very much more stable than hardened, high-carbon steel, is nevertheless in an unstable condition, as evidenced by the fact that relatively quick cooling is required for its production. On reheating (shall we say tempering?) manganese steel, therefore, we may logically anticipate a partial transformation of gamma iron into beta and, possibly, into alpha iron—and, indeed, the authors show that reheating non-magnetic, austenitic manganese steel for a long period of time to some 500° C. or higher causes the return of considerable magnetism and greatly increased hardness. While they consider this occurrence fatal to the allotropic theory, it appears to me, on the contrary, to carry with it a striking confirmation of the correctness of that theory. If there were no hard allotropic modifications of iron *between* the soft alpha and moderately hard gamma varieties, the transformation taking place in reheating manganese steel could only imply the formation of some alpha iron and, therefore, *decreased* hardness. The fact that the treated steel becomes magnetic also points to the formation of some alpha, together with beta iron. This is also consistent with the allotropic transformation observed in carbon steel when we find that whenever a notable quantity of beta iron is formed, causing hardness, some alpha iron also forms, imparting magnetism to the metal; hence the hardness and magnetic properties of hardened carbon steel. It does not seem possible, in the presence of carbon at least, to produce much beta iron to the exclusion of alpha iron. These two allotropic varieties probably form isomorphous crystals which are both hard and magnetic.

The authors seem to have some difficulty in explaining, in accordance with the allotropic theory, the fact that on heat treating non-magnetic manganese steel, both the hardness and the specific magnetism may *increase*, their argument being that if magnetism is due to the formation of soft alpha iron, then increased magnetism must mean increased softness. This, however, is only an apparent inconsistency, as I have attempted to show graphically in Fig. 1. We may assume that at 400° C. originally non-magnetic manganese steel remains non-magnetic, the iron present being wholly in the gamma condition. After a long heating, say at 450° C., some of the gamma iron is converted into beta and into alpha iron. Of the iron present, $AB$ per cent. may be assumed to occur now as alpha iron, $BC$ per cent. as beta iron, and $CD$ per cent. as gamma iron. Heating to a higher temperature, or possibly a longer heating at the same temperature,

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causes further transformation, *both* beta and alpha iron *increasing*, the metal containing now $A'B'$ per cent. of alpha and $B'C'$ per cent. of beta iron; hence its increased magnetism and increased hardness.

Some beta and alpha iron having now been formed, it is evident that on heating the metal to a sufficiently high temperature, a critical point must necessarily be found, caused by the return of alpha and beta iron to the gamma condition—the only one stable at a high temperature. The authors have located this point at about 675° C., as shown in my Fig. 1. On again cooling the metal relatively quickly, no critical point should be detected, since the steel remains non-magnetic, that is, in the gamma condition.

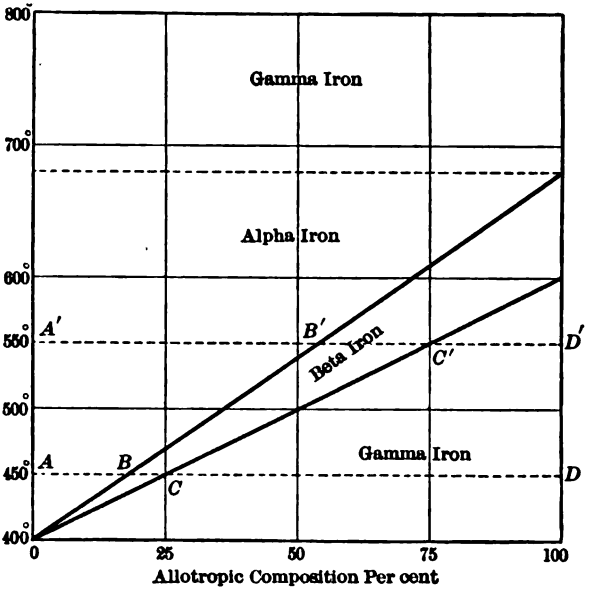


FIG. 1.

The fact alluded to by the authors that manganese steel can be so softened by suitable heat treatment as to make it machineable is another evidence of the formation of alpha iron.

From the fact that cast manganese steel is much less ductile than the water-toughened metal, although likewise non-magnetic, we not illogically infer that some beta iron may have been formed during the relatively slow cooling of the casting, while the usual lack of ductility of steel castings in general and the possible separation of carbide may be additional reasons for the properties of cast, untreated manganese steel.

The fact that manganese steel does not become magnetic on cooling it in liquid air, while non-magnetic nickel-steel with some 22 per cent. nickel regains its magnetism after such treatment, does not point to

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different influences being at work. It can be satisfactorily accounted for, as shown in Fig. 2, in which it is reasonably assumed that with 1 to 1.25 per cent. carbon it requires some 7 per cent. of manganese to lower the critical point from its normal position (some 700° C.) to atmospheric temperature, whereas some 18 per cent. nickel may be required to cause the same depression. Extending below atmospheric temperature the lines indicating the lowering of the critical points, it is seen that with some 12 per cent. manganese the critical transformation may be lowered to some −400° C., while nickel steel with 22 per cent. nickel would have a critical point at −100° C. and therefore detectable on immersion in liquid air. It will, of course, be understood that no claim is made here in regard to the exact position of the lines or curves indicating the

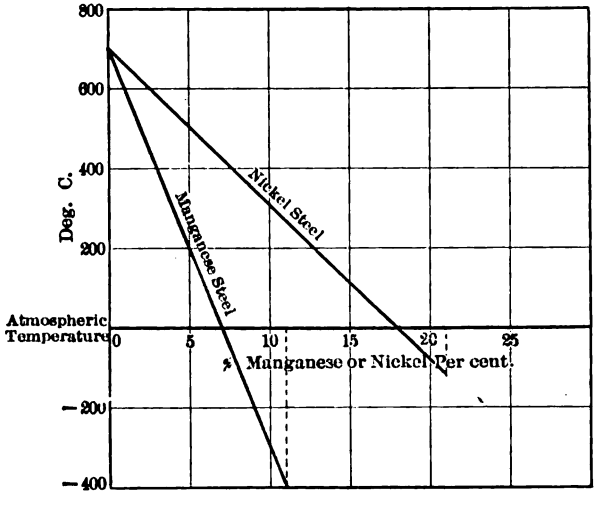


FIG. 2.

depression of the magnetic change with increased manganese or nickle. These positions, so far as I know, have never been accurately determined, but, from the results available, the diagram of Fig. 2 is not altogether a fantastic one.

The authors quote at length from the important work of Pierre Weiss, who has developed a new theory of magnetic transformation in metals.

The attitude of a scientific man toward new theories should be one of open-mindedness and receptiveness—even of reverence—when the new thought springs from an exalted source; but it should also be one of caution. He should guard against too hasty and enthusiastic acceptance. He should be mindful of the theoretical scrap heaps lining both sides of the pathway of scientific progress. For a while, at least, a new theory must be placed on probation, while being subjected to searching tests. It is the present position of Professor Weiss's theory.

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With a view to securing, if possible, additional information in regard to the relation between the heat treatment of manganese steel, and its critical points, structure and physical properties, some experiments were conducted in the Metallographical Laboratory of Harvard University which will be briefly described and illustrated.

### *Steel Used*

The manganese steel used in these experiments was supplied by the Taylor-Wharton Iron and Steel Co. of High Bridge, N. J. in the shape of cast bars $\frac{1}{2}$ by $\frac{3}{4}$ in. in cross section and about 12 in. long. The steel contained 1.25 per cent. carbon and about 12.50 per cent. manganese.

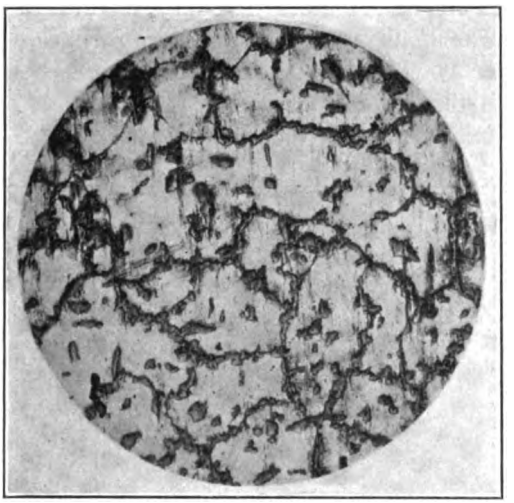

FIG. 3.—CAST MANGANESE STEEL. MAGNIFIED 100 DIAMETERS. CEMENTITO-AUSTENITIC STRUCTURE.

*Cast Untreated Manganese Steel.*—The structure of the cast metal is illustrated in Fig. 3. It consists of polyhedric grains of austenite surrounded by a considerable amount of a constituent generally held to be a double carbide of iron and manganese (manganiferous cementite), or a mixture (possibly a solid solution) of the carbide of iron $Fe_3C$ and the carbide of manganese, $Mn_3C$. As is well known manganese steel in this condition has a low tensile strength, low elastic limit and little ductility. It is non-magnetic. Its Brinell hardness number[2] was found to be 196. In the light of the allotropic theory the absence of ferromagnetism is to be

[2] The hardness tests mentioned in this paper were made at the Watertown Arsenal, Watertown, Mass. They will be found tabulated at the end of the paper.

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ascribed to the absence of alpha iron while the lack of ductility of the metal may be accounted for through the formation during the relatively slow cooling of the castings of some beta iron, although, undoubtedly, the presence of segregated carbide surrounding the austenite grains and the usual structural coarseness of cast steel may be also regarded as causes of brittleness.

### *Heat Treatments*

Some of the steel bars were subjected to the following treatments:

Heating to 1,100° C. and quenching in water.
Heating to 1,100° C. and cooling in furnace.
Heating to 1,100° C. and quenching in water followed by:
Heating to 800° for 2 hr. and cooling in furnace,
Heating to 700° for 2 hr. and cooling in furnace,
Heating to 600° for 2 hr. and cooling in furnace,
Heating to 500° for 2 hr. and cooling in furnace,
Heating to 400° for 2 hr. and cooling in furnace,
Heating to 575° for 90 hr. and cooling in furnace.

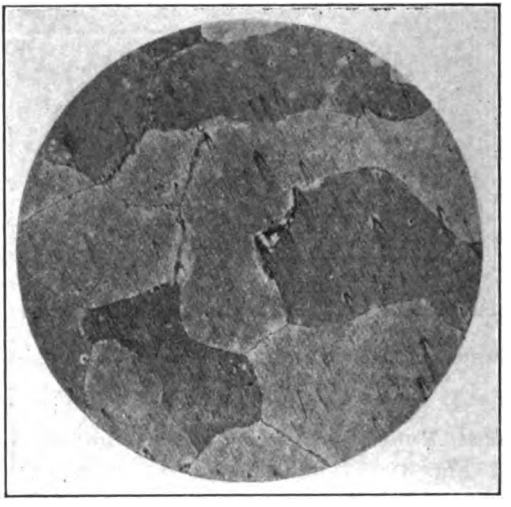

FIG. 4.—MANGANESE STEEL HEATED TO 1,100° C. AND QUENCHED IN WATER. MAGNIFIED 100 DIAMETERS. AUSTENITIC STRUCTURE.

Before subjecting the treated specimens to microscopical examination and to the Brinell test for hardness the outside, decarburized portion was removed by grinding.

*Bar Heated to* 1,100° *C. and Quenched in Water.*—The structure of this bar is illustrated in Fig. 4. The steel is now made up of polyhedric grains of austenite, quenching from a high temperature having caused the disappearance of the segregated carbide so conspicious in the struc-

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ture of the cast metal (Fig. 3). We naturally infer that at a high temperature this carbide or mixture of carbides dissolves in the austenite and that rapid cooling does not permit its re-precipitation. To the absence of this carbide must be ascribed in part at least the much greater toughness and tenacity of the quenched metal, although it may reasonably be claimed that the marked change of properties may also be due to the now complete absence of beta iron. As is always the case with austenitic steel, the elastic limit remains low. Two bars subjected to water quenching from 1,100° were tested for hardness giving respectively as factors

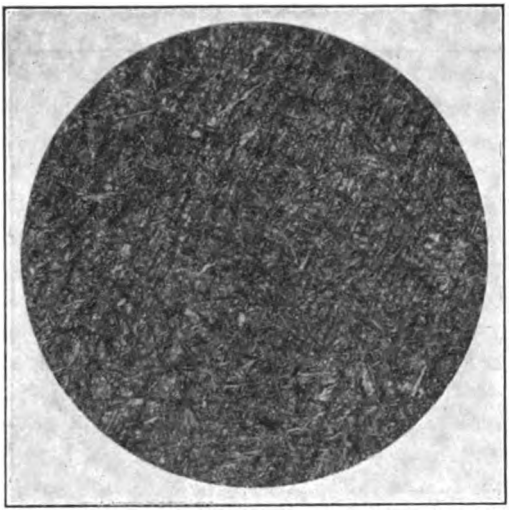

FIG. 5.—MANGANESE STEEL HEATED TO 1,100° C., QUENCHED IN WATER, REHEATED AT 700° FOR 2 HR. SLOWLY COOLED IN FURNACE. MAGNIFIED 100 DIAMETERS. AUSTENITO-MARTENSITIC STRUCTURE.

of hardness 174 and 181. This slightly decreased mineralogical hardness as compared to the hardness of the untreated metal (196) is in harmony with the conception of the formation of some beta iron during the relatively slow cooling of the cast bars and complete absence of it in the quenched bars. The quenched samples because of the absence of alpha iron are of course non-magnetic.

*Bar Heated to* 1,100° *and Slowly Cooled in the Furnace.*—The structure of this bar was similar to the structure of the cast untreated metal (Fig. 3). This was to be anticipated. Heating to 1,100° caused the segregated carbide or carbides to dissolve in the austenite but the very slow cooling that followed resulted in its being again precipitated. The metal was brittle and lacking in tenacity. Its hardness number was found to be 212, a slight increase over the hardness of the untreated metal (196). This may be explained on the ground that the furnace-

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cooled bar cooled more slowly than the cast bars resulting in the formation of a greater amount of beta iron. From the fact that this bar is non-magnetic we infer that although slow, the cooling was not slow enough to permit the formation of any appreciable quantity of alpha iron.

*Bar Heated to* 800° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar was quite similar to that of the cast, untreated bar, but from its greater hardness, which is now 245 as compared to 196, we infer that the long exposure to 800° followed by slow cooling made possible the formation of a greater amount of beta iron. The bar, however, is still non-magnetic which points to the non-formation of any appreciable quantity of alpha iron.

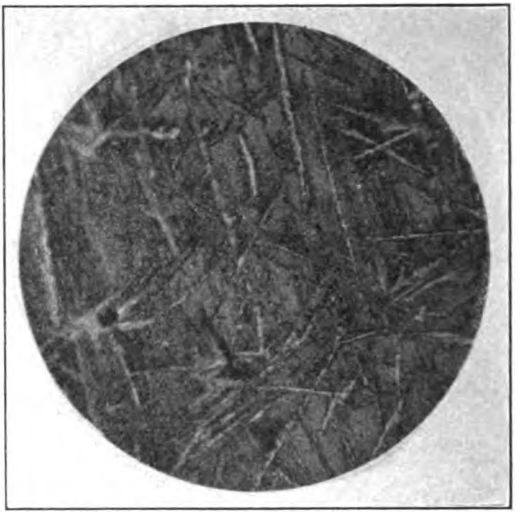

FIG. 6.—SAME AS FIG. 5. MAGNIFIED 500 DIAMETERS.

*Bar Heated to* 700° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar, illustrated in Figs. 5 and 6 under two different magnifications, clearly indicates the formation of martensite and hence of considerable beta iron. From our knowledge of the properties of martensite we shall expect the steel to be ferro-magnetic because of the usual occurrence of an appreciable amount of alpha iron in martensite. We shall also expect the steel to be now mineralogically harder than when in an austenitic condition. These expectations are fulfilled. Manganese steel so treated is very appreciably ferro-magnetic and its hardness is now 266, or 70 points higher than in the cast condition and some 90 points higher than after quenching from 1,100°.

*Bar Heated to* 600° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar was similar to that of the preceding bar, that is, decidedly martensitic while the bar is ferro-magnetic from which it

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follows that the heat treatment to which it was subjected resulted in the production of beta and alpha iron. Its hardness number, 335, indicates a very great increase of hardness. Evidently this bar must contain more beta iron than the bar heated for 2 hr. to 700°. This is readily explained on the ground that 700° is very near the range of temperature in which gamma iron is the stable condition.

*Bar Heated to* 500° *for* 2 *hr. and Slowly Cooled in the Furnace.*—The structure of this bar was also found to be martensitic while the metal was ferro-magnetic but to a decidedly less degree than the preceding bar. Clearly, at 500° tempering does not proceed as far as it does at 600 with the result that less beta iron and less alpha iron are formed. This is perfectly consistent with the allotropic theory. The hardness number of the bar was 255.

It seems evident that a temperature of 600° or thereabout is the most effective one to render manganese steel martensitic and, therefore, hard and ferro-magnetic.

*Bar Heated to* 400° *and Slowly Cooled in the Furnace.*—The structure of this bar was practically that of the untreated quenched metal. It was non-magnetic and its hardness number was 163. Clearly, heating to 400° for 2 hr. is not sufficient to cause any appreciable tempering of the quenched metal. There is no apparent explanation for the fact that the steel is now somewhat softer than in the quenched condition.

*Bar Heated to* 575° *for* 90 *hr. and Slowly Cooled in the Furnace.*—Seeing that magnetism is readily produced by heating in the vicinity of 600°, while the structure of the metal becomes decidedly martensitic, a quenched bar was kept for 90 hr. at a temperature of 575°. The structure of this bar was of the type illustrated in Figs. 5 and 6; its hardness was 356 and the metal was so magnetic that the bar could be readily picked up by a permanent magnet.

## *Conclusions*

From the above data it appears that manganese steel of the Hadfield type may occur under three distinct conditions: (1) as austenitic steel mixed with a considerable amount of free carbide, a condition which is produced by slow cooling from a temperature exceeding 700°; (2) as austenitic steel practically free from segregated carbides, a condition produced by rapid cooling from a high temperature and (3) as martensitic or possibly austenito-martensitic steel, a condition most readily produced by reheating austenitic steel for a sufficient length of time to a temperature exceeding 500° but not 700°.

In the first condition, which may be described as cementito-austenitic, the metal is non-magnetic, weak and lacking in ductility, while its hardness number is in the vicinity of 200. In its austenitic condition

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*Brinell Hardness Tests*

| Marked (Treatment) | Readings mm. | Average diameter mm. | Hardness Factor |
|---|---|---|---|
| Untreated unmarked..... | 4.30<br>4.30<br>4.30 | 4.30 | 196 |
| 1100W..... | 4.40<br>4.50<br>4.55 | 4.48 | 181 |
| 1100W duplicate..... | 4.55<br>4.55<br>4.55 | 4.55 | 174 |
| 1100F..... | 4.15<br>4.15<br>4.15 | 4.15 | 212 |
| 800–2–F..... | 3.90<br>3.85<br>3.85 | 3.87 | 245 |
| 700–2–F..... | 3.70<br>3.70<br>3.75 | 3.72 | 266 |
| 600–2–F..... | 3.35<br>3.30<br>3.35 | 3.33 | 335 |
| 500–2–F..... | 3.80<br>3.80<br>3.80 | 3.80 | 255 |
| 400–2–F..... | 4.70<br>4.70<br>4.70 | 4.70 | 163 |
| 575–90–F..... | 3.20<br>3.25<br>3.25 | 3.23 | 356 |

All tests were made under load of 3,000 Kg. with 10-mm. ball pressure held for 30 sec. 1100W signifies heated to 1,100° C. and quenched in water; 800-2-F signifies heated to 800° for 2 hr. and furnace-cooled, etc.

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the metal has a high tenacity, great ductility, is non-magnetic and its hardness number is in the vicinity of 180.[3]

The martensitic condition results from the tempering of austenite as previously explained. The metal is now ferro-magnetic because of the presence of some alpha iron, while its hardness varies between 250° and 350,[4] an increase due to the formation of a considerable amount of beta iron.

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[3] Sir Robert Hadfield mentions 200 as the average ball hardness of quenched manganese steel.

[4] In submitting quenched manganese steel to similar heat treatments, *i.e.*, prolonged heating between 500 and 600° C. followed by slow cooling, Sir Robert Hadfield increased its hardness to as much as 495.

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# Tin and Coal Deposits of the Fu Chuan District, China

BY M. B. YUNG, HONG KONG, CHINA

(Pittsburg Meeting, October, 1914)

## *Location*

THE Fu Chuan district lies near the eastern boundary of Kwang Si province in southern China and is connected with the coast by the West river and its branches.

## *The Journey*

The first part of the journey from the coast is made very comfortably in one of the West river steamers which runs to Wuchow, the great distributing center for the whole province. From here there are two ways of reaching the Fu Chuan district. One is to drop down the West river for 15 miles and ascend the Yuen river for 120 miles. The other is to ascend the Fu river, which joins the West river at Wuchow, and at Ping Lok take chairs across country. Of the two the former is the more direct and shorter but for other reasons I chose the latter.

At Wuchow one must provide his own means of transportation, which may be a native boat anywhere up to 40 ft. in length, but not over 18-in. draft.

The method of working these river boats against the swift current which in many places develops into rapids is interesting and worthy of note. Tracking along the bank with the tow line attached to the mast head is the most common procedure, but poling, sculling, sailing and winding up the rapids by a windlass are used as the case may require.

The boatmen are the hardiest set of fellows I have ever seen often putting in 15 hr. of the most exhausting labor. The average day's run against the current is 15 miles. On this trip one should have his own food, bedding and servants, and a guard of 10 or more soldiers is almost a necessity as certain parts of the river are infested with pirates.

The first 60 miles of the Fu river are all through sandstones, above here the metamorphic rocks come in, mica schists and quartzite predominating. In this belt of metamorphic rocks are several alluvial gold areas, both in

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the river itself and along the banks where the valley widens out, which are worthy of further investigation.

Surface samples from some of the big quartzite ridges gave only from 30 to 50c. per ton in gold.

In this belt of metamorphics, which is about 60 miles wide, the country becomes much more mountainous. Above this belt the river enters the most marvelous and beautiful scenery, caused by the erosion of a great limestone area leaving its typical topography of spires, domes and fantastic shapes which rise sheer from the clear waters of the stream. Ping Lok, our stopping place, lies in this limestone country. The journey from Hong Kong to Wuchow, a distance of about 300 miles, takes 2 days by steamer; from Wuchow to Ping Lok, a distance of about 150 miles, about 8 days by boat, and from Ping Lok to the mines, a distance of 80 miles, 3 days by chair. Going by the Yuen river from Wuchow the trip may be made in 8 or 9 days.

At Ping Lok the boat was abandoned and we started across country in sedan chairs, the luggage being carried by porters.

The country passed over the first day was all limestone and sandstone but the second day we began paralleling a range of granite mountains which at the end of another day's journey brought us to the tin district which lies along this granite.

### *The Nam Shan Granite*

At this point I beg to digress for a little to call attention to this granite and to make a tentative proposition with regard to it which I hope, when the opportunity offers, will invite further research.

Extending along the coast line of southwestern Asia from the Yangtze valley to the Malay Peninsula and for a distance of 200 miles inland is a great mountain system covering over 300,000 square miles.

Von Richtofen describes this system in one of his letters on China, giving it the general name of Nam Shan. Owing to its being so broken up and to its absence of any central or dominant range, the existing maps do not recognize it as a system.

Granite forms a very important part of the rocks which go to make up these mountains, especially along the coast of China, where it appears almost continuously in the white barren-looking hills and islands seen from the steamer's deck on the run from Shanghai to Hong Kong. Moreover, this granite does not end with this mountain system proper but appears in Japan projecting beyond its northwestern limit. The mountains along the inland sea of Japan are made up of this same granite, which with its white weathering and characteristic topography presents strikingly similar appearance wherever seen.

On the south, the granite continues far into the Malay Peninsula where it plays an important part in the tin deposits of the Malay States.

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Proceeding inland from the coast line of China the granite is generally covered by metamorphics and sedimentaries, being itself responsible for much of the metamorphism.

For several hundred miles inland, however, wherever erosion has cut deeply, inliers of granite are found, varying in size from a few acres up to many miles in area.

The following are a few common characteristics noted in the various areas visited.

1. The granite is always found underlying older bedded deposits which generally show contact metamorphism.

2. Regional metamorphism is also extensively developed in many places by the granite intrusion.

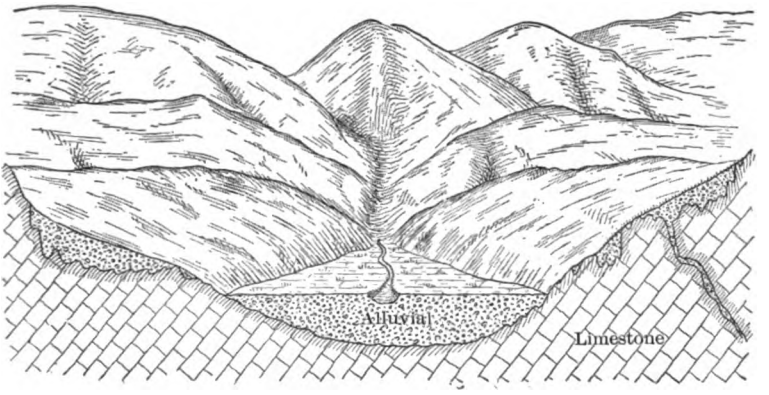


FIG. 1.—IDEAL SECTION OF VALLEY AT WANG KA.

3. Its composition and texture while varying somewhat is fairly constant in its manner of variation. When the granite is fresh and removed from any contact it has a coarse texture with large phenocrysts of potash feldspar associated with quartz and hornblende and with mica in subordinate quantities. In rare localities the quartz in this granite has apparently segregated out in large masses of many acres in extent. In different localities the amounts of mica vary.

4. Its appearance is white or mottled more or less speckled with the dark hornblende or mica.

5. Its topography is quite characteristic. In the case of the larger more prominent mountains, generally found in the interior, where erosion has kept the summits clear of laterite, a massive rounded or dome-like surface is formed with large boulders abundant which weather to a black surface. Where the hills are low and erosion has been unable to keep pace with decomposition, the surface crumbles into a coarse sand which scarcely supports vegetation and gives the characteristic white appearance seen

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along the coast. This latter phase is by far the most common. In many places one may look for miles over a homogeneous mass of little mountains or hills, which appear like a great collection of ant hills. In this material are found boulders buried in the laterite to a depth of 50 ft. or more, due to concentric weathering of which they are the core.

6. All the tin properties which I have seen or heard of in southeastern China either lie in or contiguous to this granite and I understand that this is generally the case in the Malay States also, so that this granite may be regarded as the ultimate source of the tin. Furthermore, a considerable percentage of the other mineral deposits within this vast area invaded by the granite owe their origin to the contact of the granite with the sedimentaries.

Upon these general characteristics and upon its geographical and geological distribution, I have felt justified in advancing the opinion that this granite has come from one parent magma.

Certainly its influence upon the geology, mineralogy and geography of this part of the world has been profound and it is therefore worthy of consideration and study.

### *The Fu Chuan Tin Deposits*

The tin deposits of this district so far discovered are all alluvial and lie scattered over a range of these granite mountains for a distance of more

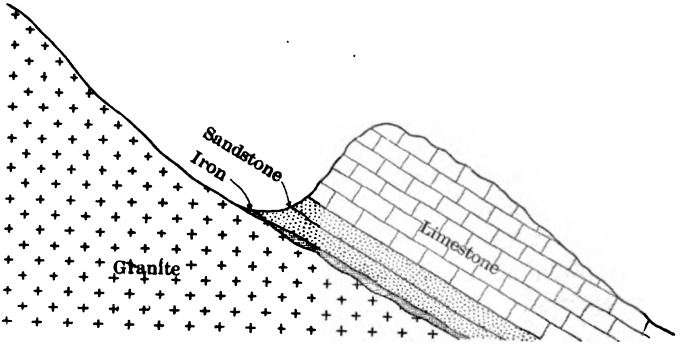


FIG. 2.—SECTION SHOWING TYPICAL CONTACT.

than 30 miles. The average latitude of this range is about 2,500 ft. It is flanked by limestone which forms foot hills and in many places along the contact has been changed to a pure white marble.

This contact of the granite with the limestone, with perhaps a little sandstone intervening, is very generally mineralized with iron existing as both oxide and sulphide. (See Fig. 2.) In one place there is a hill 500 yd. long by 100 yd. wide which is entirely covered with good iron ore in the form of hematite and magnetite.

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The tin, which is in the form of cassiterite, is found in the streams which come down from the hills, but most of it comes from the hillsides where it lies in the cuplike depressions of the limestone and can be easily gained without the removal of much overburden or interference from water. The natives dig it out of these pits, which in extreme cases are 30 ft. deep, but generally much less, and carry it in baskets to the nearest stream where it is washed in short sluices.

The deposition of the tin high up on these hillsides is evidence of the erosion that has taken place since the deposit of the tin, there often being a limestone summit between these pockets and the granite from which they came.

There are many level valleys among the foot hills ranging from 300 ft. in width up to a mile. Nothing is being done in these valleys, where one would expect the best ground to be, nor is anything known of their contents. This is due chiefly to the fact that the only parties working in this district are native miners, each working for himself, who depend upon their daily product of concentrate for their daily rice. It is entirely beyond the resources of these people to prospect or work these valley areas, because in the first place cultivated land would have to be bought, and drilling and dredging used to prospect and work the ground, as the ground water level is only a few feet below the surface in most cases. (See Fig. 1.)

The ground worked by the native yields cassiterite from 3 lb. per cubic yard up. It is free from clay and boulders and is ideal ground to wash.

No veins have yet been found, but I was given a piece of cassiterite weighing over ½ lb. which must have come from a vein. No prospecting has been done in the granite which is not recognized by the natives as a source of the tin, but they have devoted considerable time to looking for pipes in the limestone, which always furnishes them with rich ore, in fact a very considerable percentage of the tin comes from this class of deposit, which is worthy of description.

Caves are of frequent occurrence in the limestone; these vary from cracks up to caves a mile in length and of variable section from large rooms to spaces through which it is difficult to crawl.

The streams have carried the tin into these caves and deposited it, together with a soft porous lime salt in a more or less stratified deposit. These deposits, which contain from 3 to 5 per cent. cassiterite, are so solid that blasting is resorted to in mining. These deposits are much sought after by the natives and to their minds not yet mystified by the various theories of ore deposits answer all the requirements of ore in place.

In one of these caves about 60 men were working. To reach the working place took three-quarters of an hour crawling through passages where cold water and foul air divided the scant space and kept one in constant doubt whether the best chance of seeing daylight again lay in keeping on or turning back.



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### *The Government Monopoly*

Up to the present the industry has been conducted as a provincial government monopoly. The government has established stations at the various localities where tin is being washed, where the concentrate is bought from the natives, smelted and marketed. The government itself does no mining. Any person can start mining without any formality whatever but he is obliged to sell his product to the government station. The price paid is generally fair, varying from 360 to 450 cash per *cattie* which is equivalent to 12 to 16c. gold per pound of 70 per cent. concentrate. This would be a most unreliable figure to count on, however, should a company commence producing tin on a large scale.

Under the new mining laws of China which have just been issued (April, 1914) it is doubtful what will become of this monopoly, since no provision seems to have been made for it. It deserves to pass out of existence as it is undoubtedly holding back the proper development of this promising field.

### *Smelting*

The smelting is done by contract, the smelting contractor guaranteeing 64 catties of tin bullion out of every 100 catties of clean concentrate he receives, and his profit is what he can make over and above this amount. He is also the man who buys the ore from the miners, which serves as a remarkably clever check on the manager of the station.

The ore is bought by inspection, no assays being made, and a certain percentage is deducted for moisture as the ore is always wet when sold.

If the ore is not up to the required standard, it is washed by the smelting contractor, who returns the discard to the miners.

The smelting is done in a small native shaft furnace. The base of the furnace is a large iron pan and the walls are built up of brick and clay bound with iron. The blast is furnished by a hand bellows made from a hollow log (Fig. 3). Charcoal is used in the proportion of 1.3 to 1 of ore and a small charge is added every 10 min. The spout is left open and the blast is not sufficiently trapped so that considerable oxidation of tin takes place at this leak.

Bullion and slag run continuously into a forehearth which is simply a hole dug in the floor. No flux is used and very little slag is formed. It is skimmed from the forehearth and divided into two classes. The top skimmings, which comprise the greater part, are granulated, crushed and washed, and the concentrate, representing about one-tenth of the lot, is returned to the furnace.

The skimmings coming from next to the bullion are returned direct to the furnace. Thus the two sources of loss are the oxidation of the tin

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and the discard from washing the slag. The capacity is 700 catties of ore and 300 catties of slag in 12 hr.

The life of a furnace varies from one to three weeks and costs to rebuild about $10.

The tin bullion generally assays over 98 per cent. and is used by the refiners in Hong Kong to improve the grade of the Yunan tin. The yearly output is about 150 tons of tin bullion.

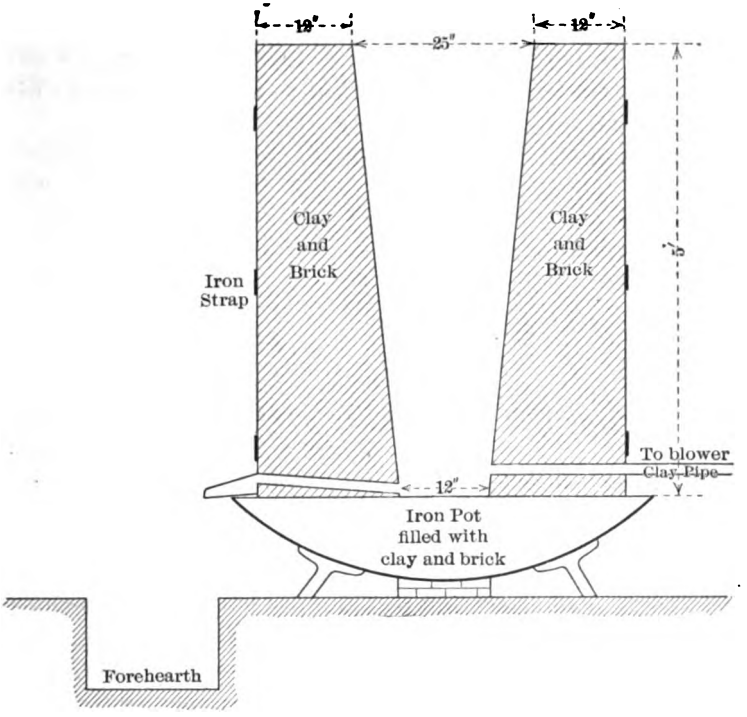


FIG. 3.—CHINESE FURNACE FOR SMELTING TIN.

Labor is abundant and of excellent quality, the wages averaging about 20c. per day.

The bullion is carried 10 to 20 miles over flat country, 120 miles in 3-ton boats down a shallow river to the West river where large steamers pass daily. The total cost of transportation from the mines to Hong Kong does not exceed $6.

Should the numerous valleys prove to be dredging ground this district might easily become a very important source of production.

### *Fu Chuan Coal*

Situated a few miles from the mountains in which the tin is found and right on the Yuen river a fine quality of coking coal is being mined also

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by the Kwang Si government. Three seams aggregating 24 ft. in thickness occur in a black shale bed not over 75 ft. thick, which in turn lies in limestone. Above the limestone lies a sandstone breccia which has been impregnated with siliceous and iron-bearing solutions resulting in the iron nodules so often seen about coal outcrops. The surface weathers into a red soil which may be taken as an indicator marking the extent of the coal basin. The coal is developed only to a vertical depth of 160 ft. and a length of 1,000 ft., but judging from various outcrops and from the geology of the surrounding country I should expect to find a coal basin several miles in extent.

The dip of the beds varies from vertical to almost horizontal with the folding of the strata.

The analysis of the coal is as follows:

| | Per Cent. |
|---|---|
| Moisture | 1.2 |
| Volatile matter | 18.5 |
| Carbon | 65.0 |
| Sulphur | 0.9 |
| Ash | 14.4 |
| Coke | 71.0 |

It burns easily with a long flame and makes an excellent quality of coke.

Transportation is the factor which renders this coal field of small importance at present. The same conditions of transportation exist for the coal as the tin.

To exploit this coal on a big scale a railroad would have to be built to the West river a distance of 120 to 130 miles but for the present a light railroad from the mine to Ho Yuen a distance of 12 miles would cut out the worst part of the river, and from Ho Yuen to the West river boats carrying 30 tons might be used if the narrow channel were deepened in a few places. On the West river barges of any size can be used. I believe this coal could be put into Canton or Hong Kong for a total cost of $3.50 per ton.

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# Investigations of Coal-Dust Explosions

BY GEORGE S. RICE, PITTSBURG, PA.

(Pittsburg Meeting, October, 1914)

THE subject of dust explosions in coal mines first appears in the *Transactions* of this Institute following the first great mine disaster that happened in bituminous mines of the United States. This was the Pocahontas explosion, which occurred Mar. 13, 1884, causing the loss of 112 men. The operating company wisely asked the technical assistance of the Institute, requesting that a committee be appointed to investigate and report. This committee consisted of Messrs. Bramwell, Buck, and Williams, who, after careful study, reached the conclusion that the explosion was due "either to dust alone or to dust quickened by an admixture of fire-damp too slight for detection by ordinary means." This is perhaps the first time in this country that coal dust was considered to be the chief agency in a mine explosion. The report is given in the *Transactions*, vol. xiii, p. 237 (1885). It is particularly worthy of note that the committee called attention to "the necessity of full-sized tests and systematic experiments."

In the same volume of the *Transactions*, there is a most interesting paper reviewing the literature relating to coal dust, by E. S. Hutchinson, entitled Notes on Coal-Dust in Colliery Explosions. Mr. Hutchinson gives a historical account of observations on the part played by coal dust in mine explosions, beginning with Messrs. Lyell and Faraday's report on the Haswell colliery explosion in 1844, in which it is said "much coal gas had been made from this dust in the very air itself in the mine by the flame of the firedamp, which raised and swept it along, and much of the carbon of the dust remained unburned only from want of air."

The subject lay dormant until 1875 when William Galloway began some coal-dust experiments in a small gallery at a mine in South Wales of which he was manager. At that time he considered the presence of firedamp necessary for continued propagation of a dust explosion; but in 1884, the year of the Pocahontas disaster, he stated that no earlier author than himself had credited coal dust with being a principal factor in mine explosions, relegating firedamp to a secondary place.

Meantime in 1876, Henry Hall and George Clark made some coal-dust



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explosion experiments in an adit 45 yd. in length, which were the first large-scale tests undertaken with coal dust. The explosions traveled the length of the adit, and issued with violence from the mouth.

In 1878 Morison and Marreco began experiments in diminutive galleries, followed in 1880 by similar experiments conducted by Professor (afterward Sir) Frederick Abel, with a view to determining the cause of the explosion disaster that occurred at the Seaham colliery. Professor Able drew the conclusion that as far as could be determined by experiments on a moderate scale the tendency of coal dust by itself to propagate flame is of a limited nature, and it required the presence of a very small percentage of firedamp; although he admitted that in the complete absence of firedamp, an explosion might be propagated to a greater distance than the results of small experiments would warrant one in assuming.

Mr. Hutchinson, in his paper, summarizes the opinions of the inspectors and prominent mining men before a Royal Commission sitting in 1879 and 1880. All those whose testimony he cited "appeared to be in accord that while dust in itself is not explosive, it is abundantly able to intensify an explosion initiated by gas." He then cites at length a series of 211 explosion tests conducted by the Chesterfield and Derbyshire Institute of Engineers, under the direction of Morison and Marreco, in a wooden tube 82 ft. long and 14 by 18 in. in section on the inside. It is interesting to note that in only a small proportion of the tests was ignition obtained and apparently in no case did propagation extend throughout the gallery, so that the conclusions of the committee were unanimous that "only *ignition* of dust (in distinction to an explosion) occurred *without* gas," and they added: "special circumstances must combine to insure ignition of dust *without* gas."

At the conclusion of Mr. Hutchinson's paper he says: "Neither assent nor dissent with regard to the views that have been quoted is intended to be expressed in this paper. But the facts, nevertheless, remain that there are dusts and *dusts*, and that the ratio of the number of serious mine explosions to the total number of blown-out or overcharged shots and local explosions of fire-damp, in dry and dusty mines, is a small one; whereas, if the coal-dust theory be true to the extent insisted upon by its most strenuous advocates, every such local occurrence, in a mine of this description, should be followed by the more or less complete wreck of the colliery." This indicates the general point of view in 1884.

The reason for the many failures to obtain explosions of coal dust in the earlier experimental galleries was perhaps due to one or more of these causes:

1. Degree of impurity of the coal dust used.
2. Large size of the particles of dust.

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3. Smallness of igniting source, that is, the flame too short in length or in duration, or, the mechanical energy insufficient to raise the dust into a cloud.

4. Taking into account the small size of the igniting charge, it is possible there was unsuitable placing of the dust, or an insufficient quantity of the dust at the point of ignition.

The next discussion of coal dust in mine explosions, which appeared in the *Transactions*, in vol. xxiv (1894), followed a paper by William Glenn on Mine Explosions Generated by Grahamite Dust, in which he evidently considers in the two explosions in 1871 and 1873 in the Ritchie mine (West Virginia) that the dust was the cause and not firedamp. In the discussion Dr. R. W. Raymond quoted at length from the report of the Prussian Fire Damp Commission which, in 1884, began a series of experiments with coal dust, and coal dust and gas, in a gallery 167 ft. long, elliptical in cross-section, the diameters of which were 4 ft. (horizontally) and 5½ ft. (vertically). As was stated by Dr. Raymond, these experiments were more systematic and thorough than any previously conducted.

The general conclusion of the Commission was that when firedamp is absent the propagation of an explosion of coal dust is limited for most varieties of dusts, but there are coal dusts which once ignited by a shot, burn, or spontaneously give flame far beyond the localities strewn with the dust.

In Austria in 1885 and 1886 coal-dust experiments were conducted in a testing gallery. In these experiments nearly all kinds of coal dust were ignited by a cartridge of 100 g. of loose dynamite, but it was stated that a small mixture of firedamp notably increased the sensitivenesss of coal dust to ignition. Further, that the fineness of the dust, as well as its dryness, greatly increases its sensitiveness and danger.

In 1886 W. N. and J. B. Atkinson, mine inspectors of England, issued an interesting book entitled *Explosions in Mines* which was the result of their studies of many mine explosions. The authors state very positively that they consider coal dust the chief factor in most of the British mine explosions of the past, and describe in detail six explosions, in five of which they consider coal dust was the chief, if not the only, agency. Their discussion of the subject shows remarkable knowledge of the new problem. They considered the most dangerous dust is the finest coal dust, or "float dust," which they term "upper dust" from its being found on the timbers and on projections of the roof and sides. In the light of the definite information obtained in recent large-scale experiments the authors possessed a remarkable intuition of the mechanics of explosions, the circumstances under which coal-dust explosions will propagate, the position of the coked dust with reference to the movement of the explosion, and many other details, which shows that their views were well in advance of the times. It is particularly interesting to note that they concluded

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"that the direction that the air was traveling had no influence on the course of an explosion after it was once started."

In 1890 to 1893, Henry Hall, British Inspector of Mines, conducted some explosion tests in disused shafts, which led him to state that "his experiments conclusively prove that a blast of gun powder in dry and dusty mines may cause serious disaster *in the entire absence of firedamp.*"

The British Royal Commission, in 1894, in its summary states: "Coal dust alone without the presence of any gas at all may cause a dangerous explosion if ignited by a blown-out shot or violent inflammation. To produce such a result, however, the conditions must be exceptional, and are only likely to be produced on rare occasions.".

The matter rested for many years following this report, not only in this country but abroad, except for occasional discussions. Nothing on the subject of coal dust in relation to mine explosions appears in the *Transactions* between 1895 and 1908, when there was a paper presented by Carl Scholz, entitled Effect of Humidity on Mine Explosions, which brought forward the danger of dry coal dust in mines, and discussed the advantages of the sprinkler-humidifying method of preventing mine explosions (vol. xxxix, p. 328 (1908)). This was followed by a series of papers on these and allied matters by Bache, Shurick, Mannakee, Raymond, and Rice, appearing in vols. xl and xli. Meantime, however, it had become apparent that long-flame explosives were dangerous to use in dusty and gaseous mines, and galleries were established in Germany in 1894 and subsequently in Belgium, Austria, and England for testing so-called "safety explosives," which through giving a shorter flame, one of less temperature, and of far less duration, would be relatively safer than black powder or dynamite. This led to the establishment of "permitted lists" of explosives in Belgium and Great Britain, and specifications of requirements in France, Germany, and Austria.

It was not until after the great Courréires disaster in 1906, which cost the lives of 1,100 men, in a colliery which had always been so remarkably free from firedamp up to that time that it was considered non-gaseous, that the fear of coal dust was reawakened in the great coal-mining countries. It led in France to the establishment by the Central Committee of Operators, of the Lievin station, which was placed under the talented government engineer, J. Taffanel. Prior to this French mining men had been led to believe, by early laboratory tests, that firedamp was the chief danger, and that coal dust would only increase the intensity of an explosion, and therefore if ignition of firedamp was prevented there would be little danger from coal dust itself.

In Great Britain the Coal Operators Association, early in 1908, erected the large Altofts gallery, which was placed under the direction of Mr. (later Sir) William E. Garforth, who had repeatedly called attention to the danger of coal dust, and who believed, as a result of his observations following the

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disaster at the Altofts collieries in 1886, that rock dust, if present in sufficient quantities, rendered coal dust non-inflammable. The important work at Altofts continued for three years, and experiments were conducted on a larger scale than had yet been tried, the gallery having a total length of about 900 ft., the explosion section proper being 600 ft. long and 7½ ft. in diameter. Much new and valuable data were obtained from these experiments regarding the propagation of dust explosions, the force developed within the limits of the gallery, the gases formed and the effectiveness of the rock-dust treatment method of prevention.[1] In 1911 this work was transferred by the Mine Owners Association to the British government and the gallery was removed to Eskmeals, where tests have since been conducted under the direction of a Royal Commission and five brief reports issued.

The Lievin station in the Pas de Calais district of France, when first established in 1907 employed a small gallery for preliminary tests, but a larger gallery was installed in 1908, and successively increased in length until the scale on which the tests were conducted was more extensive than in the British experiments, the main gallery being 1,000 ft. in length. Mr. Taffanel has advanced the knowledge concerning coal-dust explosions to an extent which places the work on a more exact scientific basis, particularly as regards the explosion wave movements under different conditions. The acquirement of information has, in part, been due to the high development of the recording instruments he employs.[2]

As already stated, explosives testing galleries were established 20 years ago in Germany and subsequently in Belgium and Austria. In these galleries tests of explosives are made in the presence of gas and dust, and some dust-explosion experiments have been conducted, but the galleries are too short to enable conclusions to be drawn regarding the wave movements of explosions except to a limited extent in the underground gallery at Rossitz, Austria. The new Westphalian surface gallery near Dortmund, Germany, now 100 m. long, it is intended will be enlarged and in the future experiments will doubtless be conducted of a size and thoroughness comparable with those being carried on at Eskmeals and Lievin.

### *Coal-Dust Experiments Conducted by the Bureau of Mines*

The coal-mining industry of the United States, as compared with that industry in European countries, was very backward in taking up experimental investigations relating to coal dust, although mine explosions attributed to coal dust had happened with sad frequency. Just why more active interest was not taken in experimental investigations is not ap-

[1] See Record of the First Series of the British Coal-Dust Experiments, 1910.

[2] See series of publications issued by the Comité Central des Houillères de France, 1909 to date.

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parent, because the magnitude of the industry in this country fully warranted such investigations. Neither the individual operators or associations nor States took the matter up, although Pennsylvania alone mines more coal annually than any foreign country except Great Britain and Germany. Finally, however, in 1907, such a series of disastrous mine explosions occurred that the country was aroused, and Congress appropriated money for the investigation of mine explosions and other mine accidents, which subsequently led to the formation of the Bureau of Mines. Thus the first, and so far, the only large-scale coal-dust experiments conducted in this country were started by the Federal government at the Pittsburg station, established in the fall of 1908. During the first year the Pittsburg gallery, a 100-ft. steel tube, 6½ ft. in diameter, was devoted to the testing of the so-called "safety explosives" in the presence of gas and dust, to establish a permissible list of those that passed the tests; but in the following year a preliminary series of experiments was conducted in the gallery to obtain information concerning the explosibility of coal dust, and how the danger might be neutralized. Information on the following points was sought:

1. To determine how small a quantity of coal dust will propagate an explosion.

2. To find the coarsest sizes of dust that will propagate an explosion.

3. To determine how much moisture coal dust must contain to prevent ignition and propagation.

4. To observe the effect of foreign inert matter or ash upon ignition and propagation of an explosion.

These tests were largely conducted with dust ground from coal of the Pittsburg bed, which is considered to produce dust of the most inflammable character. It was found that, when very fine (through 200 mesh), as small a quantity of dust as 32 g. per cubic meter of space (0.032 oz. per cubic foot) would ignite and propagate, at least within the short limits of the explosive zone employed. (See *Bulletin No. 20, U. S. Bureau of Mines*, Explosibility of Dust.)

As regards the second inquiry, it was found that with coal screened through a 20-mesh screen and over 40 mesh, ignition and partial propagation would ensue, and as it had been found that with every increase in the igniting charge, up to the largest amount that could be put in the cannon, there was an increasing ease of ignition, it seemed probable that a larger charge of powder in the initiating shot would cause complete propagation of dust between 20 and 40 mesh. It was therefore tentatively decided to accept 20 mesh as the dividing line between small-sized coal and what might be considered dust. While this limit is empiric, it is found to be a good division in practice, and has since been adhered to in gathering samples of road dusts; if larger sizes are gathered, that part over 20 mesh is rejected.

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In the third inquiry, as to the effect of humidity and moisture, it was found that the humidity of the atmosphere itself had no appreciable effect upon an explosion of inflammable dust, but when the moisture content of the dust was such as to prevent its being thrown in suspension, then no explosion resulted. These features have since received abundant verification by numerous tests at the experimental mine.

As regards the fourth principal inquiry, only a few tests were conducted to determine the effect of the ash or admixture of coal dust with foreign material. These tests indicated that the presence of ash up to 40 per cent. of the dust of the kinds experimented with did not prevent propagation; that in one instance 50 per cent. did not prevent, but in another case did prevent propagation.

Concerning the bearing of the carbon-hydrogen ratio in ignition and propagation, systematic tests like those conducted in France by Mr. Taffanel were not made on account of insufficient facilities, but those that were made at the station, and subsequently at the experimental mine, confirm his findings rather than those of the English investigators, that the ratio is of great importance; that is, with other things equal (ash, moisture, and size of dust) the higher the percentage of volatile *combustible* matter the more easy is it to obtain ignition, and also propagation, though this does not mean that the violence of an explosion once started is greater with a dust of high volatile content. It means that the flame of the source of ignition must be hotter or of longer duration with dusts with low volatile content.

A decrease in volatile matter, speaking broadly, decreases the range of explosibility, into which enter the factors of ash, moisture, size of dust particles, structure of particles. A coal dust in which the fixed carbon is more than four times the volatile combustible is harder to ignite by a blown-out shot, or an electric arc, than one in which the ratio is 2 to 1 or 3 to 1. Anthracite that has a ratio of 10 to 1 or over has so far not proved to propagate an explosion even with a small percentage of firedamp present, but semi-anthracite in which the ratio is less than 10 to 1 has shown propagation with 2 per cent. of firedamp present, but not without it.

Welsh anthracite dust is considered by the British investigators as explosive, and they cite experiments in which charcoal dust proved explosive, to show that there is danger from anthracite dust without regard to the volatile matter present.

It was found, as had been anticipated, that the gallery was too short, being only 100 ft. long, to carry out explosion experiments which could be considered at all conclusive as regards propagation. From the beginning it had been expected that large-scale experiments would be carried on in the underground entries or headings of a mine as soon as one could be secured. Accordingly, the following year, 1910, plans were definitely

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made looking to securing such a mine. No old mine suitable for the purpose was found, and it was finally decided to open up a mine by stages, primarily for the purpose of coal-dust explosion tests. The mine was opened when the state of the allotments permitted, and was prepared for the preliminary test at the end of 1911.

Fifteen explosion tests of a preliminary nature were conducted[3] in 1911 and 1912. The work was then suspended on account of the funds being exhausted. The mine was enlarged the early part of the fiscal year 1912–1913 and a few experiments conducted during the sessions of the International Mine Experiment Station Conference, held at Pittsburg, in 1912. For the fiscal year 1913–14 funds were allotted which permitted carrying on a most important series of over 100 mine-explosion experiments, the results of which are now being tabulated and prepared for publication.

The mine consists of two parallel entries (see Fig. 1), driven into the Pittsburg coal bed from the outcrop. These are now 1,300 ft. in length. The right entry of the pair is called the "main entry" and the left entry is called the "air course." There is a stub 100 ft. in length, turned at right angles off the main entry at a point 775 ft. from the mouth; and a pair of butt entries turned at right angles off the air course, the first of which is 850 ft. from the mouth. During the recent tests, or what is termed the "second series of tests," these entries were 200 and 250 ft. respectively in length, but each of them has been extended 100 ft. during this past summer. Two rooms had been turned at right angles off the first butt entry during the second series of tests, and now two more rooms have been turned.

At 150 ft. from the mouth of the air course a diagonal entry comes in from the outcrop, and at the outer end of same there is a 120-ft. steel gallery, which by several offsets is connected to the fan. The outer 200 ft. of the main entry, air course, and slant are lined with reinforced concrete, but the early explosion experiments badly ruptured and broke up this lining and lifted the roof over the entries for 150 ft. from the respective mouths. The inner parts of the entries are like ordinary entries in the Pittsburg bed, 9 ft. wide and from 6 to 7½ ft. high. The coal bed itself varies from 5 to 5½ ft. in thickness, over which there is a draw slate from a few inches to 1½ ft. thick, and above this a top coal, which so far has furnished a very good roof, although somewhat skaken in places. Fortunately, so far, very little timbering has been required.

In the main entry, and in the inner part of the air course, large instrument stations have been made every 200 ft. and intermediate to these small flame and pressure circuit-breaker stations. The main instrument

[3] See *Bulletin No. 56, U. S. Bureau of Mines*, First Series of Tests at Experimental Mine.

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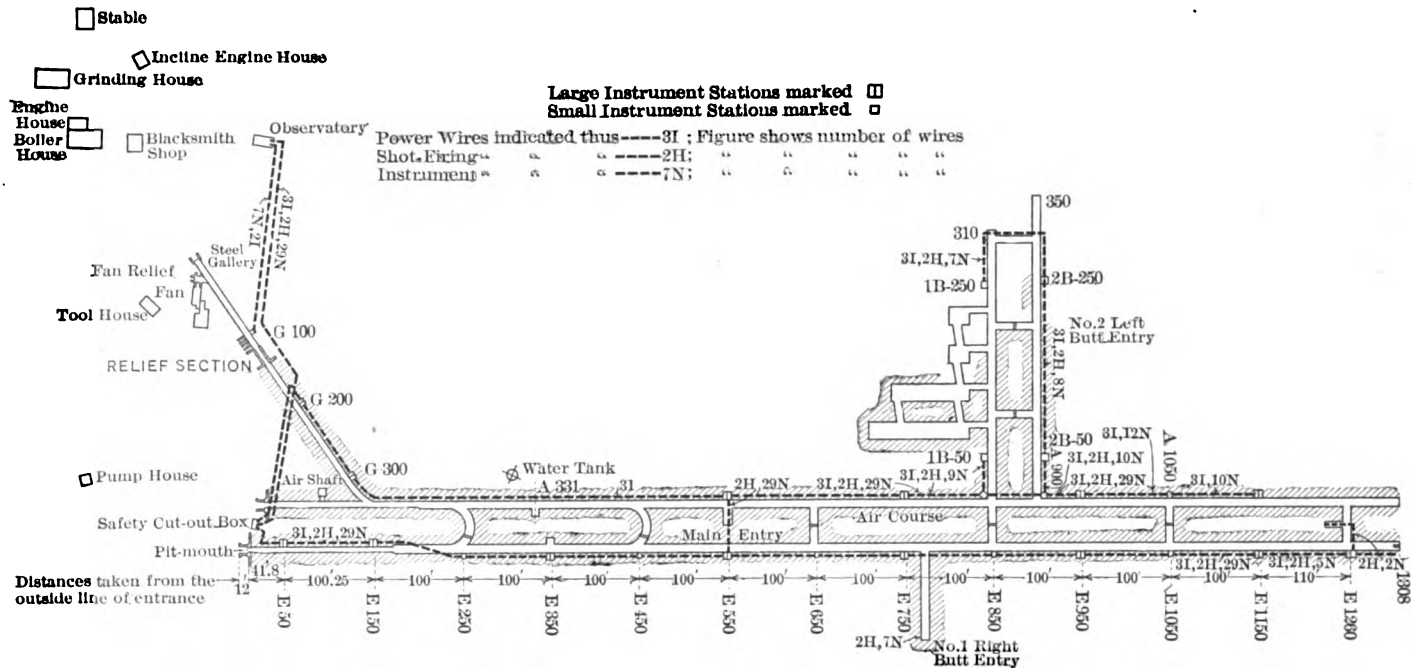


The unshaded portions of Nos. 1 and 2 Butt Entries show extensions since the second series of tests were concluded in July, 1914.

FIG. 1.—DIAGRAM OF EXPERIMENTAL MINE, SHOWING OBSERVATORY, STATIONS, AND CABLES.

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stations are chambers constructed in the rib and lined with concrete, with heavy steel plates shutting off the entry, through which the instruments are connected with the mine atmosphere. Doors are placed to one side, permitting entrance in behind the instruments which are placed upon shelves. The chief instruments, are the pressure manometers electrically rotated for recording the varying pressures, as the explosion passes. Pressure circuit breakers and flame circuit breakers for interrupting electric circuits are placed in the main instrument stations, and also in the small intermediate stations. The stations are connected by wires formed into cables, which pass through pipes placed with compressed air and water pipes in a horizontal groove in the rib. This is faced up with concrete for protection of the cable and pipes. The wires go to the observatory on the surface, where the recording instruments are located. It is from this observatory that the explosions are started by firing through an electric circuit blown-out shots of black powder from a cannon. Premature firing is safeguarded by cut-out switches at the mouth of the mine and at the observatory.

The mine is equipped with six pressure manometers, three being like those used in the British experiments, and three like those employed in the French tests. These may be moved from station to station, in accordance with how the tests are arranged. In the British manometers, the pressure is transmitted through a bath of oil to a plunger operating against a triangular spring which carries the scribing style and that traces the pressure curve on a band of smoked paper, wrapped around a revolving cylinder. In the French manometers, the pressure acts directly on a small plunger which presses on a small deflecting mirror. A beam of light from an incandescent lamp within the apparatus is reflected on to a photographic paper around a revolving drum. The movement of plunger and mirror is so small that the inertia is negligible. For both kinds of manometers there are electrical devices and timing arrangements for marking the time on the pressure records with reference to the igniting shot.

Some modification, of these instruments have been made, and it is expected that others, embracing new features, will be constructed at the Bureau's instrument shop.

Connected with the mine there is a grinding plant for making coal dust, very large amounts of which must be used in some tests. This also serves for grinding shale dust or rock dust for mixing with the coal dust in determining the explosiveness of mixtures of different percentages of ash and for use in explosion-checking devices.

The fan is relatively large for a mine of this size. It will either force or exhaust about 50,000 cu. ft. of air per minute, and is quickly reversible. The offsets in the air passages, with a number of explosion doors, permit running the fan either way desired while an explosion is taking place.

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A natural-gas pipe line passes near the mine, and a connection has been made to same so as to permit the introduction of gas into the mine.

The mine itself is in what might be termed a "peninsula" of the coal bed, being surrounded by the coal outcrop, except on one side parallel with the main entry. Firedamp is not found to an appreciable extent. Many analyses of samples taken even when there had been no ventilation for some time have not shown more than a few hundredths of 1 per cent. methane and, with a moving current,none, or occasionally a trace, can be detected by analysis.

### *Method of Carrying on Coal-Dust Explosion Tests*

*Loading of Coal Dust.*—The coal dust for the tests cannot be placed on the floor of the entry, as is the practice in European galleries, because the floor is frequently wet, and, as is the case in commercial mines, the track road bed is more or less rough. In all the tests except recently the coal dust was placed on side shelves of 3 by 4 timbers, fastened by bolts to posts recessed into the ribs, or, in the concreted section, bolted directly to the concrete. There are five shelves on each side of the passageway. It was thought that the shelves being parallel with the path of the explosion, and being strongly bolted to the posts, which in turn are fastened by heavy iron bands into the rib, would not be damaged by the explosions. This, however, has not proved to be the case when the explosion is of a violent nature. This method of loading on the side shelves proved entirely successful when pure coal dust was used, and the mine was dry; but when the walls were wet, also in mixed-dust tests, all of the dust was not brought into suspension, and inconsistencies were found. Accordingly a system of cross shelves, placed close to the roof, was employed to supplement the side shelves. Such cross shelves correspond to the collars of timber sets in a commercial mine. It was found that this made a great change in the results of tests where the dust was near the border line of explosibility, and enabled the obtaining of consistent results, regardless of whether the walls and floor were wet.

*Method of Ignition.*—Explosions are initiated by blown-out shots, as is the usual method abroad. It is much the easiest method, as compared with employing a body of firedamp. In the earlier demonstrations the shots for blowing out were placed in the face and cased with pipe in order to insure the blowing out. This method, while successful, takes more time to arrange, and also, as some energy is expended on the walls of the drill hole, the pipe usually being burst, more consistent results were obtained by the use of a cannon with a bore hole of about 2¾ in., and 20 in. deep. Black powder was used in almost all cases, the charge usually being 4 lb., stemmed with clay. Circuit-breaker wires are placed across the mouth of the drill hole, so that the moment of firing may be recorded. As

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already stated, the shots are fired electrically from the observatory, and switches at the mouth of the mine cut out all wires entering the mine until every one has left the mine. The mining engineer in charge and the mine foreman connect up the shot-firing wires to the detonator leads after everyone else is out and then come out themselves.

*Point of Ignition.*—In the majority of tests the ignition shot was placed at the face of the main entry, pointing outward, parallel to the axis, the coal-dust loading, either pure or mixed, extending outbye from the shot in whatever arrangement may have been planned. The cannon is placed either on a mine truck, or latterly on the floor of the gallery, and a platform of boards, nailed to the ties, extends outward from the cannon. On this, 25 lb. of dust is scattered.

In a few cases two cannons were used, and a cross fire obtained, but the shock waves from the two shots complicated some of the manometric records so that a single shot was considered preferable. In some of the later tests the ignition point was in a short stub off the last crosscut midway in the pillar between the two entries. This was in order to secure double tests, using both the air course and entry. These entries furnish symmetrical conditions. In two tests the ignition shot was located at the head of the first left stub, and in a few other cases there was special placing of the ignition shot.

Necessarily many tests were made to standardize the method of ignition, the method of loading, and the instruments. The first objects were to find out how coal dust was ignited, and how the explosion was propagated from point to point; the pressure waves set up by the explosion; the movements of the pressure waves; the relation of the rise and fall of pressure at different points in the course of an explosion; the position of the flame with reference to the pressure waves; the velocity with which the pressure waves and the flame travel under different conditions, and the character of the combustion as far as that could be determined by the gathering of gas samples automatically at different stages of the explosion.

The second object was the determination of the effect of mixtures of shale dust or limestone dust on the ignition and upon the propagation. In this connection, the term "ignition" means the prolongation 50 to 100 ft. or more of flame originated by the shot; the term "propagation" refers to continuation of flame throughout the mixture or dust being tried.

At this point it may be well to explain, as far as our present knowledge goes, the way in which an explosion starts and propagates. When the igniting shot of black powder is fired, a flame is projected from the hole for 10 to 15 ft. the distance depending on the size of the charge and size of the grains of powder. This starts a shock wave, which travels at the rate of a sound wave. Tests in the mine have determined that this travels in the passageways at an average speed of 1,150 ft. per second. The effect on the sensitive French manometer is to cause a sharp rise

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from the zero line to a point where a pressure of 3 to 4 lb. is indicated. This line is jagged, showing vibratory movements of the wave. The shock-wave pressure decreases in intensity, though not in velocity, as it progresses, and is barely discernible on the pressure instruments (which measure the side pressure only) located 500 to 600 ft. from the cannon. A portion of the dust on the floor platform is raised by the blast, and is ignited by the flame of the explosive. This burns rapidly, and other dust brought from the side shelves by the eddying of the air currents is carried into the zone of combustion. As far as the manometers show, there is an interval of $\frac{1}{2}$ sec., more or less, before the pressure produced by the burning dust becomes appreciable. If the zone of dust loading is short, or if there is a high proportion of ash so that the mixture is barely explosive, the initial pressure near the origin will not be more than a pound or two, but with pure dry dust the pressure will rise rapidly. The flame from the burning dust takes from half a second to a second, depending on the conditions, to traverse the first 100 ft. In a pure coal-dust explosion, the second hundred feet is traversed more rapidly, that is from $\frac{1}{2}$ to $\frac{1}{4}$ sec. or less, and from this point onward the velocity rapidly increases until velocities of 2,000 ft. per second and upward are attained. As the explosion progresses in an explosive dust cloud, the pressure rapidly rises, and at a point 350 ft. from the origin pressures of 63 lb. per square inch have been attained, at 550 ft. from the origin pressures of 73 lb., and at 750 ft. 119 lb. per square inch have been reached. Where an explosion is abruptly checked by means of barriers it is found that the maximum pressure is attained at the moment of passage of the flame, but there is a pressure wave in advance of the flame. This is the unfortunate feature which permits propagation of a dust explosion, inasmuch as the dust is raised by this pressure wave. The English term it the "pioneering wave."

When the coal dust is uniformly loaded, and is of a highly explosive character, the pressure from point to point keeps on rising, as far as can be determined within the length of the passage of the experimental mine; as this increasing pressure wave advances, reflex waves are thrown off which go back toward the origin through the column of moving gases. In other words, at the area of high pressure, waves go in both directions to the areas of low pressure. This explains many of the anomalous features which have puzzled investigators of disastrous mine explosions. That is, evidence of violent movements in the direction opposite to that of the explosion are sometimes observed. At the experimental mine large wooden blocks, marked for identification, have been placed at regular intervals, and in an explosion of increasing intensity it is found that up to the point where the maximum pressure has been reached the blocks have gone toward the origin, and beyond this point, outbye or in the direction of the advance of the explosion.

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The same phenomenon has been demonstrated in some tests by placing loaded mine cars at intervals along the passageway; these have been moved toward the origin in the instances where the maximum pressure was attained beyond, although doubtless they first moved outward as the explosion wave passed before being hurled inward. In one instance a loaded car was thrown toward the origin 147 ft. and overturned.

It is frequently a matter of speculation in the investigation of explosion disasters, what pressures may be attained to produce the disruptive effects observed. With abundant inflammable dust there seems to be no limit, as the effect is cumulative. Let us suppose the pressure has attained 120 lb. per square inch, or about eight atmospheres, there are then eight times as many atoms of oxygen present in a cubic foot as there were in a cubic foot at the initiation of the explosion, so that if there is enough dust present to combine with all the oxygen there is eight times the energy liberated by the combustion per unit of space as at the beginning of the explosion.

As the explosion attains high velocity, the compression of each successive layer of air ahead is probably nearly adiabatic, so that the temperature of the air and dust is instantly raised to the point of combustion. Therefore actual transfer of heat from point to point by radiation and convection, which would be necessary at the beginning, might not be when the velocity of the explosion was 2,000 to 3,000 ft. or more per second.

### *Composition of Afterdamp*

The investigation of explosion phenomena would not be complete without obtaining samples of the resultant gases and analyzing and correlating them. In the French gallery the samples were taken in evacuated flasks, the glass stems of which projected into the gallery and were ruptured by the explosion and the gases forced into the flask. Observers could immediately after the explosion turn off the stop cocks of the flasks. This method was not practicable in the mine, so various arrangements were made to seal automatically the flask connections, but there was some uncertainty about the time the sealing was done.

In the British Altofts tests automatic sampling bottles were employed and a similar one was tried at the experimental mine, but it did not prove suitable to the conditions.

A special automatic sampler was then devised and built, in which there are three evacuated steel bottles, the glass ones first used being frequently broken, although put in a recess in the rib and protected by a heavy steel cover. Each bottle has the end closed by a thin plate of glass; a special mechanically operated plunger breaks the glass and seats itself like a cork in the metal neck. This mechanical arrangement is set in motion by the interruption of an electric circuit through the burn-

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ing out of a tin-foil strip exposed to the flame of the explosion. A clock work controls the time of operating the mechanism of the second and third bottle, which can be regulated to obtain any desired time intervals. The time between the burning out of the tin-foil strip and the closing of the first bottle is very small; tests have determined it to be but 0.03 sec. It could not be made less and obtain a full sample in spite of the evacuation of the bottles and the exterior pressure of the gases; as it now operates sometimes the bottle is only partly filled at atmospheric pressures.

At first the tin foil for starting the operation was placed at the sampling point, but in recent tests it has been placed 100 ft. nearer the origin in the effort to secure samples either in the first or second bottle, which would show the preliminary stages of the combustion. The difficulty of getting decisive results regarding this early chemical change is that the samples necessarily are taken on the side of the passageway. The desired point would be at the center, but the violence of the explosions makes this difficult, though some method may yet be devised of accomplishing it. It is known from Mr. Taffanel's investigation in his surface gallery that the flame of the explosion first advances on the center of the passageway, the point of flame being from a few feet to many feet in advance of the flame at the walls.

The analyses of the samples at the experimental mine are most interesting, but it is too early to draw conclusions therefrom. Two typical sets of analyses from the automatic sampler are as follows:

| Test No. | No. of Sample | Seconds[b] after Passage Flame | $CO_2$ | CO | $O_2$ | N | $CH_4$ | H | $C_2H_4$ |
|---|---|---|---|---|---|---|---|---|---|
| 92 | 1 | 0.04[c] | 13.56 | 5.36 | 1.03 | 75.21 | 1.15 | 2.38 | 1.31 |
| | 2 | 0.55 | 7.93 | 5.14 | 7.05 | 76.34 | 0.97 | 2.45 | 0.12 |
| | 3 | 3.00 | 9.98 | 7.38 | 2.60 | 74.76 | 1.51 | 3.52 | 0.25 |
| 114 | 1[a] | +0.04[d] | 0.60 | 0.35 | 19.82 | 79.18 | 0.05 | 0 | 0 |
| | 2 | 0.33 | 0.71 | 0.20 | 19.59 | 79.20 | 0.30 | 0 | 0 |
| | 3 | 1.53 | 7.55 | 6.92 | 4.35 | 75.96 | 1.41 | 3.63 | 0.18 |

[a] Normal air.
[b] Time of closing valve after circuit broken by flame.
[c] Probably sample gathered later through lag of valve.
[d] In advance.

Perhaps the most interesting feature of many of the analyses taken during the series of tests is the increase in carbon monoxide in the second or third samples over that in the first or second sample gathered in the respective test.

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Another interesting feature is the distillation of the coal dust after the first passage of the flame. In Test 114, the analyses from which are given above, the first sample was taken before the explosion reached the sampler, the tin-foil circuit breaker having been advanced 100 ft., the second sample seems to have been taken immediately following the first dart of flame before the combustion was general. The velocity of the flame for the 100 ft. preceding was 1,408 ft. per second as registered by the flame circuit breakers.

### *Development of Mine-Explosion Prevention Methods*

It is needless to say that this extensive and costly experimentation would not have been undertaken for mere demonstrations that coal dust is an explosive agency (necessary as the educational work was for the first year or two). The final object was to obtain such knowledge of the phenomenon that practical methods could be devised for preventing or limiting mine explosions. The first consideration of operators, naturally, is the prevention of ignition of coal dust by use of the well-known though often neglected precautions already available, such as the employment of safety lamps in mines making gas, the use of permissible explosives, safe-guarding electrical installations, lessening the production of coal dust, rigid inspection, etc. These features will not be gone into, but we will confine the present discussion to the methods of making coal dust inert, for in bituminous and lignite mines the dust is always with us, so it must be made harmless in one way or another; and to explosion checking or limiting devices.

Tests in the Pittsburg gallery, and many practical illustrations in coal mines of the United States, had already demonstrated that coal dust so damp that it cannot be brought up into the air by concussion, is not dangerous. The way to judge whether this condition is attained, suggested by the writer of this paper as a result of early Pittsburg tests, is to gather a handful of the dust: if it will stay balled up after packing in the hand it will not be raised in a cloud by a concussion or blast, and therefore will be harmless.

Methods of wetting coal dust have been experimented with at the experimental mine, and, as in commercial mines, great difficulty was found in wetting pure coal dust when it lay in piles, either by humidifying or direct watering. As every coal-mining man knows, coal dust repels water, and water may run over it or under it, like quicksilver, without wetting the dust, even after months of contact. However, in commercial mines where the walls and floor are thoroughly wet, the particles of dust falling on the road will tend to become damp. One fact has recently been found, which seems to be of great value in aiding the wetting of dust by watering or humidifying methods, namely, that where coal dust

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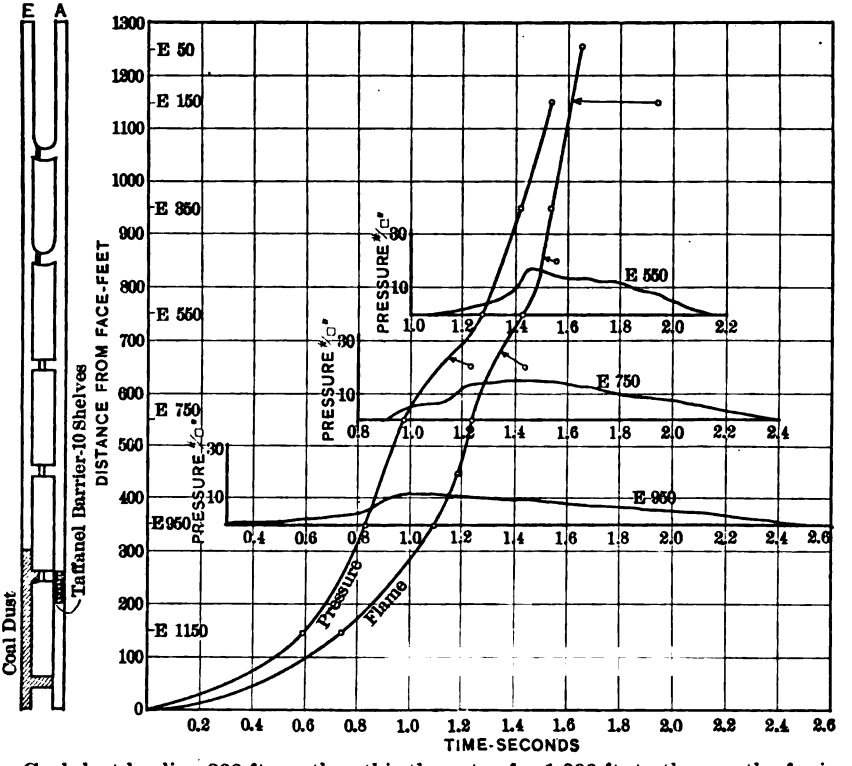
is mixed with shale dust or limestone dust, the repulsion of the coal dust is nullified and water quickly penetrates masses of the mixed dust. This suggests that where watering or humidifying methods are used, it would be of advantage to scatter fine shale dust through the mine, or preferably limestone dust on account of its whitening the ribs, roof, and floor, hus making the illumination of the passageway easier.

Dustless conditions have been proposed as a preventive of dust explosions, to be obtained by attempts to clean carefully the surfaces, and




Coal-dust loading 300 ft.; outbye this the entry for 1,000 ft. to the mouth of mine was cleaned by sweeping and use of compressed-air jets. The explosion traversed the so-called dustless zone, and flame issued from the mouth of the mine. In the air course, 150 ft. from the last open cross cut there was a Taffanel barrier, which stopped the explosion in that direction.

FIG. 2.—DIAGRAM OF TEST 79.

possibly through the employment of vacuum cleaners. The latter have been tried in Europe, but so far have not proven effective, as there was great difficulty in cleaning up coal dust from the rough surfaces of roof, walls, and floor. Striking illustrations of the ineffectiveness of the most careful cleaning have been demonstrated by the attempts to secure dustless zones for testing purposes at the experimental mine. It is neces-



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sary to clean very carefully between experiments, and this is done by means of shovels, brooms, blowing of dust from the walls and floor with compressed air, aided by the ventilating current. In spite of these precautions, in tests with ignition zones of coal dust at the head of the main entry of only 100 to 300 ft. in length, upon being ignited, the explosion has gone 500 to 1,000 ft. through carefully cleaned entry when the walls of the entry have been dry. In one such instances the explosion increased in intensity as it went out (see Fig. 2). There is another factor which enters into this besides the difficulty of cleaning the fine dust out of crevices and recesses in the ribs, namely, that the air wave in advance of the flame carries timbers, pieces of rock, coal and other débris, which abrades the ribs, and it seems possible, from the tests at the experimental mine and observations made in mine disasters, that this may be an additional source of fuel for the explosions, especially in mines where the coal is friable. In considering this it must be remembered how little dust is necessary to propagate an explosion.

As already mentioned, on one occasion 0.032 oz. of coal dust in a cubic foot of space was ignited in the Pittsburg gallery. Mr. Taffanel reports one ignition with 0.023 oz.; and obtained explosions quite regularly with most dusts he tested, employing only 0.070 oz. per cubic foot of space. The cross-section of the experimental mine contains about 60 sq. ft., which for the amount of 0.070 oz. per cubic foot of space would require only 4.2 oz. per linear foot of entry. This amount distributed on the floor and the two ribs would make but 0.19 oz. per square foot of surface, or a layer of dust less than 0.003 in. in thickness. Most of the experimental mine tests are made with five times this amount of dust in order to insure maximum conditions for an explosion.

Many persons do not realize how thick a dust cloud has to be to obtain ignition which propagates beyond the immediate influence of the igniting flame. It has been found in laboratory and surface gallery tests at Pittsburg, that a cloud of dust which will ignite appears very dense to the eye; in fact, in small-scale tests made with electrical ignition it is found the cloud must be so dense that it entirely obscures a 16-c.p. light placed a foot from the glass window. Coal dust is capable of extremely fine subdivision. It has been shown by laboratory tests, as well as gallery tests, that the finer the dust is ground, the more readily it is ignited. Dust through 200 mesh is the standard for testing in the laboratory inflammability apparatus at the Pittsburg station. In the coal dust used in the experimental-mine tests, about 80 per cent. of it will pass through a 200-mesh sieve. The openings of a standard 200-mesh screen are 0.0029 in. square. If it be assumed that the average size of the particles of dust passing through the 200-mesh sieve is equivalent to a sphere of one-half the diameter of the screen opening, or, 0.00145 in., in a cubic foot of such dust cloud weighing 0.07 oz. there

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would be 2,735,000,000 such particles. Microscopic study makes it seem probable that there is a much larger number of coal particles in a total weight of 0.07 oz. The vast number of dust particles that may exist in such a dust cloud makes it easier to comprehend that a cloud of fine dust behaves essentially like an inflammable gas.

Dry dustless zones, at least where these are in entries having coal ribs, proving futile to stop or even check an explosion; the alternatives are to wet the dust, or to use inert dust. Watering throughout a mine, if thoroughly done, is unquestionably a preventive of coal-dust explosions. Short watered zones, however, are not effectual, as the excess of dust from the zone behind is borne along by the advance wave. The explosion tests at the experimental mine which have been made during the spring and summer months, through natural agencies, have had to be conducted with thoroughly wet walls and floor, the side walls and roof being so wet as to be slimy to the touch. When the coal dust has been placed on shelves, particularly cross shelves, the wetness of the walls and floor has not had an appreciable effect in checking the violence of the explosion. Further, the flame has frequently penetrated along dampened zones 500 to 600 ft. in length, there being sufficient coal dust carried along by the explosion to sustain combustion throughout such zones. Many similar instances are found in explosion disasters, which have unfortunately led some to believe that all watering is ineffectual; rather, it points to the necessity of having the watering extend throughout the mine.

*Rock-Dust Treatment.*—The use of rock dust as a preventive of explosions was first urged by William E. Garforth, as a result of his observations following the Altofts explosion disaster of 1886, that certain airways covered with rock dust, which naturally drops from the roof and sides in this longwall mine, were in no instances penetrated by flame. It is generally claimed that in British coal mines, also in French and Belgian mines, extensive watering will seriously damage the walls and roof; although in Germany thorough watering is compulsory, and sprinkling is more or less used in South Wales.

Largely as a result of Mr. Garforth's influence the Altofts gallery was first established, and one of the objectives was to try out rock-dust treatment. Resulting from the favorable tests, the method was adopted immediately at the Altofts colliery, and subsequently at a number of other collieries in Great Britain. The method was therefore tried at the experimental mine, largely in view of the fact that many mine operators of the United States claimed that watering methods are inefficient, and investigation has found that only a very small fraction of the mines, even where watering is done, wet down the ribs and roof, where much of the most dangerous dust (the float dust) collects. Again, in the arid Rocky Mountain regions, the great scarcity of water and extreme dryness of the

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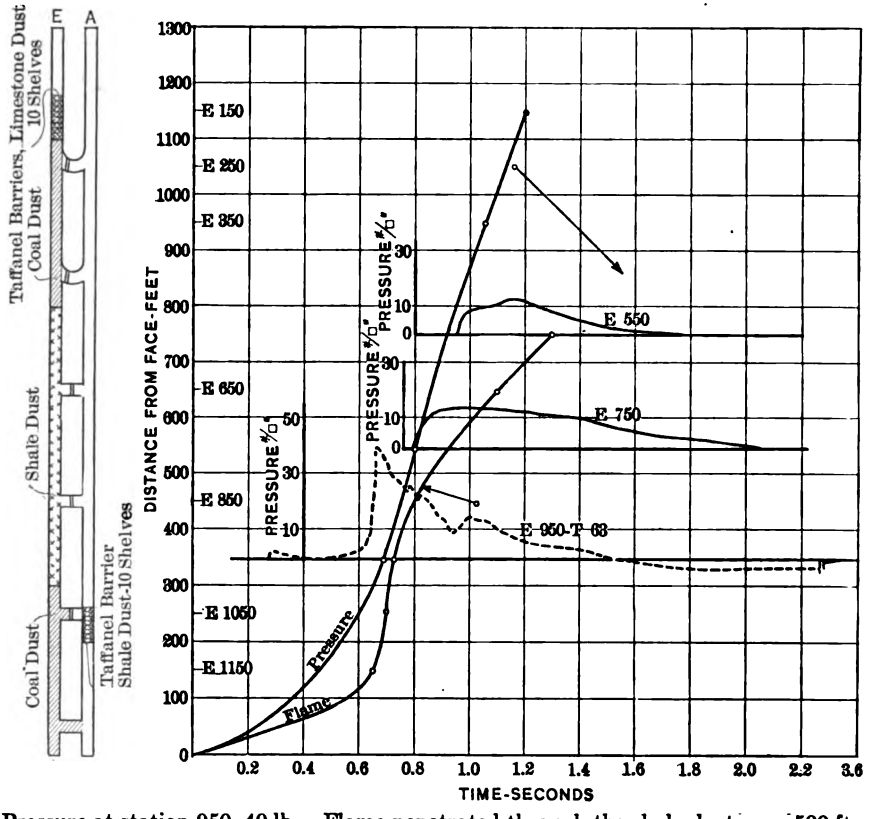
air entering the mine make watering a particularly difficult problem. Several great explosion diasters, which have occurred in that district in mines where considerable watering has been done, indicate the need of providing other means than watering in mines in which there is great difficulty in properly wetting the dust. In fact, one of the greatest disadvantages of watering methods is that so much depends upon the individual men employed to do the watering, for it must be done, not occasionally, but every day in the year.




Pressure at station 950, 40 lb. Flame penetrated through the shale-dust zone 500 ft., but did not ignite the coal-dust loading in the zone beyond.

FIG. 3.—DIAGRAM OF TEST NO. 86, TO DETERMINE PENETRATION OF SHALE-DUST ZONE WITH A STRONG INITIAL EXPLOSION OF PURE DUST.

Humidifying by steam jets is not open to this latter objection, and this method has been found to be excellent in wetting the roof, ribs, and floor in tests at the experimental mine, although, as already stated, the pure dust does not become wet when it is in undisturbed layers or piles.

In the rock-dust tests at the experimental mine shale dust, obtained by grinding roof material, was chiefly employed. This was not ideal in

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that it contained from 8 to 10 per cent. of combustible matter. It would be better if shale could be obtained without combustible matter, as is the case with many of the roof shales found in Great Britain. However, as it is a material which is convenient and available throughout the fields mining the Pittsburg bed, from which a large percentage of the bituminous output of the country is derived, it was thought best to test it out thoroughly, especially since if effectual the proportions of shale to coal dust would be on the safe side. This shale dust, distributed along side shelves, was successful in stopping small initial explosions of coal dust in a few hundred feet; but with an initiating zone of 300 ft. of coal dust, with resultant high pressures of 30 to 50 lb. per square inch on entering the shale-dust zone, the flame penetrated from 250 to 650 ft.; as, for example, in mine test No. 86 (see Fig. 3 showing flame extent and velocity, also pressure records at certain stations), in which the initial pressure from the coal dust was 40 lb., the flame traveled for about 300 ft. before dying away. In Test 87, in which the pressure was but 30 lb., it died away in less than 250 ft.

Limestone-dust screenings, which are a waste product in many districts, were also tried in zonal treatment, but owing to the greater specific gravity of the limestone, the results were no more effectual than with the shale dust placed on the side shelves, although the limestone contains no combustible matter.

The conclusions drawn from these tests, and others in which the rock dust was placed on cross shelves, are, that the most effectual method is to place the dust on cross shelves, and on overhead timbering, if there be such; then the advance air wave causes the entire dust load to fall in a shower and practically all of it is brought into play, whereas when placed along the ribs and floor it is not so easily brought into suspension when the explosion pressures are low. With this cross-shelf arrangement, which is practically a continuous Taffanel barrier, the greater specific gravity of the limestone dust has no appreciable disadvantage since, after tests, it is found widely scattered, and on the other hand there is some liberation of carbon dioxide, which further tends to quench the combustion.

In connection with the extinguishment of the flame of an explosion, it is interesting to note that in an effort to start a strong ignition of coal dust 75 lb. of coal dust was placed on eight cross shelves from immediately in front of the cannon to 50 ft. distant. To the surprise of all present at the test, the flame did not pass beyond these cross shelves. The test was repeated five times in all, with the same results except in one case when propagation occurred. The representatives of the foreign testing stations were asked if they had experienced such results. Subsequently M. Watteyne of Belgium reported he had tried the same experiment, and obtained confirmatory results. Coal dust placed in this way appeared to have acted like a rock-dust barrier; the coal dust falling rapidly across

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the path of the flame cooled it below the ignition point of the dust. It must be clearly understood, however, that an excess of coal dust is not a protection even in an ignition under any ordinary circumstances, as it is almost impossible to conceive of its being similarly placed under working conditions. If there is merely an excess of dust on the floor, or in the gob, only a portion of it rises in suspension, which would ignite and propagate if other conditions were favorable. In other words, it was not simply an excess of dust, as there was no more than in the usual test, but it was the way it was placed so that it dropped across the initiatory flame as a mass, and absorbed the heat. It was interesting to note that in each case there was much burning dust found on the platform in front of the cannon after the test. When there is an excess of coal dust along the entry this is not a hindrance to an explosion as the advance waves carry it along until the flame reaches the dust; they automatically drop it if the waves slow up. Theoretically, 0.123 oz. of Pittsburg dust, if burned completely, will combine with the oxygen in a cubic foot of space at ordinary pressures. However, when an explosion becomes violent as it advances along a passageway, the pressures may rise to 7 or 8 atmospheres or more; the dust cloud at such a point must contain a proportionately larger amount of dust to use up or combine with all the oxygen in a given space. It has been observed that in most violent mine explosions there is usually a great excess of dust over that necessary to combine with all or most all of the oxygen in the zone of violence.

### *Rock-Dust Barriers*

Despite the greatest precautions taken, it is inevitable that dust explosions will occur from one cause or another, and on this account it is deemed advisable that there should be a secondary line of defense to limit or stop them from going out of the originating district. Mr. Taffanel, director of the French Station, was the first to plan what has been termed a "barrier." He tried heaps of dirt across the passageway and observed that they had an important effect in choking explosions. Manifestly such methods could not be used in mines, so he devoted himself to making arrangements which would give equivalent results. This led to the adoption of the system of cross shelves placed close together, about 2 yd. apart, on which shale dust, flue dust, or fine ashes was placed, each shelf being ½ m. (20 in.) wide. At first he specified 10 such shelves, but later as a result of further experiments he found it advisable to increase the number to 15. The principle of this is that the air wave in advance of the flame launches the large mass of incombustible dust previously placed on the shelves, into the air and makes a dense cloud, which cools the flame of the explosion. This arrangement was adopted by many of the mines in France. At the experimental mine the barriers, which in com-

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pliment to Mr. Taffanel are termed "Taffanel barriers" (Fig. 4), were employed to limit the zone of violence, as otherwise it would have been impossible to make tests frequently, owing to the extensive damage which results from explosions of pure coal dust, when permitted to traverse over 500 or 600 ft., to the interior of the mine and the roof and concrete linings near the entrances; the arched linings have been badly disrupted by some of the extensive explosion experiments. In Fig. 5 is shown a diagram of a

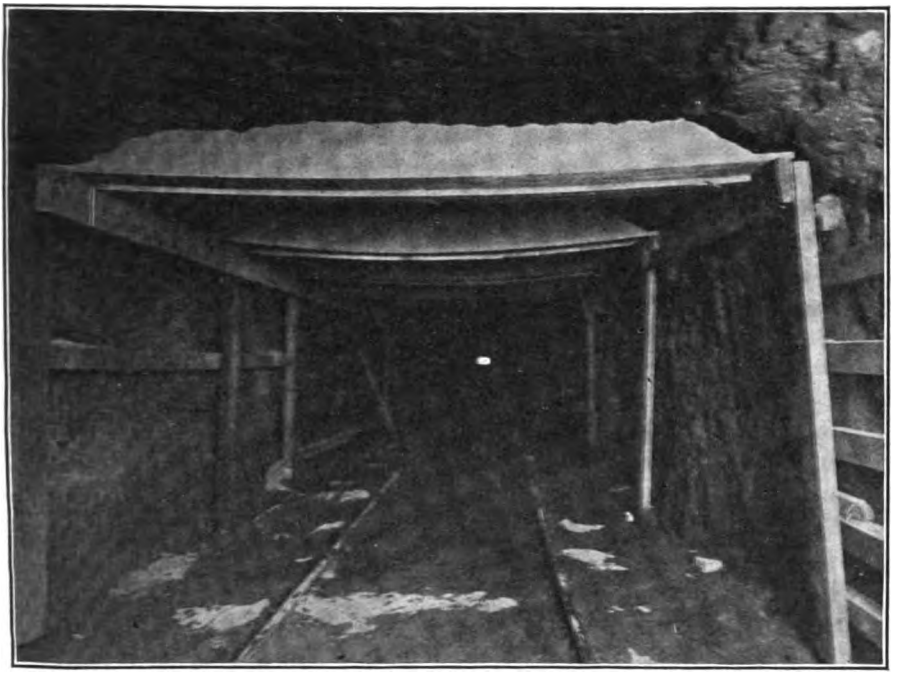

FIG. 4.—TAFFANEL ROCK-DUST SHELF BARRIER.

test in which the flame of the explosion was extinguished by the rock-dust barrier, the pressure dropping from 119 to 12 lb. in 400 ft. The barriers proved uniformly successful in stopping explosions, except in a few instances of very light explosions or other inflammations where the flame passed the barrier. There is also the disadvantage that where the air current was strong, as it generally is along the haulage ways of American mines, the dust was blown from the shelves. Again, in mines where the watering system is employed, the dust will likely become dampened, and when wet it is of course ineffectual in quenching explosions.

These disadvantages led the writer to design some special forms of inclosed barriers which would operate with lighter pressures than the Taffanel open barriers. There are two forms of these: One the box barrier, where six boxes containing about 700 lb. of dust are hung

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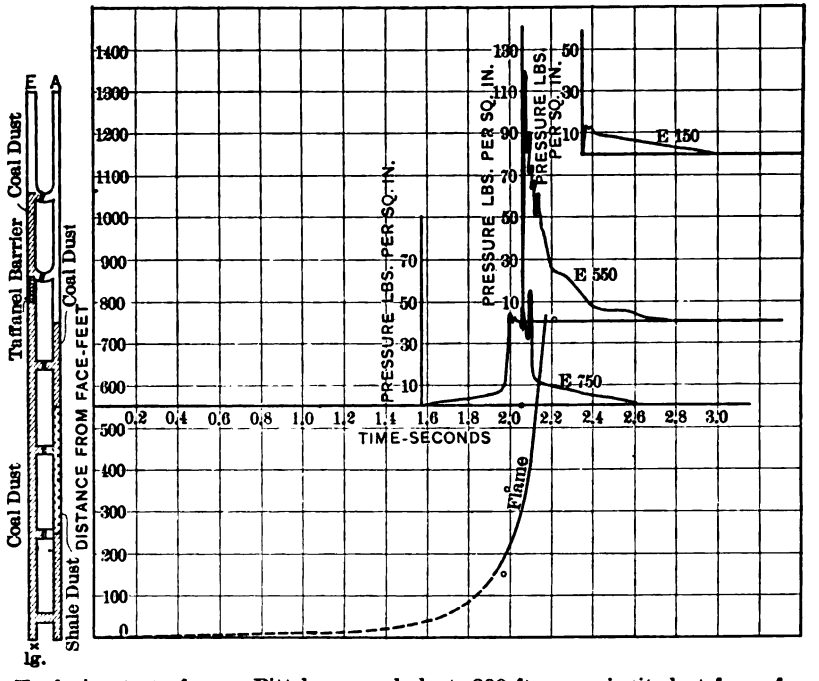
from the roof and timbers across the entry. In the final design (Figs. 6 and 7) the arrangement comprises a light open box hung by four freely suspended rods, with eyes, from open hooks fastened in the roof or overhead timbering. The box within limits may swing freely; when, however, the movement is over $\frac{5}{8}$ in. or thereabouts from the middle position, the two suspending rods on the side toward which the movement is directed, which fasten to the bottom edge




Explosion test of pure Pittsburg coal dust, 800 ft. zone, ignited at face of main entry. Explosion started with unusual slowness, but developed into one of great violence, maximum pressure at station 550 being 119 lb. The explosion in main entry was stopped by the Taffanel barrier at end of coal-dust zone. The coal-dust zone outbye this was not ignited, and the pressure fell from 119 lb. at station 550 ft. from mouth to 12 lb. at station 150. In the air course the coal-dust loading extended for 250 ft.; outbye this there was a 300-ft. zone of shale dust, with coal dust beyond. The flame penetrated through the shale-dust zone, but did not ignite the coal dust beyond. The explosion showed very violent results in the interior of the mine.

FIG. 5.—DIAGRAM OF TEST 46.

of the box, are struck by cleats fastened to the upper edge of the box, and if the movement is sustained by the advance wave of the explosion, the eyes of the suspending rods are pushed off the open hooks and the box swings downward, pivoting on the points of suspension of the rods on the opposite side of the box. The rock dust in the box is partly discharged, but some is held by the bottom of the box which, is constructed of two loose boards resting on cleats attached to the side of the box. The boards are hung

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by chains attached to the roof or overhead timbers, which permit the boards to drop 3 or 4 in., then they are caught by the chains so that a portion of the rock dust is held on them, unless the explosion is so violent that all parts of the box are ruptured. In the latter event the rock dust is automatically launched and mixed with the air and coal-dust cloud.

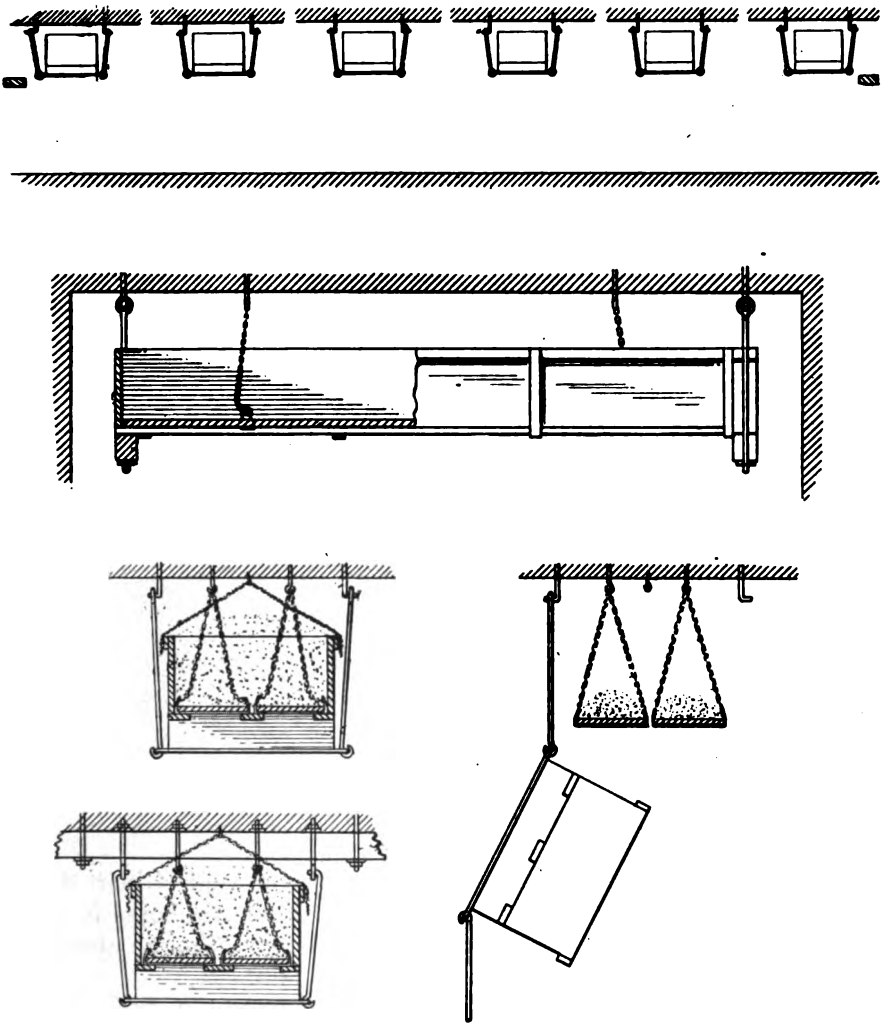

FIG. 6.—RICE-BOX BARRIER.

The arrangement of hanging the bottom boards, which allows some of the retained dust to sift down slowly, is to take care of the more difficult situation brought about in light explosions, when there may be a considerable time interval between the advance air wave and the flame. It

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has the further merit that if a box is accidentally tripped off, the bulk of the weight is caught by the chains, and it is not believed a serious injury would be caused to a man who might be caught underneath.

To protect the rock dust, in an ordinary mine working, from becoming damp or admixed with float coal dust, the box should be covered by a waterproof cloth attached over the center line of the box to the roof, so that as the box dumps the cloth is automatically pulled from its top.

As indicated above, there is less difficulty in stopping a violent explosion than a light one, because in the violent explosion the whole

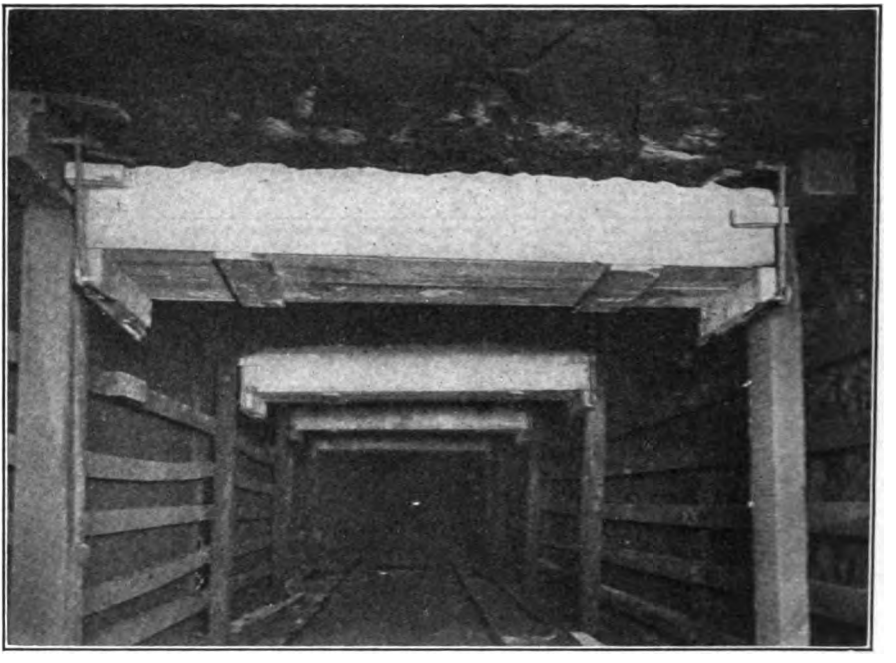

FIG. 7.—BOX ROCK-DUST BARRIER.

mass is disrupted, and the rock dust forms a thick cloud of inert dust. Many tests have shown that the rock dust from the boxes is very widely distributed; and when limestone dust is used it can be observed ejected from the mouth of the mine by the air blast.

Another form of barrier, which is illustrated by Figs. 9, 10, and 11, is termed the "concentrated barrier." In this the rock dust is placed on a large platform hung close to the roof. The platform consists of boards, hinged separately to a cross timber, and held up at the other end by an angle iron. This angle iron, in turn, is held up by a system of levers which are tripped by vanes. There are advance vanes connected by a wire with the lever which pushes upon the tripping lever. In the experi-

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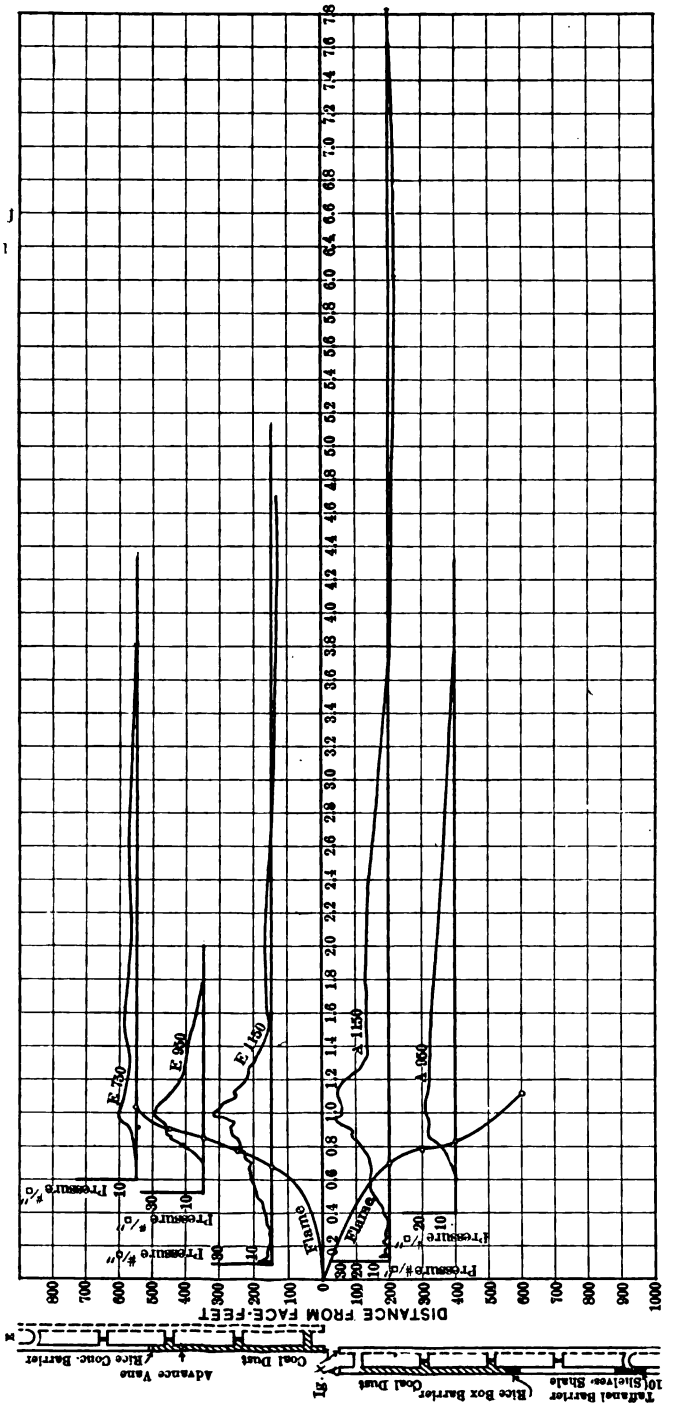


This shows a symmetrical explosion of pure coal dust in the main entry and air course to determine if the concentrated barrier and box barrier in the respective entries would stop the explosion. The explosion was stopped by them.

FIG. 8.—DIAGRAM OF TEST 105.

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ments at Bruceton the advance vane has been placed about 100 ft. from the concentrated barrier. As erected in a commercial mine there would be a similar vane on the other side, as the barrier is symmetrical and will extinguish an explosion approaching from either side. In case of possible accidental tripping, and danger when a man or trip of cars is passing, the inner end of each of the platform boards is fastened by a loose chain to the roof, or to a cross timber, permitting only a limited drop. As hitherto arranged, half of the boards dropped about 9 in., and the other half about 18 in. The platform holds about 3 to 4 tons of rock dust, and is quite compact, occupying the space between timber sets, which are placed 7 ft. apart. In mines with low roof it may be

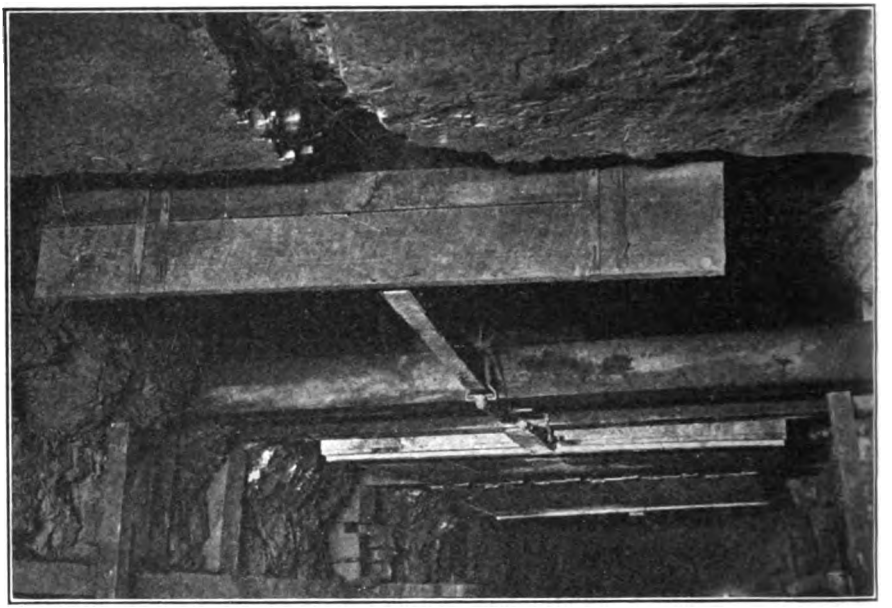

FIG. 9.—CONCENTRATED BARRIER, SHOWING UNDER SIDE OF SAME, AND BUMPING VANE IN FRONT; ALSO TRIPPING WIRE FROM FIRST VANE, 100 FT. IN ADVANCE.

necessary to brush down the top to make room. The appearance of the barrier in passing under it is like that of an ordinary overcast.

The leverage system of operating the concentrated barriers can be arranged so that it will operate with almost any desired pressure. As arranged at the experimental mine, it will not discharge until the pressure is equivalent to that of a wind of about 40 miles per hour (3,500 ft. per minute). In other words, a heavy gale. It is thought that the proper places for such concentrated barriers are at intervals in the main entries, and in each side entry near its mouth, so that if the mine is paneled, an explosion will be confined to one panel.

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It is interesting to note that Mr. Taffanel had independently decided that it was desirable to have such inclosed barriers, and had designed a concentrated barrier just prior to his coming to Pittsburg recently for collaboration on special tests. This was also tried out successfully at the experimental mine, but it did not have the advance vane for tripping it, and this Mr. Taffanel thought was a considerable gain to provide for very rapid explosions.

As far as the tests have gone, the box barriers and the concentrated barriers are equally effectual, and the adoption of one or the other at

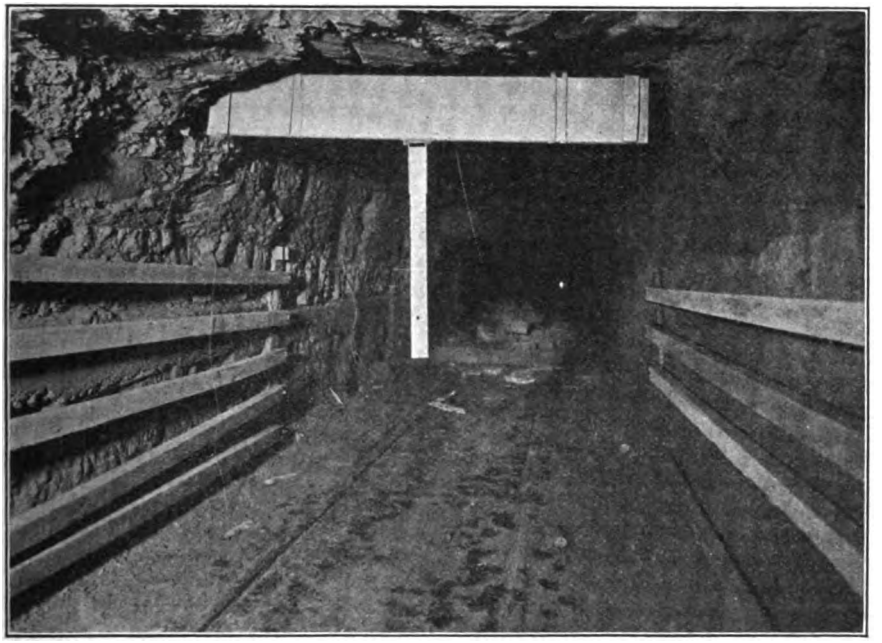

FIG. 10.—CONCENTRATED BARRIER AFTER EXTINGUISHING A VIOLENT EXPLOSION.

the commercial mines will perhaps depend somewhat on local conditions.

The success of the foregoing barriers led to the designing of a rock-dust protected ventilating door, since ventilating doors are invariably destroyed in all explosions of any extent; and further most of them being near the mouths of side entries, it would be of the greatest value to have an explosion wave arrested there (see Fig. 12).

The principle underlying the door is to have the inclosing frame work put together without nails, and keyed in position by the door frame, so that when the door frame is blown out the mass of dust held behind the side and top boards is dislodged, and forms a dust cloud.

Again, it was found advisable to sustain a certain amount of dust close to the roof, and this is accomplished by attaching to the boards over the

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door frame loose chains, hung by bolts into the roof or to higher timbers. These sustain some of the dust after operation of the barrier and allow it to sift down in case there is a considerable time interval between the arrival of the pressure wave and the advance of the flame.

It has already been found that the door frame must not be made too light or it is likely to be blown down by the shock wave from the cannon. It can therefore be a substantial frame, well braced to resist the ordinary shock wave from blown-out shots.

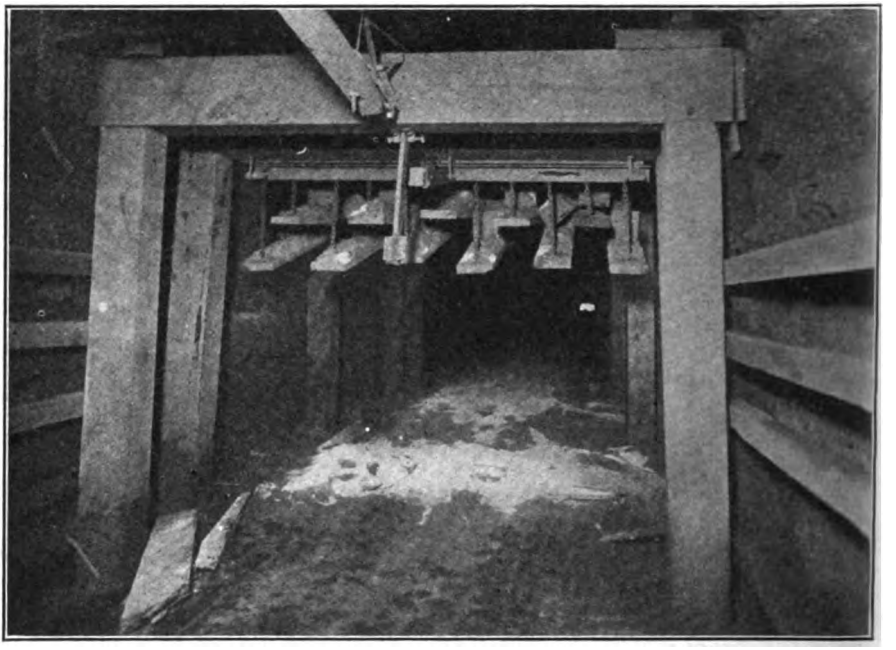

FIG. 11.—CONCENTRATED BARRIER AFTER LIGHT EXPLOSION, WHICH IT EXTINGUISHED.

Another device with a similar object is a rock-dust protected overcast. As everyone knows who has investigated explosion disasters, overcasts are particularly vulnerable, though built of concrete and iron. Even with a small pressure per square inch, the large surface exposed gives a tremendous total pressure, which it is practically impossible to withstand by any ordinary construction. Therefore, where natural overcasts cannot be made which will provide a good thickness of natural strata between the airway and entry, it seems just as well to provide comparatively light overcasts made of wood, or if preferable, of light concrete or brick construction, not reinforced, making the walls double, and filling same with rock dust, or flue dust, holes being provided for occasional inspection, and if necessary, renewal of the dust. By having

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such rock-dust protection, if the overcast gives way by the pressure of an explosion, the flame is prevented from entering the airway. Oftentimes this might isolate a large district of the mine, and thus save the men in it.

To complete the devices for the protection of vulnerable points a rock-dust stopping was designed, which presents some unique features. The first tests were simply hollow walls, but the objection to this was that the stoppings are likely to go out quickly at the first pressure wave, so if there is a considerable interval between this wave and the flame there may be a chance for the rock dust to settle and the flame to pass

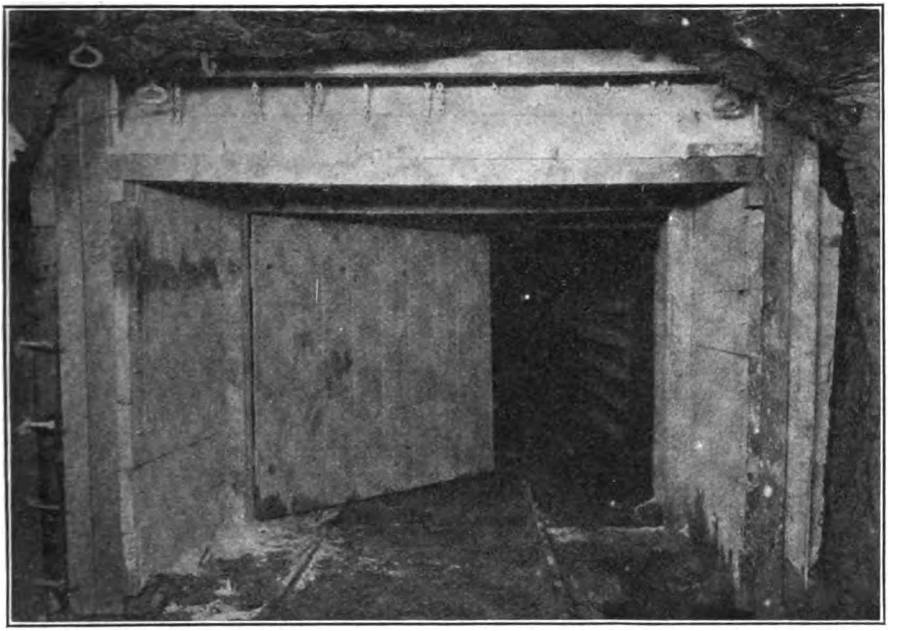

FIG. 12.—ROCK DUST PROTECTED DOOR WITH FRONT BOARD REMOVED TO SHOW LOOSE CHAINS TO SUPPORT BOARDS OVER DOOR FRAME IF EXPLOSION OCCURS. TAKEN BEFORE SUCCESSFUL OPERATION.

overhead; further, the stopping is liable in go out in a mass, and the rock dust would not be well distributed. In the design which was tested with successful results, and which is illustrated in Fig. 13, the ease with which stoppings are blown out and their vulnerability are taken into account. The dust protection is practically in two halves, one independent of the other, and advantage is taken of the angle of repose of the rock dust. Practically, the device in each half consists of a series of shelves one over the other, with a space between the two halves in which there may be any kind of stopping material, either rock dust, ordinary masonry, concrete, or wood. When the pressure is first thrown on the

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stopping it will blow out as a mass the inbye side, but if the explosion is not a heavy one, the outbye, or first, set of shelves will remain standing with a certain amount of rock dust on same, to provide material for extinguishing the flame. To protect the rock dust from getting wet, and yet at the same time to permit it to be inspected, separate shallow cur-

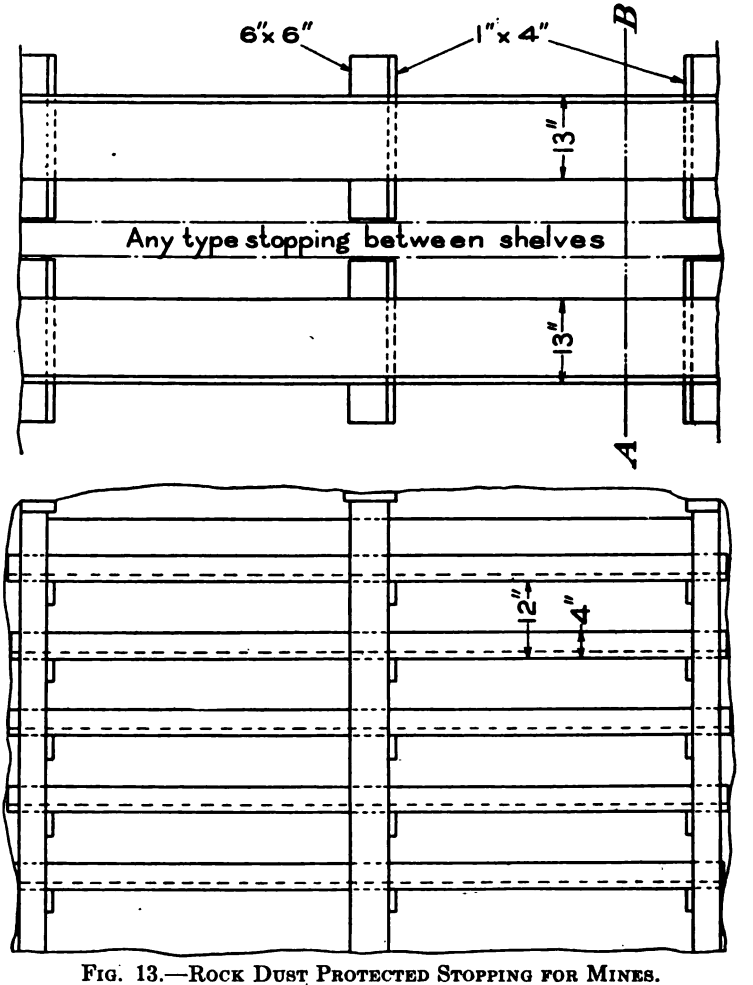


FIG. 13.—ROCK DUST PROTECTED STOPPING FOR MINES.

tains are hung from shelf to shelf. It is not desired to make these in one large curtain, since when the whole stopping is under pressure there is a liability of its going out in a mass, but if the curtains are in narrow strips they will flap up when the pressure wave strikes the stopping barrier.[4]

[4] Patents have been applied for by the author for the foregoing described devices, prospective rights being assigned to the Bureau of Mines, for the benefit of the public.

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*Future Investigations at the Experimental Mine*

Besides the investigation into protective devices, it has become apparent that there is need for testing dusts from coal beds of different districts in order to determine the degree of inflammability or explosibility of the pure coal dust, and then to determine what admixture of roof material, or where artificially ground rock dust is introduced, the amount of the foreign inert material, which will make the coal dust harmless; in other words, to determine the explosive limit of the particular dust (*a*) to prevent ignition, (*b*) to prevent propagation. For example, for Pittsburg dust it requires 70 per cent. of the roof shale in the mixture to

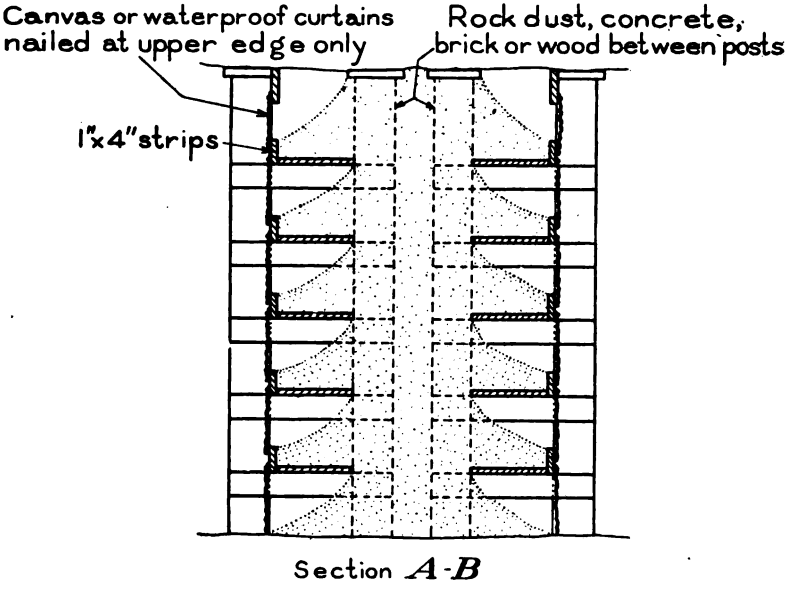
Canvas or waterproof curtains nailed at upper edge only
Rock dust, concrete, brick or wood between posts
1"x4" strips
Section A-B

FIG. 13A.—SECTION ON LINE *A-B* OF FIG. 13.

prevent ignition, but to prevent propagation it requires about 80 per cent. This is when the local shale is used; but when limestone is employed in the mixture, owing to there being no combustible matter in it, 60 per cent. of the mixture will prevent ignition, and 75 per cent., propagation. One low-volatile high-carbon coal which was tested would not propagate an explosion with only 20 per cent. admixture of shale, so that a mine producing coal of this character would apparently be reasonably safe from dust explosions by the introduction of a relatively small quantity of rock dust. On the other hand, this coal in a mixture with 40 per cent. shale propagated an explosion when 1½ per cent. methane was present.

In tests of dusts from other districts at the experimental mine, owing



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to the possible admixture of particles of coal abraded from the ribs, which would alter the results, it seemed necessary to line with concrete a portion of the passageways where such tests are conducted, and this is now being arranged at the time of preparing this paper.

One of the most important series that will have to be undertaken is that of determining how much the limit of explosibility of each dust is changed by the introduction of small quantities of firedamp. Some tests have been made, but not enough from which to draw conclusions. Special arrangements are being made for the introduction of gas at the face of one of the entries so that it may be put in behind a diaphragm, then mixed with the air by a fan, in any desired proportions. At the present time it is only feasible to use 2 per cent. or less throughout the whole mine, whereas it is often desirable to make a test with an explosive mixture of gas at the face.

It is hardly necessary to say that the number of kinds of experiments that might be tried is almost infinite, and the work will have to progress slowly, step by step.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Plant of the Duplex Process for Making Steel

BY J. K. FURST,* NEW CASTLE, PA.

(Pittsburg Meeting, October, 1914)

THE reasons for manufacturing steel by the duplex process are, briefly: saving of time; increasing output for capital invested; and avoiding the difficulty sometimes experienced in obtaining scrap. On account of the latter reason, some plants duplex only at such times as the price and quantity of scrap in the open market warrant. At some plants the use of the duplex process enables the blast-furnace practice to be simplified, because the furnaces run more smoothly and produce more iron when not held to a strict specification as to the silicon and phosphorus therein.

The duplex process was practiced in Witkowitz as far back as 1878, but it was left to the steel makers of this country to develop it along practical, commercial lines. The American companies manufacturing steel by the duplex process at the present time are: Tennessee Coal, Iron & Railroad Co., Maryland Steel Co., Bethlehem Steel Co., Pennsylvania Steel Co., Jones & Laughlin Steel Co., Colorado Fuel & Iron Co., Lackawanna Steel Co., Dominion Iron & Steel Co. of Canada.

All of these companies, except the Dominion Iron & Steel Co., combine the acid Bessemer with the basic open-hearth process, while the latter combines the basic Bessemer with the basic open hearth.

## *The Tennessee Coal, Iron & Railroad Co.'s Plant*

The installation of a converter of 15 tons capacity at the Ensley plant of the Tennessee Coal, Iron & Railroad Co. in 1904 marked the introduction into this country of the duplex system. The experience gained in the employment of the Bessemer converter had demonstrated that there was a shortening of the time required in the open-hearth furnace and a reduction of the outlay for fuel and lime, thus affording substantial evidence of the economies resulting from this method.

The scarcity of steel scrap in the Birmingham district; its dependence on its own resources for pig iron; its lower cost of fuel; and the lower lime cost with the use of the direct metal in the converter, were the factors

* Chief Engineer, Pennsylvania Engineering Works.

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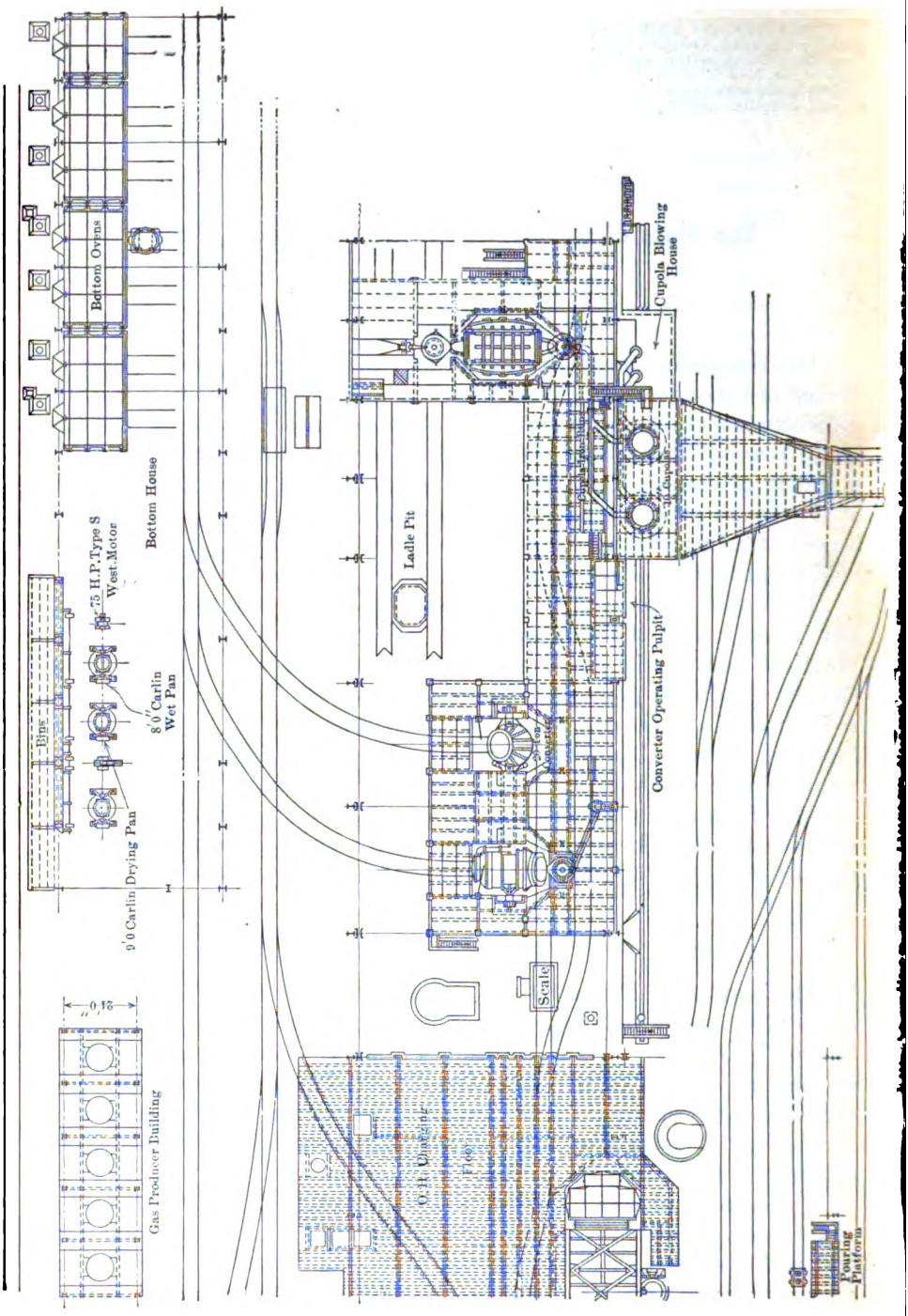


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which brought about the construction of the second duplexing plant, which went into operation in the fall of 1907 and has been operated continuously ever since. At the time the original 15-ton converter was installed at the Tennessee Coal, Iron & Railroad Co.'s works, the steel-making plant consisted of six blast furnaces, one 250-ton primary regenerative furnace, and eleven 50-ton open-hearth furnaces, of which one was stationary and ten were tilting. The original converter was erected between the 250-ton regenerative furnace and the open-hearth plant, and was served by an electric traveling crane. It was operated by one 75-h.p. electric motor through worm, pinion, and gear, and was the first converter in this country to be operated electrically.

The second plant built by this company marked the beginning of duplexing in this country on a large scale. All the details of this plant were worked out so carefully that practically no changes or improvements have been found necessary, or even desirable, since then. As shown in Fig. 1, this plant consists of six modern blast furnaces, and two batteries of basic open-hearth furnaces, each consisting of four 100-ton hydraulically tilting furnaces, having hearths 15 ft. wide by 44 ft. 2 in. long. Between the two batteries of open-hearth furnaces, and in line with them, is located the Bessemer building. The converters and the converter-building floor are served by a 100-ton traveling crane. This building houses the two 20-ton converters; one 250-ton and one 600-ton hot-metal mixer; and two 10-ton cupolas for melting scrap, located directly back of the pouring-end of the mixers, with the necessary equipment of troughs so that they can be tapped into the converter-charging ladle which plies between the mixers and converters.

The iron comes from the blast furnaces in 25-ton hot-metal cars, and is poured into the mixers by means of hydraulic lifts. The iron is poured from the mixers into the converter-charging ladle, which travels over a charging floor at an elevation of 14 ft. 7 in. above the open-hearth charging floor. The molten iron is charged into the converter by means of a hydraulic post crane. After it has been blown, it is discharged into a 20-ton ladle car and carried directly to the open-hearth furnace. The bottom house, situated directly opposite the converters, contains seven bottom-drying ovens, crusher, wet and dry pans, and the necessary bins, etc., for making up the bottoms. After the bottoms have been made up here and dried, they are placed on the hydraulic jack cars, of which there are two, and transported to the converters as required.

The distinctive feature of this plant is that the converters are located directly between, and in line with, the two batteries of open-hearth furnaces, thus minimizing the haul of the blown metal. The track over which the blown metal is conveyed to the open-hearth furnaces is located in the open-hearth building, adjacent to the furnaces and between the charging-box track and the furnaces. The ladles are drawn up in front

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of the furnace doors and the contents poured directly from the ladle, without removing it from the car, into the charging spout. The ladle is tilted by an overhead traveling crane.

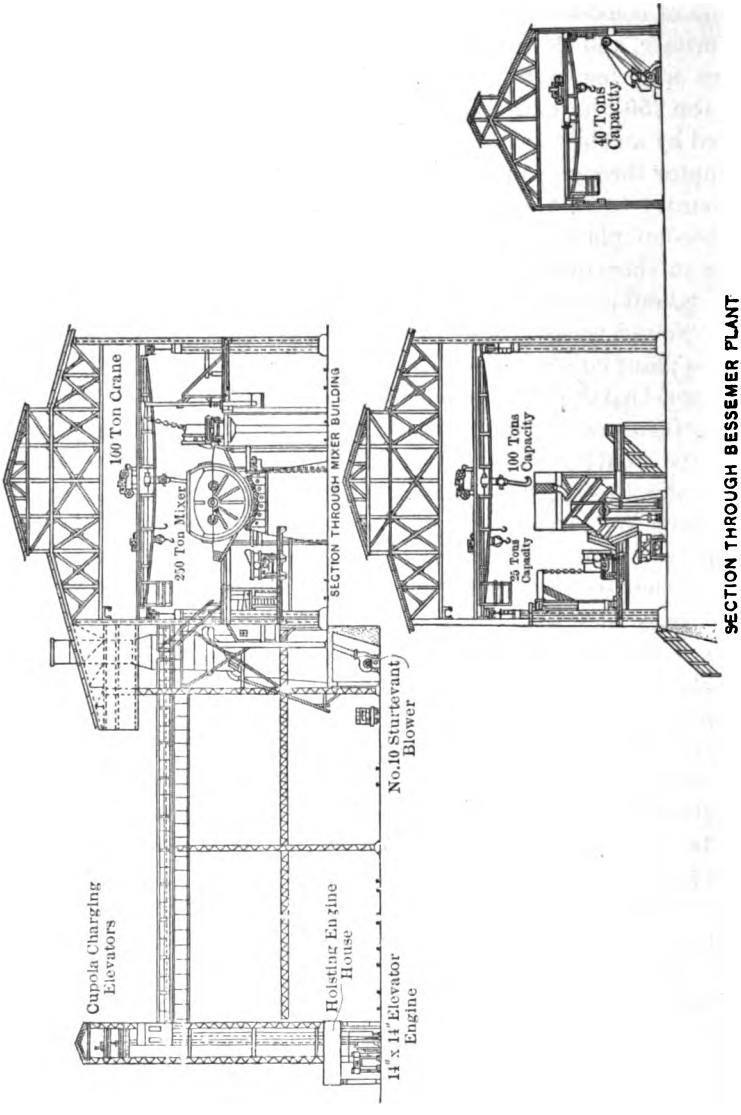


FIG. 1B.—SECTIONS THROUGH PLANT OF TENNESSEE COAL, IRON & RAILROAD CO.
SECTION THROUGH BESSEMER PLANT

The vessels, as shown in Fig. 2, are swung on well-designed, heavy, cast-iron column stands and are 40 ft. center to center. Instead of following the usual practice of riveting the trunnions directly to the shell, they are secured to an annular supporting ring, the vessel being secured

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in place by means of lugs and keys. This supporting ring not only serves to prevent the wearing of the sides of the vessel in the rivet holes which have become distorted by the constant reversal of stresses on account of turning the vessel from time to time, and which has been a source of great inconvenience in the past at some of the works manufacturing Bessemer

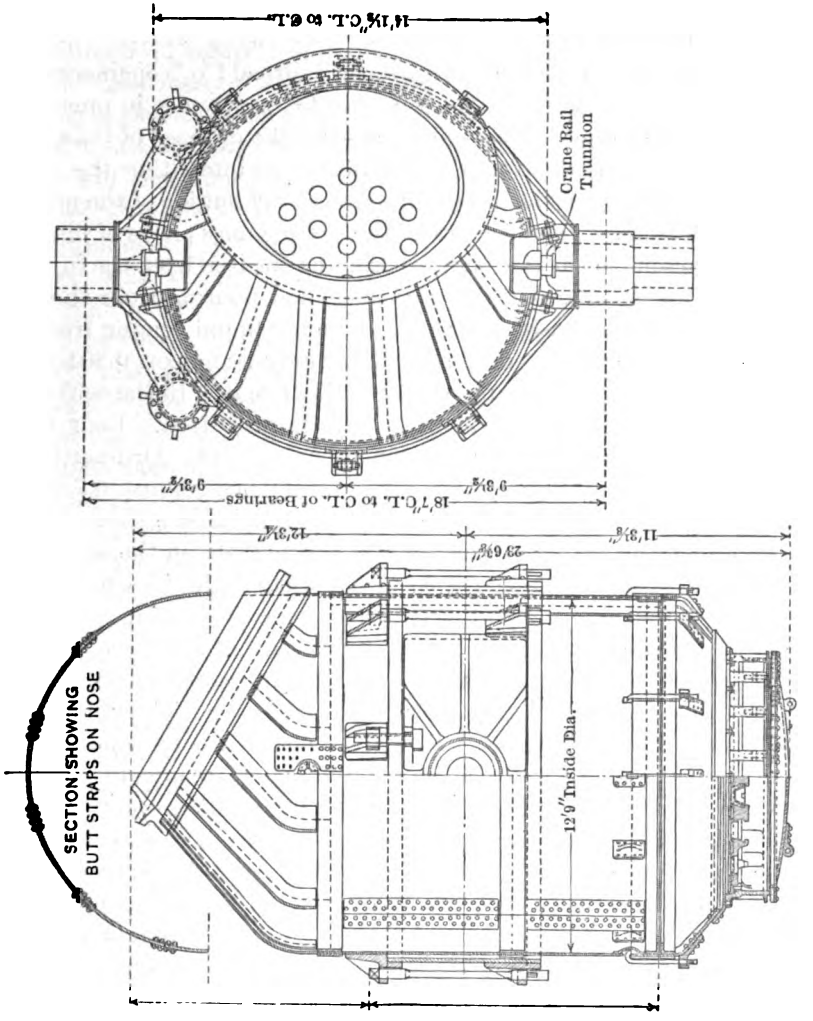


FIG. 2.—20-TON CONVERTER, TENNESSEE COAL, IRON & RAILROAD CO.

steel, but also facilitates the removal of the vessel for relining and repairs. A spare vessel, lined and ready for service, is kept standing convenient to the vessels in operation. The tuyère plate has 26 tuyère openings. The vessels are operated hydraulically by two vertical racks working on one pinion for each vessel. Blast is supplied to the converter from an

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Allis horizontal, cross compound, Corliss blowing engine (46 by 88 by 84 in. and 60 by 84 in.) through a 30-in. diameter air line to the 24-in. air valves beneath the pulpit, thence through an 18-in. air line to the blast trunnion of each vessel. The blast line, just before connecting with the trunnion sleeve, is provided with a gas-escape valve, which allows any back flow of gas to escape to the atmosphere rather than into the main air line. The blast elbow and wind box are of the usual design. The unusual size of the converters was a matter that received careful consideration at the outset by the Tennessee Coal, Iron & Railroad Co.'s engineers, and the 20-ton vessel was adopted, since it afforded economies in operation, by decreasing the refractory cost and reducing the expense of blowing.

While it is not the intention of this article to go into either the chemistry or the metallurgy of the duplex process, a brief outline of the method of operation of some of the followers of the process may prove of interest: The methods employed at this plant of bringing the iron from the furnace, charging the mixer, charging the converter, and pouring from the converter, have been previously described. The hot metal coming from the blast furnaces to the mixer is of the following analysis: Silicon, 0.80 to 1.25; phosphorus, 0.9 to 1.0; manganese, 0.3 to 0.4 per cent. In the converter, all of the metal is desiliconized and partly decarbonized. Four ladles of blown metal are required for each open-hearth heat. Ordinarily the first two are blown soft, analyzing: carbon, 0.1; phosphorus, 0.7 to 1.0; manganese, 0.08 per cent.; and the second two ladles are partly decarbonized, the percentage of carbon blown out depending on the amount of scrap charged in the open-hearth furnace. Before the blown metal is poured into the open-hearth furnaces, burnt lime, iron oxide (the latter in the form of scale or ore), and about 15 per cent. of scrap are charged; then the two ladles blown soft are poured in, and lastly the two ladles partly decarbonized, containing 2 per cent. carbon, or slightly more, are added. When the second two ladles are poured into the open-hearth furnace a violent reaction takes place, which transfers practically all of the phosphorus into the slag. It is here that the advantage of the tilting type of open-hearth furnace, over the stationary type, comes in, as at this juncture the furnace is tilted back, allowing the excess slag to run out of the doors into the slag cars beneath the furnace. Recarbonizing of the steel at the end of a heat is not often found necessary, the practice making it possible to catch the carbon on the way down. Ferro-manganese is added for the manganese. It took from 1 to 1½ hr. to charge the blown metal into the furnace, and about 1 hr. to finish the heat. Records kept show a variation of from 4 to 8 hr. per heat. The time required for blowing the heats in the converters varies, depending on the silicon content, the blast pressure, and other factors. Desiliconizing takes from 2 to 10 min., and decarbonizing from 12 to 20 min.

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*The Dominion Iron & Steel Co.'s Plant*[1]

At about the same time the Tennessee Coal, Iron & Railroad Co., situated at the southern American boundary of the steel industry, was considering the question of duplexing on a large scale, the Dominion Iron & Steel Co., situated at the extreme northern boundary of the steel industry, concluded to install a duplexing plant. The Nova Scotian ores were similar in many respects, but had a higher phosphorus content, and the chief reasons why duplexing was considered advantageous were the long period of time otherwise required in the open hearth, and the difficulty of operating the blast furnaces under a strict specification as to silicon and phosphorus. Consequently, in July, 1906, they let the contract for two 15-ton Bessemer converters, with the necessary building and equipment. Construction started in December of the same year and the first heat was blown in May, 1907. The vessels were operated for some time with an acid Bessemer lining with very satisfactory results, which, however, seemed short of the maximum possibilities. It was thought that, by operating with a basic lining, better results might be achieved. This was at the time considered somewhat experimental in view of the fact that in Europe, where the basic Bessemer practice was commonly pursued, it was considered that at least 1.75 to 2.25 per cent. phosphorus was necessary. However, the necessary alterations in the bottom house were made, the mica schist lining of the vessels was removed and one of stamped dolomite and tar substituted, and the basic Bessemer process was begun. This method has been in use ever since.

In the year 1910 a third vessel, of the same size and design, was added to the original plant. At this time, the steel-manufacturing department consisted of four blast furnaces, one 300-ton hot-metal mixer, three 15-ton Bessemer converters, two in actual service and one spare, and ten 50-ton basic open-hearth furnaces of the Campbell tilting type. Of the ten open-hearth furnaces, nine are operated according to strict open-hearth practice, furnace No. 1 alone being used in conjunction with the duplex process. Also, there were under construction at this time two blast furnaces and two 500-ton mixers. The mixer building and the converter building are situated between the blast furnaces and the open-hearth plant. The vessels are served by a traveling crane, which operates throughout the full length of the building. The three vessels are arranged in line in the Bessemer building, 36 ft. centers. The vessels are of the eccentric type, having a shell diameter of 10 ft. 9 in., with trunnions riveted directly to the shell. The wind box, blast elbow, and trunnions are of the usual design; but the tuyère plate, instead of being of the usual type with 6-in. openings in which refractory tuyères are placed, is a solid plate provided with 73 $\frac{3}{4}$-in. tuyère openings. The vessel is electrically operated, the power being transmitted through worm, pinion, and gear

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directly to the trunnion, which gives ample power at all times, although when the vessel is in a horizontal position with bottom off, it requires about all of the power to bring it to the vertical position. The blast for blowing the vessel is furnished by the blast-furnace blowing engines through a 36-in. diameter main, 1,365 ft. long. We mention this length on account of the unusual distance over which the blast is carried. The 18-in. branch blast mains on each vessel are provided with air-relief valves. The blast is delivered to the vessels at a pressure of 18 to 20 lb.

At this plant the iron coming from the blast furnaces has the following average analysis: Total carbon, 4.25; silicon, 1.00; sulphur, 0.05; phosphorus, 1.50; manganese, 0.20 per cent. The iron is delivered from the blast furnace to the mixer, and from the mixer to the converters, by means of hot-metal ladle cars, and charged into the mixer with overhead traveling crane, and into the converters by means of an electric ladle hoist.

From 2,600 to 2,800 lb. of burned lime is charged into the empty converter, after which is charged about 11 tons of fluid pig iron from the hot-metal mixer. After the metal has been blown, the slag is skimmed into a cast-iron box car made for the purpose, and the metal is then poured into the ladle car, which transports it to the open-hearth furnace, the entire blow consuming from 12 to 15 min. Under good average conditions the blown metal has the following analysis: Carbon, 0.03; phosphorus, 0.07; sulphur, 0.05 per cent.; manganese, none; and the slag is constituted as follows: Silica, 13.0 to 14; alumina, 1.0; lime, 48.0 to 51; magnesia, 2.0 to 4; phosphoric acid, 17.0 to 19; manganous oxide, 1.5; iron protoxide, 13.0 to 15. Five ladles of the blown metal are charged into the open-hearth furnace as they are delivered from the converters, but, prior to this, the open-hearth furnace has been charged with about 4,000 lb. of burned lime and 6 to 8 tons of molten iron, direct from the hot-metal mixer, the latter iron being depended upon to give sufficient carbon for the chemical reaction. Ten to twelve heats are made in No. 1 open-hearth furnace in 24 hr. One of the economies resulting from the practice of the basic duplex process is the revenue derived from the Bessemer slag after it has been ground and prepared for agricultural purposes.

### *The Bethlehem Steel Co.'s Plant*

In 1910 the Bethlehem Steel Co., desiring to increase and add flexibility to the output of its new Saucon plant and to render itself more independent with respect to the fluctuations of the scrap market, decided to adopt the duplex process, and, consequently, in 1911, a plant was installed. Rather than further burden its own organization, it let the contract to the Pennsylvania Engineering Works for the complete plant, which was designed, built, and turned over to it in operating condition. The

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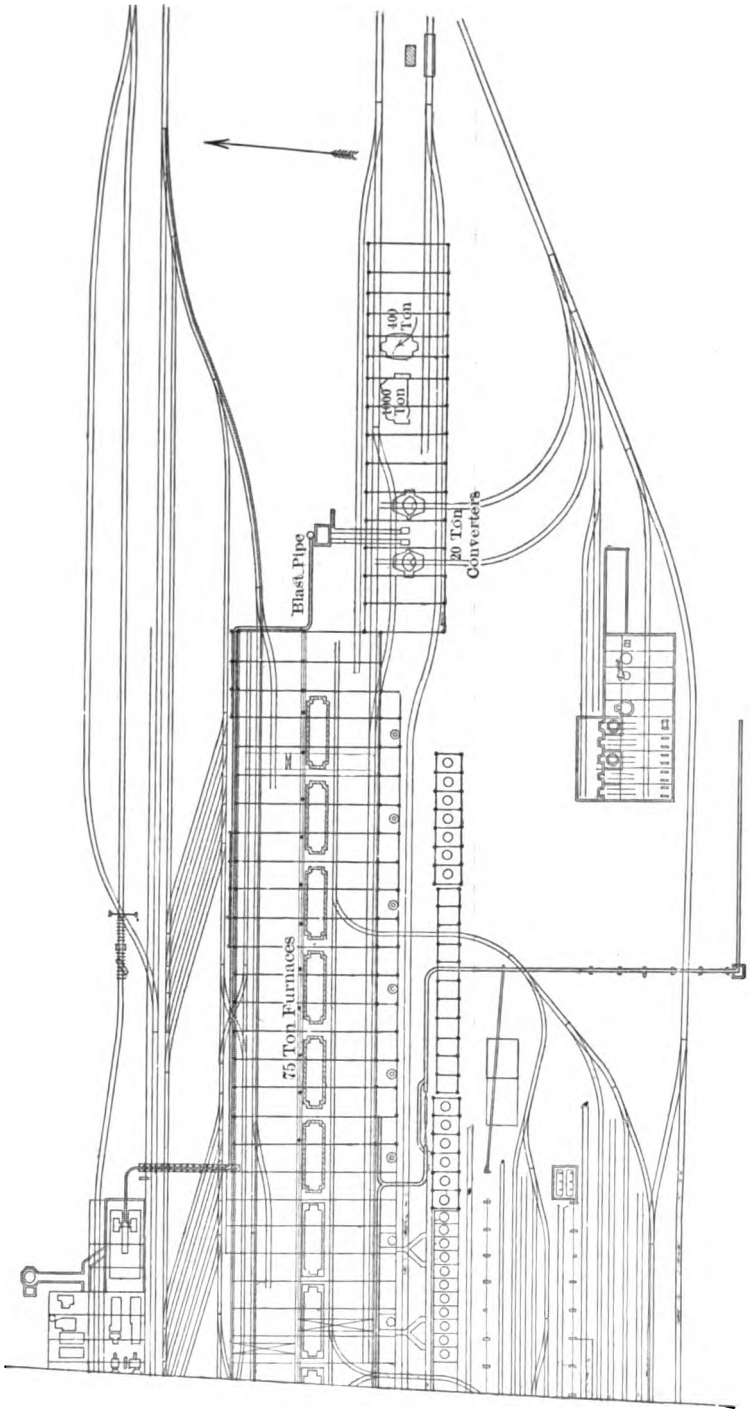


FIG. 3.—PLAN OF THE DUPLEX PLANT OF THE BETHLEHEM STEEL CO.

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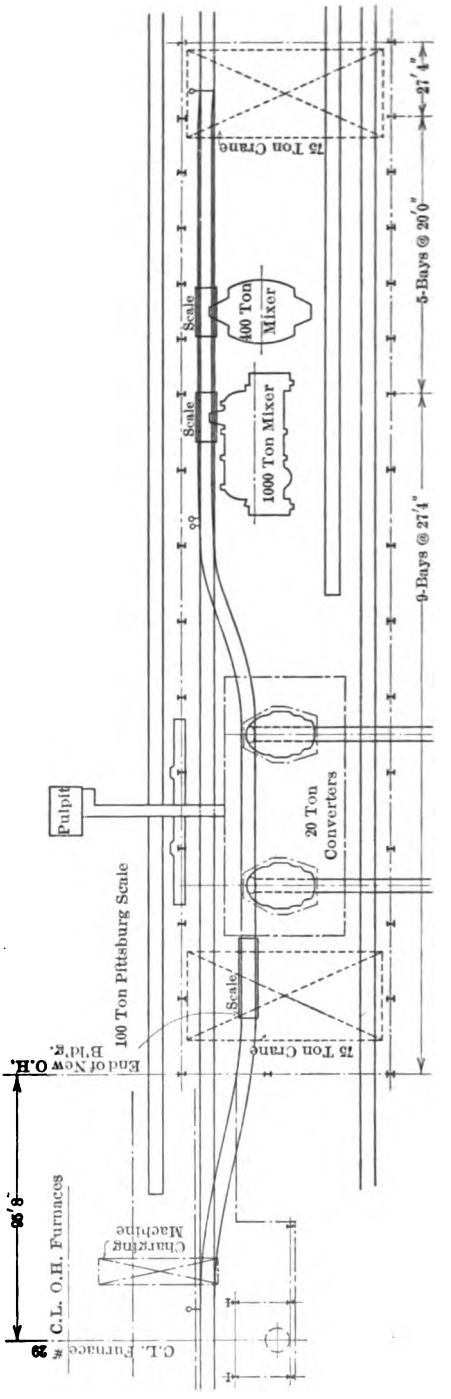


FIG. 4.—PLAN OF CONVERTER AND MIXER DEPARTMENTS, BETHLEHEM STEEL CO.

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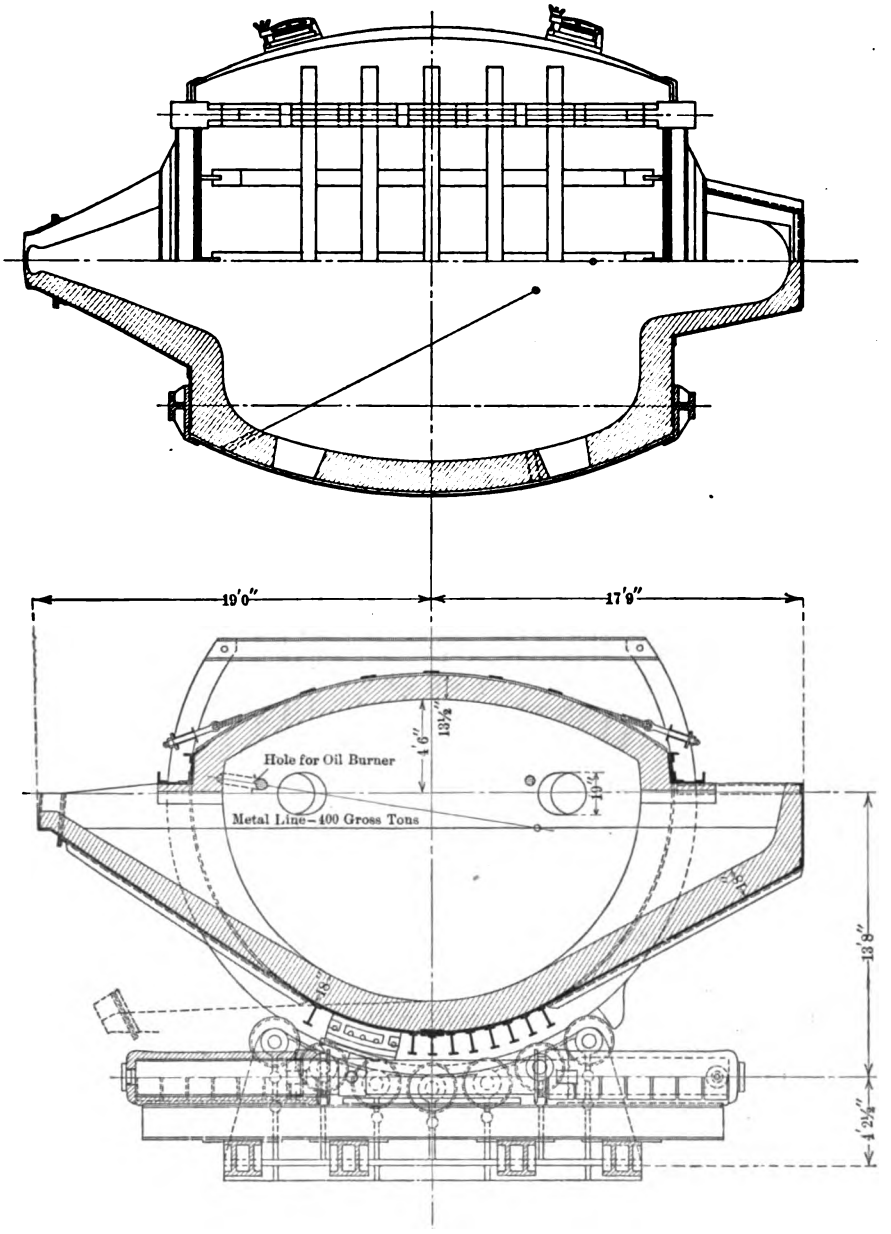


FIG. 5A.—400-TON HOT-METAL MIXER, BETHLEHEM STEEL CO.

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original Saucon Steel Works, which were put into operation in 1907, consisted of ten 60-ton stationary open-hearth furnaces and one 250-ton hot-metal mixer, a Gray universal structural mill for rolling wide-flange I-beams and H-column sections, a standard structural shape mill, and a rail mill. In 1913, the Bethlehem Steel Co. added to the open-hearth department of the Saucon Works, six 75-ton open-hearth furnaces of the stationary type, locating them on the east end of the original open-hearth plant, and added to the mixer capacity one 1,000-ton hot-metal mixer of what is known as the German type. The Bessemer converting plant occupies a position east of, adjacent to, and in line with, the open-hearth building, thus making it possible to

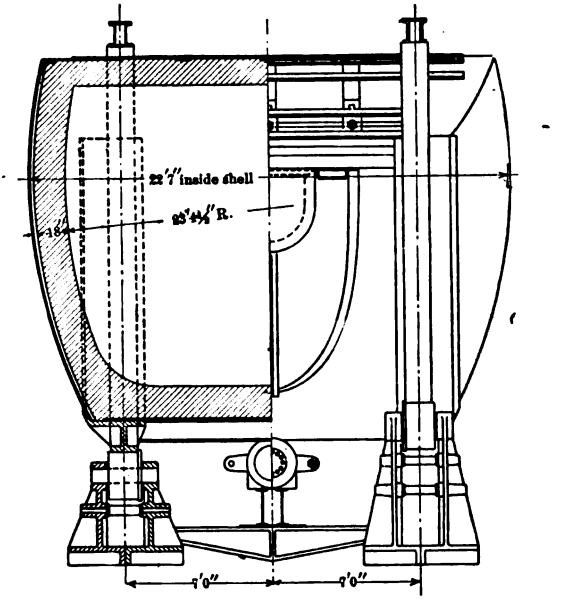


FIG. 5B.—400-TON HOT-METAL MIXER, BETHLEHEM STEEL CO.

connect the track for the transport of the blown metal from the converters to the open-hearth department directly with the track in the open-hearth building running parallel with the furnaces, but back of the charging-machine track. We might point out here that the location of the converters with respect to the open-hearth furnaces in this process is of the utmost importance, because the quick and easy transfer of the blown metal is one of the essential factors of success. The output of this department at present, operating on a straight open-hearth basis, is 70,000 to 75,000 tons of ingots monthly. As duplexing is practiced only part of the time, it cannot be said just what the output would be if the works were operated continuously under the duplexing method; but actual results have shown that for a given tonnage of steel there is a saving of about 65 per cent. in time.

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The Bessemer converting plant, the general arrangement of which is shown in Figs. 3 and 4, is contained in two buildings, one known as the mixer-converter building, where the hot-metal mixers and converters are housed, and the other as the bottom house, where the bottoms are made up, dried, ladles repaired, etc. The floors of these buildings are on the same level as the charging floor of the open-hearth building. The mixer-converter building is of steel, and of the usual heavy mill type construction. It is provided with two 75-ton capacity, 70-ft. span, traveling cranes for serving the hot-metal mixers and converters. The converters occupy the west end of the building nearest the open-hearth plant, while the hot-metal mixers occupy the east end.

There are two hot-metal mixers, one of 400 tons capacity, of the type indicated in Fig. 5, and one of 1,000 tons capacity, of what is known as the German type. While these mixers have a combined nominal capacity of 1,400 gross tons, their actual capacity is considerably in excess thereof, since the larger mixer will hold at least 1,200 gross tons of molten iron. The 400-ton mixer was originally built to operate hydraulically, but has since been converted to operate electrically. It is mounted on heavy, cast-iron rocker stands, which furnish tracks for the cast-iron rollers. The two cast-steel rockers serve the dual purpose of a cradle for supporting the plate work and a tire for revolving the mixer on the rollers. The shell, which is semi-cylindrical in form with bulging convex ends and domed roof, with receiving and pouring spouts on the straight portion of the cylinder, is made of 1-in. steel plates. The mixer is lined throughout, to well above the slag line, with 9-in. thick magnesite brick, backed up by 9 in. of good grade fire brick, while the roof has a 13½-in. lining of furnace-roof brick.

The 1,000-ton hot-metal mixer, constituting the other unit of this plant, is interesting on account of its large capacity as compared to former mixers in this country, 600 tons being the limit of capacity up to that time. At the time of making the contract for the building of the mixer. the Bethlehem Steel Co. stipulated that the builder should, before designing the mixer, send his engineers to Germany to investigate both the design and method of operation of the large mixers of that country, the result of which investigation was that the engineers came home and designed the present mixer after the German type, making such alterations and improvements as were found necessary to meet American conditions. (See Fig. 6.) The mixer consists of a cylindrical shell with spherical ends, It is provided with receiving spout on one side and pouring spout on the opposite side. The receiving spout, except when metal is being poured into the mixer, is sealed by a door, which is opened and closed by means by a 7½-h.p. electric motor. The pouring spout is closed by means of a number of flat brick arches held in steel stirrups, except a very small opening for the egress of the metal, thus making it possible to conserve

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from 90 to 96 per cent. of the original heat of the iron charged into the mixer.

A comparison might be made here between the two types of mixers: In the case of the 400-ton mixer it will be noted that the area of the surface of the metal is large compared with the depth and total volume of the metal, whereas, in the 1,000-ton mixer, the area of the surface of the metal is small as compared to the depth and total volume. This feature also

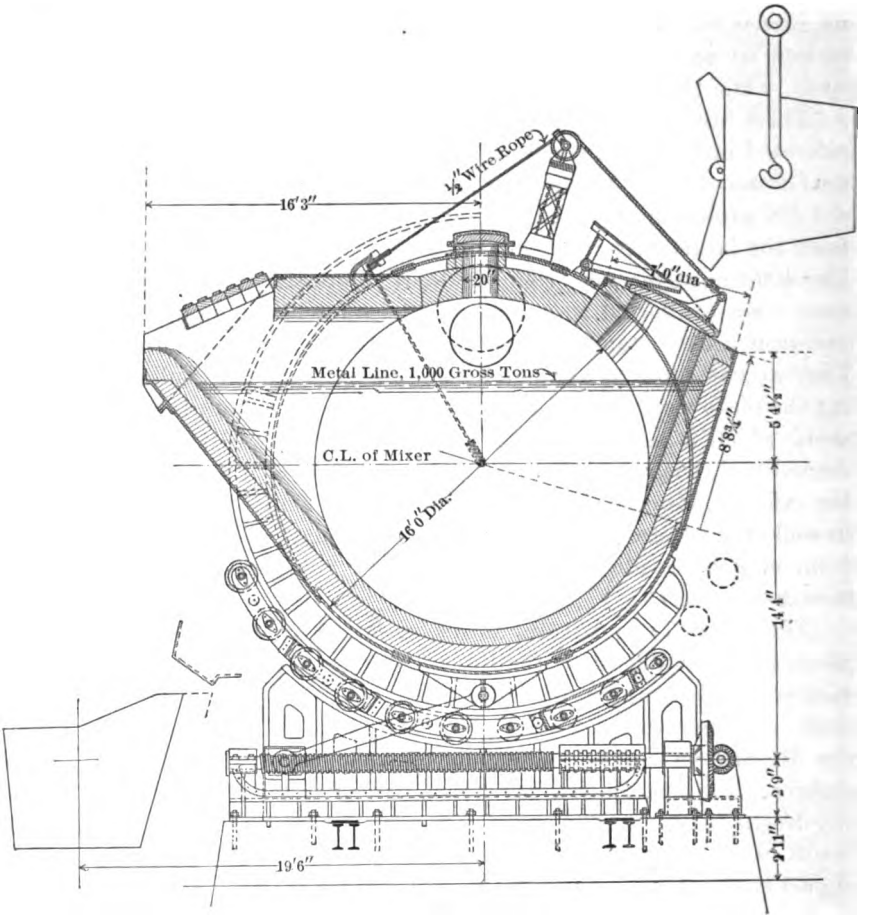


FIG. 6.—1,000-TON HOT-METAL MIXER, BETHLEHEM STEEL CO.

has its bearing on the conservation of heat. The 1,000-ton mixer is provided with a heating apparatus in which oil, producer, coke-oven, or blast-furnace gas is used. During the investigation of mixers in Germany it was found that, while most of the mixers were provided with a means of artificial heating, nevertheless the artificial heat was unnecessary except in one instance, and, in most cases, the ports for this heat entry were

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blanked off. At South Bethlehem it has been found to be of advantage, however, to use a small amount of artificial heat. The cylindrical shell of the 1,000-ton mixer is provided with four cast-steel bands, spaced uniformly between the two ends. The two outer bands encircle the cylinder about half way, while the two inner bands form a complete circle. These bands serve the dual purpose of reinforcing the shell and supplying tires on which the mixer rotates on the rollers. There are four groups of 11 cast-steel rollers, 16 in. in diameter with 16-in. tread. Each group of rollers is formed into a cradle by means of heavy side bars, with shafts extending through the side bars and rollers, thus insuring equal spacing of the rollers in the groups at all times. Two lines of heavy struts are provided between the cradles, to insure the action of all the groups as one unit. The weight of the mixer is transmitted directly from the rollers to four heavy cast-iron roller stands, thus producing true roller bearings, and minimizing the friction. It may be worthy of mention here that, with this type of mixer, the center of rotation coincides with the center of gravity, whereas, in the former type, it is necessary to lift the center of gravity about the center of rotation, from which it will easily be seen that the larger type of mixer requires, relatively, considerably less power to operate. In order to insure alignment of the rocker stands, and to tie the mixer more firmly to the foundations, two heavy I-beam girders are provided, extending longitudinally under the rocker stands the full length of the mixer. These girders are provided with shear plates, while the bases of the stands are provided with shear lugs. This feature is also of considerable advantage when lining up the mixer at the time of erection. The 1,000-ton mixer is operated by two 75-h.p. electric motors, either of which is capable of furnishing the required power. The motors are controlled by a magnetic switch-type controller, and are arranged so that they will operate in series during the pouring period and in parallel during the return of the mixer to its normal position, thus reducing the time for pouring and the return of the mixer to a minimum. The motors are placed one at either end of the mixer and are both connected to the line shaft, which is provided with a clutch adjacent to either motor, which may be disengaged in case of accident or for repairs. The 1,000-ton mixer has a surface lining 9 in. thick of magnesite fire brick extending well above the slag line, which is backed with 13½ in. of fire brick, which, with the packing between the lining and the shell, makes a total thickness of lining of about 2 ft. The extraordinary thickness of the lining also tends to conserve the heat of the mixer contents.

The converters, which are of 20 tons capacity, are spaced 40 ft. center to center, and are similar to those described at the plant of the Tennessee Coal, Iron & Railroad Co., with the exception that in this case it was thought desirable to contract the body of the vessel where it joins the bottom, in order to make the diameter of the vessel less, thus bringing



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the tuyères nearer to the side walls. This has a tendency to deepen the bath slightly for a given charge of metal; but, on the other hand, brings the bath more directly over the tuyère openings. The converters are operated by means of hydraulic cylinders through racks and pinions, similar to those of the Tennessee Coal, Iron & Railraod Co. The hydraulic cylinders are on a pressure line of 550 lb., which is provided with tank accumulator to balance the pressure. The converters are operated by means of Aiken valves, and the pressure lines throughout are made of double extra pipe. The blast for blowing the converters is furnished by Southwark horizontal cross-compound engine with barometric condenser, with steam cylinders 46 and 84 by 60 in. and air cylinders 84 by 60 in. The engine is designed to furnish 45,000 ft. of free air per minute, at 30 lb. pressure, and is located in the power house adjacent to the open-hearth

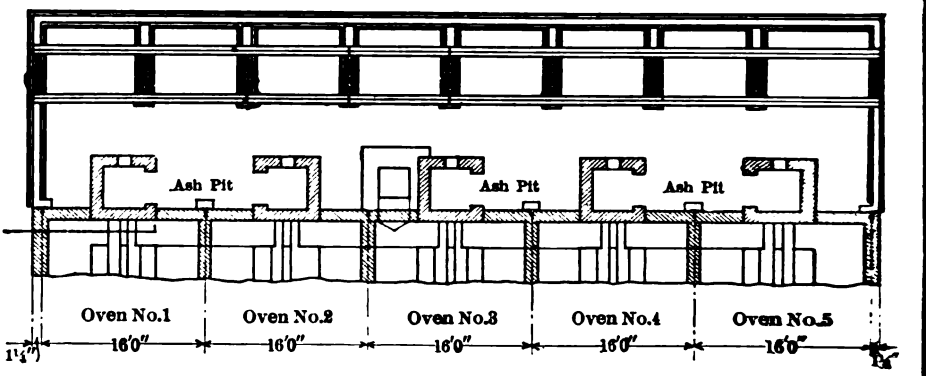


FIG. 7.—PLAN OF BOTTOM OVENS, BETHLEHEM STEEL CO.

plant. The blast is conducted to the converters through a 30-in. main about 500 ft. in length, which is provided with air-relief valve to prevent back pressure to engine. It is also provided with a tank air receiver near the converters to balance the pressure. The air is delivered at a pressure of 18 to 20 lb. from the receiver tank to each converter through an 18-in. air line provided with an 18-in. air valve operated from the pulpit. Heavy steel construction operating floors and platforms are provided about both the mixers and the converters. One 75-ton track scale is provided directly under the pouring spout of each mixer, and another between the converters and the open-hearth furnaces, on which the metal is weighed prior to charging into the converters and again after it has been blown, on its way from the converters to the open-hearth department.

The bottom house is located just south of the mixer-converter building. It has a 16-ft. leanto extending throughout the full length of the building, which is divded into ten equal bays. Five of these bays in the leanto are occupied by the five ovens for drying the bottoms. The ovens, shown in Figs. 7 and 8, have flue connection with a common draft stack, and are

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heated by firing from the rear with small anthacite coal, for which special grates are provided. Forced draft is furnished by an Eynon & Evans blower on each fire box. The ovens are equipped with Kinnear roller curtain doors and each one has a floor space of 14 by 16 ft. This house

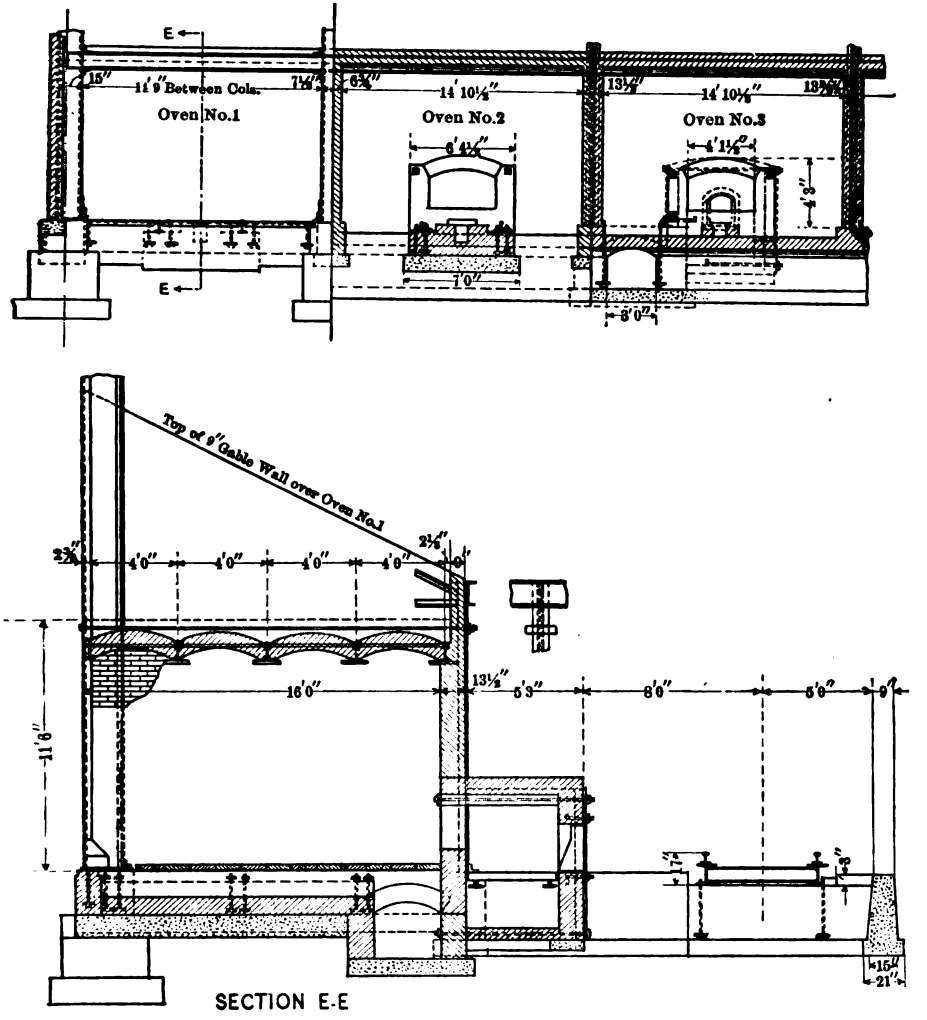


FIG. 8.—ELEVATION AND SECTION OF BOTTOM OVENS, BETHLEHEM STEEL CO.

contains a bottom pit, over which the bottoms are made up, a Blake crusher for crushing stone for linings, an 8-ft. wet pan, a 9-ft. dry pan, and the necessary bins and conveyor for handling the raw material. The machinery is driven by a 100-h.p. electric motor. There are also ten ladle rests for the repair of the linings of the steel ladles. For lifting

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the bottoms on and off the cars and bottom pit, handling the ladles, etc., there is provided a 25-ton electric traveling crane. The burned-out bottoms are brought from the converters, and the newly made up bottoms returned to the converters, on hydraulic jack cars.

The method of operation at the Saucon Works is practically as follows: The molten pig iron is brought from the blast furnaces situated at the Lehigh Works, 1½ miles distant, in trains of 40-ton capacity hot-metal cars drawn by the usual standard-gauge locomotive. Just before entering the mixer building, the metal is weighed on a 100-ton capacity track scale. It is then drawn into the building directly opposite the mixers and is poured into the mixers by means of the 75-ton overhead electric traveling crane. The metal is poured from the mixer into a 25-ton capacity ladle car, weighed on the 75-ton track scale placed in the floor directly under the pouring spout of each mixer, and then transported to the converter, where it is charged by means of the overhead traveling crane. The ladle is of special design with respect to the pouring spout, the object being to retain as much of the slag as possible in the ladle. For this purpose the spout is made so that the metal will pour through a narrow and rather deep opening. The slag is retained in the ladle for several charges and then dumped out. At some plants skimming is resorted to. This is especially true in Germany, where the ladles of metal are skimmed both before pouring into the mixer and before charging into the converter. After the metal has been blown in the converter, it is poured into 25-ton ladle cars (two of which are in constant use for the transfer of metal when duplexing), and transported by an electric locomotive over a standard-gauge track to the open-hearth furnaces. The hot metal is poured by the overhead traveling crane into the furnace, through a portable spout, which is placed in position by the charging machine at time of charging. Five of the ten 60-ton stationary open-hearth furnaces are used in the duplexing process; the remaining five 60-ton furnaces and the six new 75-ton furnaces are kept on the straight open-hearth practice making steel from the hot-metal mixer iron and scrap. Ordinarily three ladles of converter iron are charged into the open-hearth furnace, one after another, as rapidly as they can be blown. All of the metal is desiliconized in the converter and, of the three ladles of metal constituting the open-hearth furnace charge, the former two have practically all of the carbon eliminated, while, in the last one, about 2 per cent. of carbon is left, to bring about the reaction in the open-hearth furnace. The average open-hearth furnace charge is 40,000 lb. scrap, 15,000 lb. burnt lime, and 95,000 lb. of converter iron. The time required in the open-hearth furnace varies in general practice from 4 to 6 hr., although a single heat has been put through in 3½ hr. At times, recarbonizing is found necessary, and, for this purpose, there is in the open-hearth building a 250-ton hot-metal mixer employed as a receiver for the recarbonizing iron, which is of a special

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Bessemer quality made from low-phosphorus ores. This metal is poured from the mixer into ladles, and added to the bath in the open-hearth furnaces, as needed.

*The Pennsylvania Steel Co.'s Plant*

The steel-making department of this company, in which duplexing is carried on, consists of six 75-ton and two 200-ton open-hearth furnaces, two 20-ton Bessemer converters, one 300-ton and one 800-ton hot-metal mixer, and a bottom house equipped with the necessary crushing and grinding machinery, drying ovens, etc., for preparing the material, making up, and drying the bottoms. The plant was built in 1913, with the exception of the six 75-ton open-hearth furnaces and the 300-ton hot-metal mixer, which, previous to this time, were run on straight open-hearth practice. The contract for the 800-ton mixer and the two converters was let on April 19 and the converters blown in on the following December 1, making a record time of 7½ months for the construction of such a plant. As indicated in Fig. 9, the mixer-converter department is situated in the main open-hearth building on the charging-floor side. Extending round about both the mixers and the converters there is a spacious working platform of steel construction, 20 ft. above the ground floor and at the same elevation as the open-hearth charging floor. The molten pig iron, coming from the blast furnaces in trains of 45-ton hot-metal cars, is poured direct into the receiving spouts of the mixers without removing the ladles from the cars. The mixers pour into ladles resting on the ground floor, which are hoisted and charged into the converters by the overhead traveling crane. The converters in turn pour into ladles resting on the ground floor, which are hoisted to the open-hearth charging floor and transported by the overhead traveling crane to the open-hearth furnace to be charged.

The hot-metal mixers are of the semi-cylindrical, bulging-end type, and are tilted by electric motors operating through suitable trains of gears and screws. The control equipment is of the magnetic switch type. The converters, shown in Fig. 10, are placed 42 ft. center to center. They are duplicates of those of the Bethlehem Steel Co., with the exception of the overturning arrangement, which is electrical. Each converter is provided with two 100-h.p. motors controlled by magnetic switch-type controller. Either motor is capable of operating the vessel and there are shaft couplings provided so that either motor may be disengaged at any time. The power is transmitted from the motors through one gear reduction on the motors, a worm and worm wheel and pinion on the worm-wheel shaft to the driving gear on the converter trunnion. Mention is made here of the worm wheel, which has the wearing faces of the teeth lined with $\frac{5}{16}$ in. of babbitt metal, which reduces the friction, thus prolonging the life of the teeth. The arrangement of this plant deserves special attention, as

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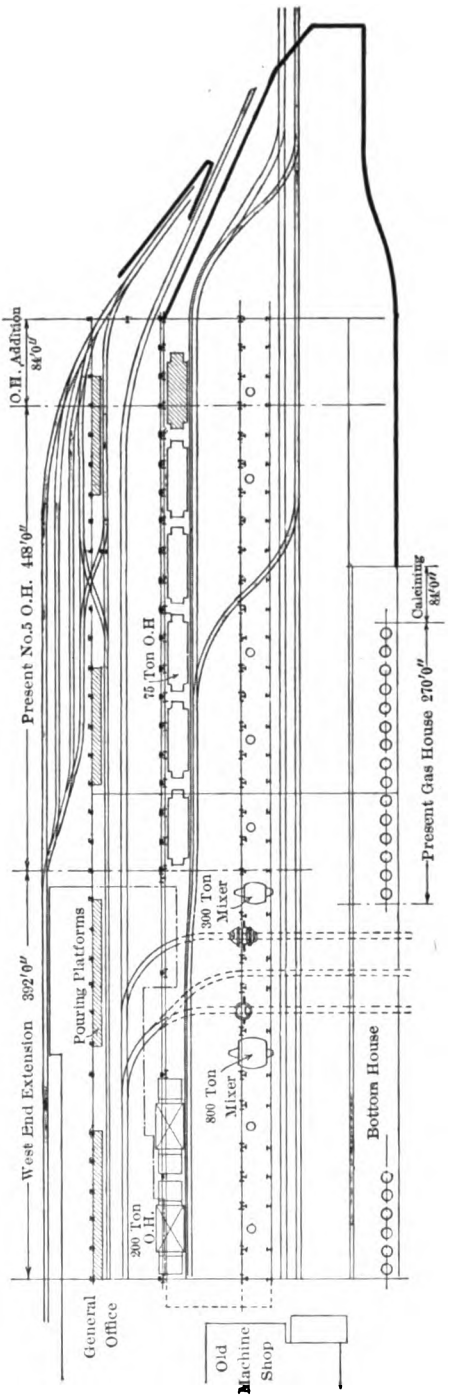


FIG. 9.—PLAN OF THE DUPLEX PLANT OF THE PENNSYLVANIA STEEL CO.

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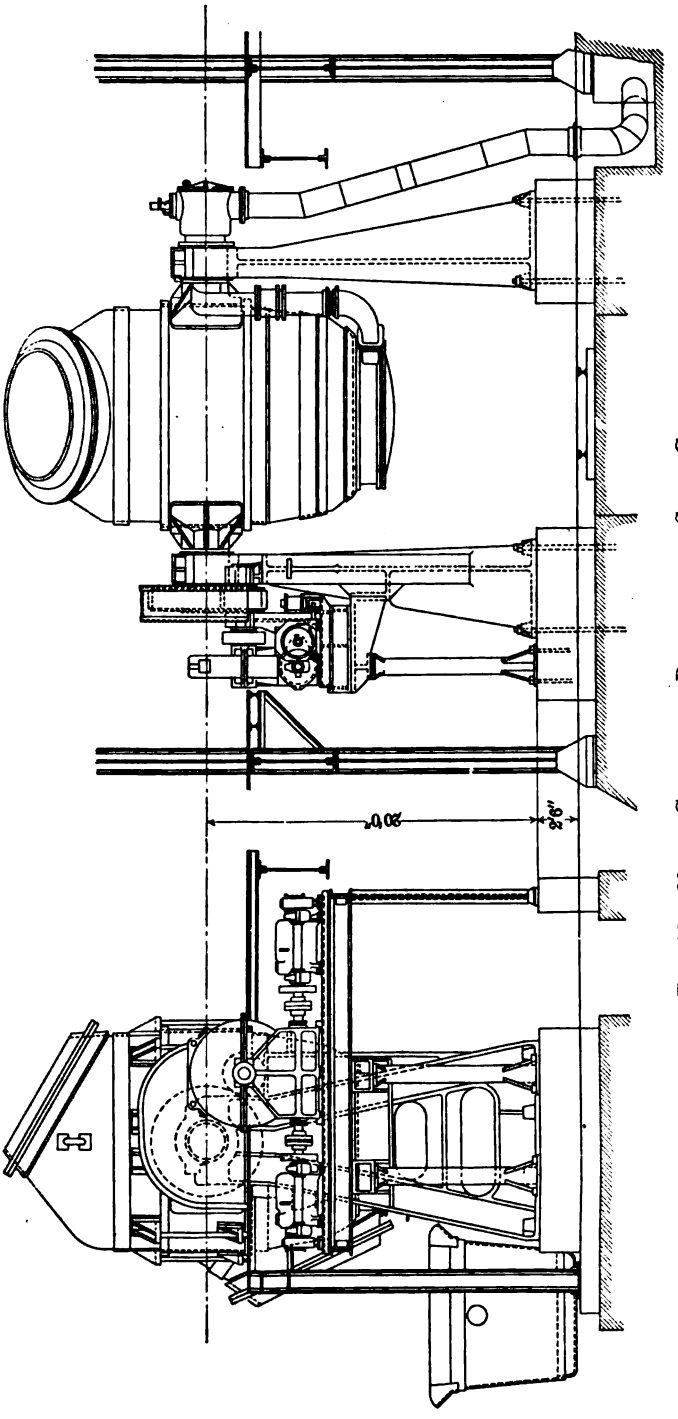


FIG. 10.—20-TON CONVERTER, PENNSYLVANIA STEEL CO.

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it represents one of the best conditions for economy and convenience of operation which has been attained up to the present time.

### *General Remarks*

The two 20-ton vessels as installed in the plants of the Tennessee Coal, Iron & Railroad Co., the Bethlehem Steel Co., the Jones & Laughlin Steel Co., and the Pennsylvania Steel Co., are capable of producing 100,000 tons of blown metal per month. The benefits derived from the duplex process for making steel have been enumerated in the reasons given for duplexing; but we may say in conclusion that the process has brought about the development of the Bessemer converting plant to a higher state of perfection and efficiency, has created a desire for larger and better hot-metal cars and mixers, and has aided those practicing it to solve some of their problems. Since the process has been practiced in this country for seven years only and is still in an imperfect state, it is hoped that some of the master minds working on it will succeed in bringing about a more perfect stage of development within the near succeeding years, and thus another step will have been taken toward the uplift of mankind.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Safety Movement in the Lake Superior Iron Region

BY EDWIN HIGGINS, PITTSBURG, PA.

(Pittsburg Meeting, October, 1914)

## INTRODUCTION

IT is the purpose of this paper to set forth the relation and functions of the various organizations and institutions engaged in the promotion of safety in the iron mines of the Lake Superior region; also to indicate the value of this work. Ten to 15 years ago there was practically no organized safety work; the accident rate was high, excessively so in some districts, and dangerous practices and conditions existed in many of the mines.

During recent years, however, a gradual change for the better has been made. The chief causes of this change have been (*a*) public opinion, which has set the stamp of disapproval on the disregard for human life; (*b*) certain State laws which have made the operators responsible in dollars and cents for injuries to workmen; and (*c*) the humanitarian attitude of many of the operators, who have always decried the great loss of life in the mines.

Today the Lake Superior region as a whole stands second to no other metal-mining district in the United States in its efforts to promote the welfare and safety of the miner. Dangerous practices in and about the mines are fast disappearing. The operators are ready and eager to adopt any expedient, rule, or device that holds forth a reasonable promise of reducing the hazards of the miner. Today the value of a mine captain or shift boss is reckoned, not alone on his ability to "get the ore," but also on his capacity for reducing accidents. While the progress made has been remarkable, there still remains much to be done, for the accident records of the Lake Superior iron mines, while lower than those of the chief metal-mining regions of the United States, still compare unfavorably with those of the metal mines of practically all foreign countries.

## ORGANIZATIONS ENGAGED IN SAFETY WORK

There are five organizations, or institutions, engaged in safety work in the Lake Superior region, viz.: The mining companies; county mine

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inspectors; co-operative range committees; Lake Superior Mining Institute; Federal Bureau of Mines.

While each of these bodies covers well-defined features of the work, their activities are correlated to a certain extent. The mining companies, primarily, are concerned with conditions in their respective mines, but they co-operate with and receive aid from the other agencies mentioned. The county mine inspectors, whose duty it is to see that the mines are operated with due regard to the State laws, are of great assistance to the mining companies in suggesting remedies for dangerous conditions or practices. The co-operative range committees, made up as they are of officials and employees of all companies operating within a given radius, are a benefit to the body of operators of their respective ranges. The committee of the Lake Superior Mining Institute on practices for the prevention of accidents, concerns itself chiefly with questions of safety of widespread interest; it of course has the co-operation of the other bodies interested. The Federal Bureau of Mines is chiefly active in training miners in the use and care of rescue apparatus and in first aid to the injured; also, it has conducted special investigations of certain problems having to do with safety and efficiency in the mines.

The unity of ultimate purpose and the strong co-operative spirit which have characterized the work of these five bodies have been important factors in the development of the conditions of today.

## The Work of the Mining Companies

The bulk of the cost of the safety work has been and is still borne by the mining companies. In the main, their work has been to provide protective devices in and about the mines, and to educate the miners, by means of rules and regulations and various other methods, so as to enable them to protect themselves from injury. These objects seem easy of accomplishment, but there are obstacles of various kinds continually arising to hinder progress toward the desired end. Some of the most serious factors, most of which still exist to a greater or less extent, are the prejudice of the old-time miner or boss toward safety regulations; carelessness and lack of interest in safety work on the part of the miners, and even the bosses; and at times the scarcity of labor, which necessitates the employment of less skilled and oftentimes ignorant men.

The greatest problem today is not to secure knowledge of how safety work should be conducted, and what protective devices to use, but how to get the miner to use these methods and devices. Probably not 10 per cent. of the miners, if subjected to an examination, would show even a passing knowledge of the contents of the books of rules and regulations. Safety devices, provided at great expense, are often found removed from their places, or disregarded entirely.

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The first problem of the mining companies was to provide for an organization to carry on the safety work; then to devise means of protecting the miner and of educating him and securing his co-operation in the prevention of accidents.

*Organization*

Practically every mining company in the district now has some individual, or organization, whose duties relate solely to safety work. The larger companies have well organized safety departments. The following brief outline of safety organizations will indicate the usual procedure in this connection:

1. The mines covered by this organization are large and are all situated on one of the iron ranges of Michigan. The safety department is in charge of an inspector and it is his duty to inspect all mines as frequently as possible and submit reports and recommendations to the manager. Periodical trips are made in and about the mines by a committee of mine foremen consisting of three members, each of whom is selected from a different mine. The inspector accompanies this committee and incorporates its recommendations in a report. Another committee, having similar duties, is made up of workmen. The activities of this committee, however, are limited to the mine from which it is selected. The members are changed after each inspection, so that in time all employees are given a chance to criticize conditions in and about the mine.

All accident reports and safety recommendations are considered by a committee of mine superintendents, the head mining captain, master mechanic, assistant auditor, secretary of the pension department, safety inspector, and the manager, who acts as an *ex-officio* member. This committee meets once a month and confirms or rejects safety recommendations.

In addition to the above committees, there is one more made up of three mine superintendents. This committee investigates all fatal accidents and makes a report thereon to the manager.

2. The following form of organization is employed by a company operating both large and small mines at various scattered points. The department is under an inspector, who, with the assistance of three experienced miners, inspects each mine of the company at least twice a week. After a mine is examined, a report, including any recommendations thought necessary, is sent to the safety inspector. The safety inspector in turn makes a weekly report to the superintendent, who looks after all recommendations having to do with upkeep. The safety inspector makes a monthly report in triplicate to the manager in which recommendations are submitted for approval. Such recommendations are made out to the head of the department concerned. When approved by the manager, one copy is returned to the safety inspector, to be kept by him

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until the indicated improvement is made. Two copies go to the superintendent, who keeps one and sends the other to the head of the department concerned. On the completion of the improvement, the head of the department sends the recommendation back to the superintendent, who then destroys his copy and sends the indorsement of completion to the safety inspector. The safety inspector destroys his record and files in its place the report showing that the improvement has been made. This report is in the form of a printed card with blank spaces filled in according to the needs.

All company bosses and first-aid men meet once every two months to discuss all accidents that have occurred during the previous two months. The subjects discussed at these meetings are safety, sanitation, first aid and welfare.

3. A similar organization to that described above is maintained by a company operating small groups of mines at scattered points. A chief inspector is in charge of the safety work at all the mines. The foremen's safety committee, consisting of four foremen from the mines of a certain district, works directly under the chief inspector. This committee makes a trip every three months through all mines of the district. Its personnel is changed after each inspection trip. The committee reports to the inspector, who, in turn, includes this report in his recommendations to the superintendent; a copy of the report also goes to the general manager.

4. This organization operates in connection with one large mine. Inasmuch as the organization was changed after the safety work was well under way, it may be well to point out the various steps in perfecting the organization. An engineer was placed in charge of a department of efficiency and safety. He first made a thorough study of conditions in the mine and determined the principal causes of injury to the men employed. Finding that the greatest number of accidents occurred from falls of rock and ore, and from men falling down unprotected places in the mine, timber inspection was doubled and every place in the mine where it was thought that there was a possible chance for a fall to occur was timbered. All open places were protected by means of doors or gates. This movement effected an immediate and marked falling off in the number of accidents from the causes mentioned.

Finally, three assistants were added to the department and each one of them was given a certain feature of the work to look after. This was necessary on account of the magnitude of the operation. The safety and efficiency work then developed into daily inspection trips by all the members of the department. Reports were made by them to the head of the department, who considered recommendations made and obtained immediate action thereon through consultation with the manager. Lately these daily inspection trips have been abandoned; the members of the organization now average two or three trips a week through the mine.

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Daily meetings, attended by the manager, superintendent, head of the efficiency and safety department, and mine captains, are held. Here all matters pertaining to efficiency and safety are discussed. As these meetings are held in the morning, it is possible to hear the reports of the shift bosses to the mine captain. In this way daily happenings and conditions in the mine receive prompt attention.

### *Protective Methods and Devices*

Under the head of protective methods and devices it may be said, in general, that the work proceeds along the following lines:

A study of mining and timbering methods with a view to greater safety.

A study of safe mtthods in every department of mine work.

The protection of dangerous places in the mine.

Protective coverings for all exposed parts of machinery.

Installation of safety devices in and about the mine.

The provision of such tools and appliances as will result in the maximum safety to the employees.

Installation of devices for protection against fire.

The inspection of all working places, shafts, and machinery at stated intervals.

The method of carrying on the inspection work is indicated in the descriptions of the various types of organizations. Nearly all of the protective methods and devices are suggested through information gained on the inspection trips.

### *Welfare and Educational Work*

It has long been recognized that various measures looking to the welfare of workmen are most desirable from many standpoints. In the study of safety work it has developed that any line of work that will serve to secure the co-operation and confidence of employees is of the greatest value in promoting safety. Without the co-operation of the workmen safety work cannot advance, and in order to gain this co-operation it is necessary first to secure the confidence of the men. Indeed, many students of safety believe that the winning of this confidence and co-operation constitutes almost the entire safety problem. Lake Superior mining companies are spending as much, if not more, money in welfare and educational work than on the actual installation of safety devices.

Among the activities of several of the large companies looking to the welfare of the miner may be mentioned the following:

The provision of a pension fund for workmen who have grown old in the service of the company; erection of club houses where the workmen may spend their idle hours; the building of model towns and houses for

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the use of workmen; the offering of cash prizes for the best-kept premises, lawns, and flower gardens; the building of expensive and commodious dry or change houses. Practically every iron mine in the Lake Superior region is provided with a dry or change house; some of these are models of cleanliness and perfection in other details.

Among the more important educational features designed to secure the cooperation of the miner, may be mentioned the following:

*Rules and Regulations.*—Practically all of the large companies issue to their workmen books of rules and regulations. In some cases these books are printed in from eight to ten different languages. In general, they cover all departments of mine work. These rule books in the past have contained many regulations that were not enforced, but of recent years there has been a noticeable tightening in this respect and the tendency now is to eliminate many of the useless rules and to be more strict in the enforcement of those retained. There is still room for improvement, however, in the matter of enforcing rules.

*Cash Bonuses.*—The matter of cash bonuses to bosses and others for the prevention of accidents has not, in the past, been given serious consideration. During the last year, however, the managements of several companies have come to the belief that an equitable system of cash bonuses will be of value in reducing accidents. There is only one company that has really put the system into operation. While the method used by this company cannot be termed strictly a bonus system, it is in principle the same thing. By paying to shift bosses salaries that were larger than those paid by other operators, but insisting that these shift bosses be safety enthusiasts, this company has conducted its operations with a minimum percentage of accidents for the district. One large company is preparing to adopt a system of paying large cash prizes to the shift bosses turning in the best records as to the number of men killed or injured while working under their supervision.

*Publicity of Accident Records.*—A method of attracting the attention of the workmen to the hazards of their employment, which has been adopted to a certain extent, is to post placards at various points in and about the mine calling attention to all serious accidents that happen, and pointing out how they might have been prevented. In some cases sketches and photographs are used as illustrations with these placards. At some mines the records of the different shift bosses, as to the number of accidents that happen to the men working under them, are posted in conspicuous places.

*Pay for Safety Suggestions.*—It is the general practice in the district to offer cash rewards for suggestions from workmen that may lead to safer working conditions.

*First-Aid and Rescue Instruction.*—Through the activities of the mining companies, in co-operation with the Bureau of Mines, miners have been

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trained in first-aid and rescue work at practically every mine in the district. This work has not been limited to three or four men at a mine, but in many cases has been continued so that in some cases as many as 20 per cent. of the employees have received first-aid training. Training in rescue work, however, has usually been limited to from five to ten men at a mine, or group of small mines.

### Work of the County Mine Inspectors

The various counties of the Lake Superior district in which mines are situated, are each provided with a mine inspector. In some cases this inspector is allowed to employ such assistants as may be necessary in his work. The various counties require that every mine shall be inspected at stated periods. The principal duty of the county mine inspector is to see that the mines are operated in accordance with the laws of the State. He is empowered with authority to cause the closing down of a mine in case the management refuses to comply with his demands. In reality, these inspectors have rendered services in excess of what is demanded of them by law. Through the study of conditions in the mines they gain information that is of common value; such information is disseminated to mine operators in the form of suggestions for the promotion of safety. One inspector has gone so far as to make and keep records of safety devices and methods. These he has communicated to all the operators in his county through the medium of circular letters and blueprints. The efforts of these county mine inspectors and their assistants have gone far toward promoting safety in the mines of the Lake Superior region.

### Work of the Co-operative Range Committees

There have been organized on the various iron ranges what may be termed co-operative range committees. During the year 1913, five such organizations were perfected. It is believed, however, that only three of them are now holding regular meetings and doing efficient work. These committees are made up of mine superintendents, mine captains, shift bosses, safety inspectors, men in charge of first-aid and rescue work and other mine employees interested in safety work. The committees have the backing of the management of the mines, which stand the bulk of the expense of carrying on their work. The organizations are made up of representatives of practically all companies within a given radius. For instance, the Gogebic Range Mining Association includes in its membership representatives from all companies operating on the Gogebic range. The purposes of this organization, as set forth in its by-laws, are to promote social intercourse and the interchange of ideas on all subjects of mining interest, for the mutual benefit of its members; and to perpetuate

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efficiency, welfare, safety, mine-rescue work and first aid to the injured in and about the mines. These purposes are accomplished by (*a*) social meetings; (*b*) remarks, discussions, and the presentation of papers by members of the organization at different times; (*c*) occasional visits to the different mines, plants and properties upon invitation of the management of same; and (*d*) occasional competitive meets for crews trained in mine-rescue work and first aid to the injured.

The greatest good done by these committees has been in disseminating, for the benefit of all the mining companies, information relating to safety, efficiency and other mine work. They have developed in some cases into organizations representing the needs and wishes of entire communities. Their opportunity for the promotion of the general welfare is unlimited.

## Work of the Lake Superior Mining Institute

The Lake Superior Mining Institute some years ago established a committee on practices for the prevention of accidents. This committee holds special and stated meetings at which are considered important problems relating to safety and mine operation. Among important subjects upon which recommendations have been made may be mentioned the uniformity of mine-accident reports. Under present conditions the mining companies make reports of accidents to several different organizations, all of which require different classifications of accidents. The work thus entailed is enormous. By providing a uniform type of report this undue work may be eliminated. Furthermore, reports of the county mine inspectors, although satisfactory as far as the needs of each county are concerned, are made in such form that it is impossible to make intelligent comparisons of records of the various counties. An effort will be made, through this committee, to standardize all reports.

In addition to other important considerations, a study is now being made of mine rules and regulations, with a view to eliminating unnecessary regulations and including others that appear to be of paramount importance.

This committee recently considered and made recommendations that led to the holding of a first-aid contest at Ishpeming, Mich., during August, 1914. This meet was attended by teams from all over the Lake Superior region.

## Work of the Federal Bureau of Mines

The Federal Bureau of Mines has headquarters at Ironwood, Mich., established in November, 1912. It has recently acquired, through lease, a small tract of ground on the right of way of the Chicago & Northwestern Railway, and a spur has been built thereon for the accommodation of the rescue car. Arrangements have been practically perfected under which

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the operators of the Gogebic range will erect a building containing the necessary office room and housing for the rescue car.

The Bureau's representatives in the district have comprised a district engineer, a foreman miner, and a first-aid miner. The rescue car has been active in training the miners of the entire region in first aid to the injured and rescue work. It has not, however, been continuously engaged in this work owing to lack of available funds. Up to the present time there have been trained in the entire district a total of approximately 700 men in first aid to the injured, and 400 men in the use of oxygen breathing apparatus. These men have trained others and it is estimated that there is now a total of 2,000 men in the district who have received training in first aid to the injured, and 1,000 men who have received training in the use of oxygen breathing apparatus.

In addition to this work of training, the district engineer has visited and examined a large proportion of the mines of the region. These examinations were followed in some cases, at the request of the management, by recommendations for increasing safety in the mines. Special investigations undertaken include those having to do with ventilation, mine fires, organization and conduct of safety work, the use of mine sign boards, and hoisting signals. Papers have been written on these subjects for publication by the Bureau and by various mining institutes. Other educational work carried on comprised illustrated lectures to the miners in and about the mines visited by the rescue car. The district engineer brought about the organization of the co-operative range committees mentioned on preceding pages.

The activities of the Bureau of Mines have doubtless been instrumental in furthering the work of safety. Its chief function has been in stimulating activity in safety, rescue, and first-aid work. The moral effect of the presence of the car and its attendants has served to call the attention of the miner to the fact that the subject of safety and first aid is of sufficient interest and importance to cause the government to take an active hand in the work. The Bureau of Mines' representatives have worked in co-operation with the operators and everyone else engaged in safety work in the region, and have been favored with most courteous treatment and co-operation in all their efforts. It might be added here that the writer has, for the past two years, served as district engineer of the Bureau in the Lake Superior region.

## Results of the Safety Movement

It is not possible at this time to prepare a statement that will indicate the full measure of benefit derived from the practice of safety work in the Lake Superior region. The safety movement is practically in its infancy, and a period of years must elapse before any fair estimate can be made of



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the actual good resulting from it. That all classes of accidents are steadily decreasing is shown by tables submitted herewith; the next few years should show a proportionate or even greater decrease.

Inquiries as to the reduction of accidents through safety work directed to all the operators of the district were answered, in a majority of cases, by the statement that records are not yet available; there were many indefinite replies stating that accidents had decreased; no replies were received stating that accidents had increased. Personal inquiry by the writer brought forth statements from the most important operators indicating that safety work has caused a material reduction in accidents, and that they are all desirous of continuing the work. In many instances a great deal of enthusiasm was displayed. An interesting statement regarding fatal injuries sustained by employees was received from the management of a large iron mine. It may be remarked that this company began active safety work early in 1912. The statement appears in Table I.

TABLE I.—*Record of Fatal Accidents in an Iron Mine*

| Year | Total Number of Men Employed per Day | Number of Fatal Accidents per Year | Fatal Accidents per Year per 1,000 Men Employed |
|---|---|---|---|
| 1889 | 185 | 2 | 10.81 |
| 1892 | 224 | 2 | 8.93 |
| 1893 | 259 | 1 | 3.86 |
| 1898 | 312 | 11 | 35.26 |
| 1905 | 372 | 3 | 8.07 |
| 1906 | 719 | 16 | 22.25 |
| 1907 | 731 | 5 | 6.84 |
| 1908 | 874 | 3 | 3.43 |
| 1909 | 1,050 | 8 | 7.62 |
| 1910 | 1,317 | 12 | 9.11 |
| 1911 | 1,235 | 6 | 4.86 |
| 1912 | 729 | 5 | 6.86 |
| 1913 | 1,548 | 5 | 3.23 |

No accurate records exist of accidents in the Lake Superior mines for a period of years. The Bureau of Mines only began the collection of accident statistics in metal mines in 1911. However, a compilation showing the fatal accidents for ten years previous to 1911 has been made by O. C. Davidson, General Superintendent, Oliver Iron Mining Co., Iron Mountain, Mich. Mr. Davidson reviewed the county mine inspectors' reports from Sept. 30, 1901, to Sept. 30, 1911. With the data thus obtained, and tonnages based on statements of shipments published by the *Iron Trade Review*, he was able to prepare this interesting statement. The compilation is submitted herewith as Table II. Attention is directed to the low death rate indicated in Dickinson county and the high rate in Iron county.

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The number of fatal accidents and other data for the years 1911 and 1912 are shown in Table III. It may be noted from this table that Minnesota stands first and Michigan second in the column showing the number of men killed per thousand employed. The other States included in this table comprise the chief metal-mining States of the country. The figures for Michigan and Minnesota serve the further purpose of showing the

TABLE II.—*Summary of Fatal Accidents in Michigan Iron Mines from Sept.* 30, 1901, *to Sept.* 30, 1911

| County | Tons of Ore Mined 10 Years | Total Number of Employees 10 Years | Number of Fatal Accidents Reported | Tons of Ore Produced per Fatal Accident | Fatal Accidents per 1,000 Men Employed |
|---|---|---|---|---|---|
| Dickinson | 22,601,474 | 31,836 | 84 | 269,065 | 2.638 |
| Marquette | 36,721,389 | 57,161 | 248 | 148,070 | 4.339 |
| Gogebic | 29,191,952 | 42,471 | 226 | 129,168 | 5.321 |
| Iron | 17,986,380 | 20,962 | 158 | 113,838 | 7.537 |
| | 106,501,195 | 152,430 | 716 | 148,745 | 4.697 |

TABLE III.—*Fatal Accidents in Various Metal-Mining States During the Calendar Years* 1911 *and* 1912

| State | Number of Operators Reporting. | | Total Number of Employees | | Number Killed | | Killed per 1,000 Employed | |
|---|---|---|---|---|---|---|---|---|
| | 1911 | 1912 | 1911 | 1912 | 1911 | 1912 | 1911 | 1912 |
| Alabama | 25 | 20 | 4,101 | 4,827 | 10 | 33 | 2.44 | 6.84 |
| Arizona | 352 | 479 | 12,768 | 15,591 | 70 | 67 | 5.48 | 4.30 |
| California | 855 | 1,048 | 10,877 | 10,312 | 38 | 40 | 3.49 | 3.88 |
| Colorado | 660 | 624 | 10,404 | 8,892 | 43 | 48 | 4.13 | 5.40 |
| Idaho | 513 | 639 | 4,801 | 6,229 | 23 | 29 | 4.79 | 4.66 |
| Michigan[a] | 74 | 79 | 31,584 | 29,469 | 134 | 96 | 4.24 | 3.26 |
| Minnesota[b] | 40 | 43 | 16,548 | 16,559 | 76 | 50 | 4.59 | 3.02 |
| Wisconsin[c] | 8 | 11 | 1,157 | 2,338 | 2 | 9 | 1.73 | 3.85 |
| Montana | 332 | 405 | 13,346 | 13,340 | 62 | 50 | 4.65 | 3.75 |
| Nevada | 472 | 554 | 6,210 | 7,547 | 50 | 34 | 8.05 | 4.51 |
| Utah | 295 | 336 | 7,710 | 8,458 | 49 | 41 | 6.36 | 4.85 |

[a] In copper mines, 14,893 men employed, 44 killed, 2.95 killed per 1,000 employed. In iron mines, 14,378 men employed, 52 killed, 3.62 killed per 1,000 employed.

[b] All iron mines.

[c] All iron mines.

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decrease in fatal accidents as compared with the 10 years indicated in Table II. These figures were obtained from *Bureau of Mines Technical Paper No.* 61, metal Mine Accidents in the United States During the Calendar Year 1912, by A. H. Fay. Figures for 1913 are not yet available. From county mine inspectors' reports, however, it is probable that the accident rate for the Lake Superior region during 1913 will show a marked decrease.

In summing up the beneficial results of the safety movement, it may be said that by far the most valuable accomplishment has been the reduction in the number of deaths, and serious and permanent injuries. This, of course, has been brought about through the improved working conditions in the mines, more vigorous inspection, and the reduction of the hazards to the worker through the use of various safety devices. Bodily suffering has been reduced, the earning power and efficiency of the worker has been increased, and mental suffering and hardship on the part of widows, orphans and other dependents has been lessened.

Not only have beneficial results been forthcoming from a humanitarian standpoint, but also from a financial standpoint. When a miner is injured the money that he can contribute toward the support of his dependents is curtailed in proportion to the seriousness of his injury. If the miner is killed the support of his dependents devolves upon others, thus giving them a double burden to bear. The employer also sustains financial loss, both in hospital expenses and in the payment of compensation. On the Marquette range, when a miner is killed, it has long been a custom for the entire mine force to cease work until the victim of the accident has been buried. The loss caused in this way amounts to approximately $2.30 per day per man involved, and an average of $500 in fixed charges to the company for every fatal accident.

Another source of financial loss is that sustained by the tax payers for the maintenance of courts for the trial of damage suits. Investigation discloses that in one Minnesota county $75,000 per annum has been spent in this manner.

In addition to the reduction in the loss of life, and the saving in money to both the miner and the operator, other benefits have developed. There appears to be a better understanding between employer and employee and the miners are beginning to realize that the safety work is being done for their benefit. Throughout the district a strong spirit of co-operation is noticeable, not only between the miner and employer, but between the officials of the various mining companies and mining districts. Information regarding safety and efficiency work is exchanged between operators with the utmost freedom, and there is hardly a mine in the region that is not open for inspection as far as safety devices and methods are concerned.

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## Cost of Safety Work

It has been impossible to arrive at even an approximate estimate of the amount of money spent on safety work in the Lake Superior district. Inquiries directed to all of the companies by the Bureau of Mines in 1913 were in a large majority of cases answered by the statement that no record has been kept of the money spent. Of course, many of the companies that have separate safety departments have kept records, but even some of these are incomplete in that they do not include money spent before the completion of the safety organization. Accurate figures should be obtained in the future, however, as safety work is becoming more and more a definite part of mine operation.

Perhaps an extreme case in the matter of expenditures in safety work is reported in the following statement taken from *Bureau of Mines Technical Paper No.* 30, "Mine-Accident Prevention at Lake Superior Iron Mines, by Dwight E. Woodbridge:

*Cost of Safety Devices Installed by One Lake Superior Iron Mining Company in* 1911.

| | |
|---|---|
| Menominee range, Michigan and Wisconsin | $8,196 |
| Gogebic range, Michigan and Wisconsin | 15,733 |
| Marquette range, Michigan | 19,937 |
| Vermillion range, Minnesota | 11,327 |
| Baraboo district, Wisconsin | 222 |
| Mesabi range, Minnesota: | |
| Hibbing district | 18,770 |
| Chisholm district | 11,142 |
| Adams district | 8,001 |
| Fayal district | 8,525 |
| Virginia district | 14,815 |
| Canisteo district | 21,152 |
| | $137,820 |

Table IV, showing expenditures per year of various companies for safety work, will serve to throw some light on this subject, although the figures are extremely variable. It may be noted that the figures in this table account for only 5,480 men, whereas there were employed in the iron mines of Michigan, Minnesota, and Wisconsin, during 1912, a total of 33,275 men. From this table it appears that the lowest sum spent per year, per man employed, was $1; while the highest is more than $48. The average for the whole is $10.10 per year per man employed, which is probably a high average for the region.

One company, employing over 2,000 men, spends approximately $6 per year per man employed. This company has a well-organized and efficient safety system. Another large company spends slightly more than this. In this connection it must be considered that in the beginning,

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TABLE IV—*Expenditures during* 1912 *for Safety Work of Some Lake Superior Iron Mining Companies*

| Annual Production, Long Tons | Number of Men Employed Daily | Amount Expended |
|---|---|---|
| 237,358 | 297 | $5,075 |
| 225,000 | 265 | 2,000 |
| 75,000 | 130 | 1,000 |
| 350,000 | 110 | 3,000 |
| 400,000 | 150 | 3,000 |
| 250,000 | 310 | 15,000 |
| 28,100 | 40 | 230 |
| 169,708 | 154 | 885 |
| 210,068 | 158 | 480 |
| 155,000 | 110 | 903 |
| 128,465 | 120 | 4,850 |
| 312,000 | 85 | 1,000 |
| 1,250,000 | 1,000 | 1,000 |
| 2,033,242 | 2,178 | 13,715 |
| 3,700 | 38 | 311 |
| 124,000 | 175 | 1,500 |
| 102,400 | 160 | 1,407 |
| | 5,480 | $55,356 |

when safety devices are being installed, the expense is greater than in succeeding years. It is probable that a fair average for maintaining safety work, after the preliminary work has been done, would be in the neighborhood of $5 per year per man employed. During the year in which safety devices are installed, if an elaborate system be adopted, the cost may run as high as $10 or more per man.

## DOES SAFETY WORK PAY?

Answering the question as to whether or not safety work pays, one may say without hesitation that it pays enormously from a humanitarian standpoint. There is no argument here. The records of many coal companies and other organizations which have practiced safety work for a number of years show undoubtedly that it pays also from a financial standpoint. As to figures on this phase of the question in the Lake Superior district, nothing accurate is possible of compilation. Without the cost of the work as a basis, acceptable figures cannot be submitted.

However, there is a method of arriving at approximate figures, based upon actual conditions in the Lake Superior region. Before going into this, it seems proper to submit some figures regarding the compensation

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that must be paid to miners for various classes of injuries. The States of Michigan, Minnesota, and Wisconsin all have in force a workman's compensation act. The following information is from the Public Acts of the State of Michigan, the provisions of which differ little from those of Wisconsin and Minnesota.

In case of a fatal injury to an employee, the employer must pay to the dependents of the injured a weekly sum equal to one-half his average weekly wages, but not more than $10 or less than $4 a week, for a period of 300 weeks from the date of the injury.

In case of permanent disability resulting from injury, the employer shall pay to the injured a weekly compensation equal to one-half his average weekly wages, but not more than $10 or less than $4; in no case shall the period covered by such compensation be greater than 500 weeks, nor shall the total amount of such compensation exceed $4,000.

Thus it may be seen that a permanent injury may cost the company more than a fatal injury. For injuries resulting in temporary disability, the injured receives a weekly compensation equal to one-half the difference between his average weekly wages before the injury and the average weekly wages which he is able to earn thereafter, but not more than $10 a week; and in no case shall the period covered by such compensation be greater than 300 weeks from the date of injury.

The above provisions are followed by a long list of payments to be made to the injured in case of the loss of a finger, a hand, a foot, an eye, etc.

In order to show what saving may be effected through the practice of safety work, let us assume an iron mine employing 1,000 men per day, in which no money is spent for safety work. Past experience has pointed out that a mine of this size, making no attempt to prevent accidents, may easily make the following yearly accident record:

| | |
|---|---|
| Number of men killed | 6 |
| Number of men seriously injured | 40 |
| Number of men slightly injured | 250 |

A serious injury may be considered as one that incapacitates the workman for more than 20 days; a slight injury one that incapacitates him for less than 20 days. The total cost in compensation for this sum of accidents, based upon the compensation stated above, may be as follows:

| | |
|---|---|
| 6 men killed, at $2,500 each | $15,000 |
| 2 men permanently injured, at $3,000 each | 6,000 |
| 13 men, average disability 20 weeks, at $7 per week | 1,820 |
| 25 men, average disability 8 weeks, at $7 per week | 1,400 |
| 250 men slightly injured, average disability 1 week, at $7 per week | 1,750 |
| Legal fees, hospital and other casualty expenses | 15,000 |
| Total | $40,970 |

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This tabulation does not include payments for the loss of hands, feet, etc. In total amount it is below many records that have been noted by the writer.

Now, let us suppose that this company had practiced safety work, and that the death rate was three men killed per 1,000 employed (approximately the Minnesota 1912 rate) and the injuries proportionately lower. This would mean a reduction of one-half, or a saving of approximately $20,000.

Of course the safety work will cost something, but even if it amounted to $10 per year per man employed, or $10,000 in this case, there would still be a balance of $10,000 saved.

## Conclusions

It has not been the intention in this paper to convey the idea that the Lake Superior iron ranges comprise the only metal-mining region in the United States where efficient safety work is done. It is acknowledged that there are individual mines elsewhere that can show accident records just as good, and possibly better than those of some of the Lake Superior mines; also that the safety movement is gaining ground throughout the various metal-mining districts of the United States. The operating companies of the Lake Superior region are for the most part large and strong financially and they can well afford to lead the way in work of this nature. In the Western metal-mining States there is a much larger proportion of small operations and prospects, the owners or lessees of which cannot afford to go to great lengths in the matter of safety.

It is hoped that all mine operators, from the insignificant prospector to the wealthy magnate, will eventually recognize the value of safety work. Sufficient records are now available, both from coal- and metal-mining districts, to prove that safety work pays from every standpoint. In this connection, it is well to be able to say that safety work pays from a financial as well as a humanitarian standpoint, for it is a sad but true commentary that there are still some operators who cannot be appealed to except by a promise of financial gain.

The writer feels that he has not done justice to the subject matter of this paper, especially that part of it dealing with the vast good that has been accomplished through the lessening of death and suffering. Again, some data concerning other phases of the subject have of necessity been omitted owing to the short time that was available in which to prepare this paper.

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# The Oil Fields of Mexico

BY EZEQUIEL ORDOÑEZ,* MEXICO CITY, MEXICO.

(Pittsburg Meeting, October, 1914)

I HAVE read in the *Bulletin* (May, 1914) a paper by H. von Höfer relating to the Origin of Petroleum, in which the author supports his and Engler's views, expressed before, of the organic origin of petroleum. Von Höfer pronounces strongly against the hypothesis of the volcanic origin of hydrocarbons, as particularly advocated by Eugene Coste.

In Mr. Coste's articles, so well known by the public, he refers incidentally to our Mexican oil occurrences, especially of the Gulf coast, as proofs of the volcanic origin of the oils. This belief comes from the fact that, in our Mexican oil fields of the east coast, the majority of the producing wells are located near volcanic necks (remnants of small volcanoes, mainly of the explosive type), or near certain gibbosities in shales, resembling mounds, some of which are really uplifts due to volcanic forces or to abortive eruptions.

The first attempt to explain the oil coming up around volcanic necks in Mexico, was made by me in a small article[1] published in 1905, soon after the investigation ordered by the Mexican government as to the outlook of the Mexican Gulf coast as a future oil producer.

When E. L. Doheny of Los Angeles, Cal., and I located the first producing well of Mexico, I was also the first to predict the great future of the oil industry in Mexico, just at the time when the government and some Mexican geologists felt rather pessimistic. The Mexican investors and the general public, as a result of these discouragements, did not pay attention to this important source of common wealth until recent days. The realization of the value of this source of wealth has been so sudden that we are menaced by the consequences brought about by a great boom.

It was quite impossible, about five years ago, to induce Mexican capitalists to invest money in oil lands. So the few large foreign companies operating since the beginning, easily secured large holdings, in spite of the trouble frequently resulting from faulty titles or ownership.

[1] Sobre algunose ejemplos probables de tubos de erupcion, *Memorias de la Sociedad Alzate*, vol. xxii (1905).

* Honorary Member.

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The main object of this short paper is to expound the fact that the connection between oil and small volcanic apparatus is exclusively mechanical, having nothing to do, of course, with the origin of the oil, which is hypothetically considered by us as of purely organic derivation. Our oil is not indigenous to the rocks in which are contained the porous seams where it is accumulated.

As is well known, the Mexican oil lands are made up of a thick shale formation intercalated in places—especially near the oil seams—with a kind of sandy shale and thin hard flinty beds of the same material. These 4,000 or 5,000 ft. of shale overlie probably a thick Cretaceous formation of limestone with other sedimentary rocks intercalated, in which probably the oil originates.

While it is inferred in von Höfer's article referred to (in quoting the views of a Mexican geologist), that the oil country is a highly disturbed one, I think it is the contrary. That section of the east coast of Mexico does not show signs of great movements. The shale formation—as seen in cuts made by rivers, or in places where the gravel and residuary material of an eroded peneplain which mantled it are washed away—shows only slight gradient undulations, monoclinal low ridges and ample vaults, never big or repeated foldings, nor great displacements by faults. The latter, according to my general observation, are usually of very small importance.

Dikes are practically unknown in the central oil lands, between the Tamesi and Tuxpam rivers, and besides the necks and small lava streams there are some rare hills of plutonic rocks like diorites and nepheline syenites. Further north in the State of Tamaulipas and south to the middle part of the State of Veracruz, some sierras breaking up the coastal plains present more complex geological structures.

Local interior areas, very much disturbed, certainly exist, but have not been proven productive as yet. Highly disturbed shales are found also as a belt in the western margin of the formation against the underlying limestone of which the foot hills and the eastern Sierra Madre mountains are built up.

The volcanic necks are seen scattered in the coastal plain commonly as isolated cones or as small groups of hills; the first type is wonderfully represented in the section between Chicontepec and Tuxpam, of the State of Veracruz.[2]

We should not make the volcanic action responsible for all the undulating and gently boss-like structure of the large shale oil bearing formation, for everything tends to prove that the main tectonic, weak as it is, is prior to the volcanic activity in the coast. The volcanic plug or pipe of ashy and compact basaltic lava, of which the neck is the upper end, has not

[2] Oil in the State of Veracruz, *Mining and Scientific Press*, vol. xcv, No. 8, p. 247 (Aug. 24, 1907).

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produced any considerable disturbance of the shale around it, and I figure that its passage through the sedimentary rock was like a screw through a piece of wood. I imagine that, during the making of the pipes and the coming of masses of hidden intrusive igneous material, there was a certain absorption of the sedimentaries by the igneous molten material in the depths, also the breaking of the sedimentaries near the intrusion and finally a narrow aureole around the plug of brecciated or broken material caused by friction, which zone is utilized in many cases for the rising of minute quantities of oil to the surface and the formation of seepages of very slow accumulation.

A number of large seepages, near which important discoveries have been made, are found at the foot of the necks between the shale and the basaltic rock; other seepages lie in the flanks of dome-shaped bosses. In no rare cases dry hard cakes of *chapopote*, with a soft freshly-risen heavy oil in the center, are encountered in the bottom of small amphitheaters, or horseshoe-like encircling spaces, or in general in the concavities made by curved rows of volcanic hills and small ranges. These different cases of seepages are very favorable for locating wells, as proved by experience (Cerro de la Pez, Chijol, Juan Casiano, Cerro Azul, etc.). Numbers of small seepages or *chapopoteras* are found in the middle of the coastal plain far from any salient topographical accident. It is frequently observed then that the oil exude is not coming directly from underneath the place, as seen from the surface, but that it has run, sometimes for considerable distance, on the shale between it and the thick argillaceous material covering that rock.

In the present state of our geological knowledge of the oil lands it is difficult to explain some important productive areas near river banks which have no apparent connection with volcanic outbreaks nor with bosses. Unfortunately, data collected during drilling and observations of the experts in this and other particular cases never come to the public domain.

I have seen many cases of very small exudes dropping slowly from narrow cracks and little faults in the cliffy banks of rivers where the shales are cut by erosion. This is the case in several places on the rivers Tecolutla, Tuxpam, Calabozo, Pánuco, Tamesí, Valles, etc.

The unimportant veins of albertite and grahamite, so well known in the coast, proceed probably from old exudes filling cracks and fissures where the *chapopote* dries slowly, becomes a little oxidized and is then subjected to pressure.

Lighter oil than that commonly found on the coast central oil lands appears to be found at shallower depths in the disturbed shales near the limestones at the foot hills of the eastern Sierra Madre mountains. This seems to coincide, as in Aquismon, with local impregnation of the limestones by light-oil products. In this case the oil probably comes up

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through the contact between the shale and the limestone which lie in unconformity.

While in this upper belt of the oil country conditions are still entirely unknown, it seems to the writer that a possible source of petroleum exists in this long belt, under conditions probably different from the areas already known, or in the way of development, nearer the Mexican Gulf shore, or in the estuaries bordering it.

I have carefully studied, under the microscope, the sandy sediment accompanying the oils coming from several wells; especially from those near Tampico. This sediment is composed of minute shale splinters and *rounded* grains of basaltic material, similar to the rock of the necks. The volcanic origin of certain mounds and the influence of volcanic action, as facilitating the accumulation of oil, is demonstrated by what we have seen of volcanic rock particles brought with the oil from wells where no necks or any other volcanic rock exposures are found in the vicinity.

The 4,000 or 5,000 ft. thickness of uniform shale oil-bearing strata is naturally very impervious. We often have to drill hundreds of feet in entirely dry shale even in close proximity to the sea and many feet below the ocean level.

If, as we suppose, the shale formation is not very much disturbed and there are no real porous continuous strata in the formation, we have to believe, as far as our incomplete knowledge goes today, that the porous seams, afterward impregnated with oil at the common depths of 2,500 to 2,700 ft. at which we have generally found the oil, are made by volcanic action. Masses of strata corroded by molten lava extend to this depth. It is evident that the strata which become porous ought to show a particular physical structure and a somewhat particular chemical composition different from the rest of the bedding, which facilitates both corrosion and partial absorption by the hot molten lava. We imagine the oil seams elongated horizontally like sills of large dimensions. In order to show the capacities, let us consider a seam or a group of seams capable of producing 3,000,000 barrels without interruption, as in the case of the first producing well of Mexico at Cerro de la Pez near El Ebano, or of the famous burned well of Dos Bocas.

The peculiarities of the strata as supposed to exist, while of a nature absolutely unknown to us, must exist, since there is in our oil-bearing strata something like an oil horizon at an average depth of 2,600 ft.

The exhaustion of a good producing well is marked by the presence of abundant salt water, mud and sulphuric acid solution. The porous spaces after giving the oil, are probably refilled partially with salt water coming up to the seams by the old deeper conduits. The ascension of the oil through the drilling holes is caused by abundant high-pressure gas; the yield of the wells is generally intermittent. There are gushers of very

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long life. I cited several times the first producing well of La Pez which produced about 1,500 barrels a day for nearly 70 months.

Particular difficulties for rapid successful drilling are to be encountered in northern Tamaulipas and south Veracruz, as well as in the east coast of Oaxaca and Tabasco, for the geological conditions of the shales are dissimulated by thick cappings of river and marine gravels in the north and muds in the south covering the vast peneplain extended in the isthmian country. The particular location of the Isthmus with regard to the rest of the eastern coast of Mexico and the Yucatan peninsula has created peculiar climatic conditions facilitating new detrital masses favored by abundant rains, winds, heat, rivers, etc.

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# An Aerial Tramway for Mining Cliff Coal

BY ARTHUR E. GIBSON, STORRS, UTAH

(Pittsburg Meeting, October, 1914)

*Synopsis.*—A new feature in coal mining, where the coal is to be conveyed from a high to a lower elevation and the topography of the country is such as to preclude surface haulage.

The distance from the mine to the tipple in this instance is about 3,000 ft. and the difference in elevation between the terminals 321 ft., of which 225 ft. occurs in the first 1,300 ft. from the loading terminal.

A mine car was designed especially for this plant consisting of two tramway buckets on one pair of trucks, the buckets being lifted from the truck, attached to the tram line automatically, and conveyed to the tipple.

THROUGHOUT eastern Utah, in the Book Cliff range and the Wasatch plateau, the coal-bearing measures are admirably exposed, in the bold and for the most part bare escarpments. The geology is simple, the rocks being almost flat, or at the most dipping not more than 10 per cent.

The coal beds occur in a formation, or group of strata, consisting of sandstone and shale, 600 to 800 ft. thick, which has been correlated with the Mesaverde formation of the Cretaceous. Above this occurs about 2,000 ft. of alternate layers of buff-colored sandstone and shale.

Below the coal lie the Colorado shales, which are estimated to be about 1,500 ft. thick, below which are the red rocks of the Jura-Trias.

As a rule the coal beds all dip in a northerly direction and under the high mountain. The canyons opening out into what is known as Castle valley all cut the coal beds, and the beds rising in the direction of the valley are found frequently high up on the escarpment, often as much as 1,000 ft. above the valley.

The surface features of the region have an economic bearing on the availability of the coal. The valleys and gulches in the Colorado shale extend out from the coal areas across the edge of the plain, so that it is impracticable to build transportation lines parallel with the boundaries of the field within several miles of the coal land. It will be necessary, therefore, to build branch lines up to the coal lands from points on the Denver & Rio Grande Railroad traversing Castle valley.

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Coal mines have been in operation in this region for the past 40 years, some of the more important mines being: Winter Quarters, Castle Gate, Clear Creek, Sunnyside, Kenilworth, Hiawatha, Mohrland, Blackhawk, Storrs, Panther, Standard, Cameron and a number of smaller mines.

The total production of bituminous coal for the State of Utah during the year 1912 was 3,088,356 tons, practically all of which was from Carbon and Emery counties. The older mines were opened in the most favorable locations, *i.e.*, where the coal was low in the canyon, or where the bottom of the canyon intersected the coal floor and where an easy grade could be procured in building a railroad to the mine.

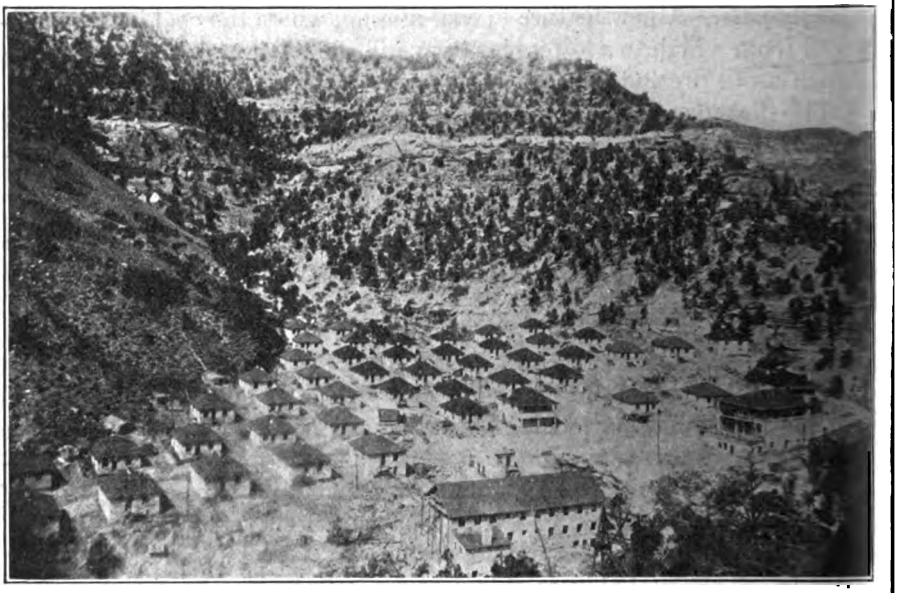

FIG. 1.—VIEW OF PART OF STORRS, SHOWING BOARDING HOUSE, STORE, AND ROCK COTTAGES.

Of late, however, all of the mines have been opened at a high elevation above the level of the surrounding country, necessitating a system of haulage which would successfully lower the coal to a level which could be reached by railroad.

In most instances gravity planes have been installed, but at Storrs an aerial tramway is in operation.

This mine was opened in the fall of 1912 and is located in Spring Canyon, which opens upon Price river, 4½ miles west of Helper, where a branch leaves the main line of the Denver & Rio Grande Railroad. The branch railroad from Helper to Storrs, having a maximum grade of 3 per cent., was built by the Spring Canyon Coal Co. and later turned over to the Denver & Rio Grande for operation.

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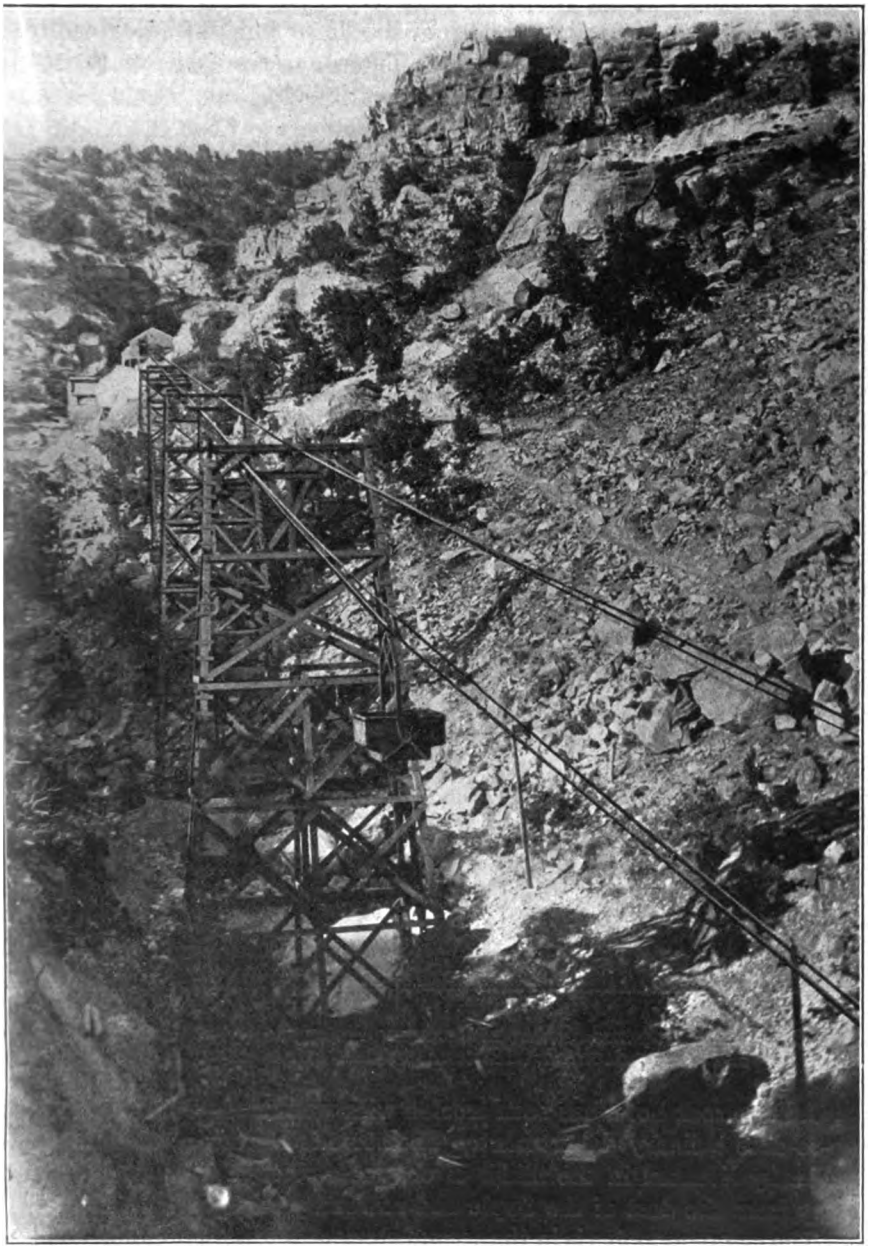

FIG. 2.—GENERAL VIEW OF COUNTRY WHERE TRAMWAY WAS CONSTRUCTED.



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The Spring Canyon Coal Co. was organized in 1912 by Jesse Knight, of Provo, Utah, the coal property consisting of about 2,000 acres of land, upon which have been erected 63 rock dwellings and 10 frame dwellings for the convenience of the employees. These houses are modern throughout, being equipped with bath, toilet, electric lights, etc. (Fig. 1.)

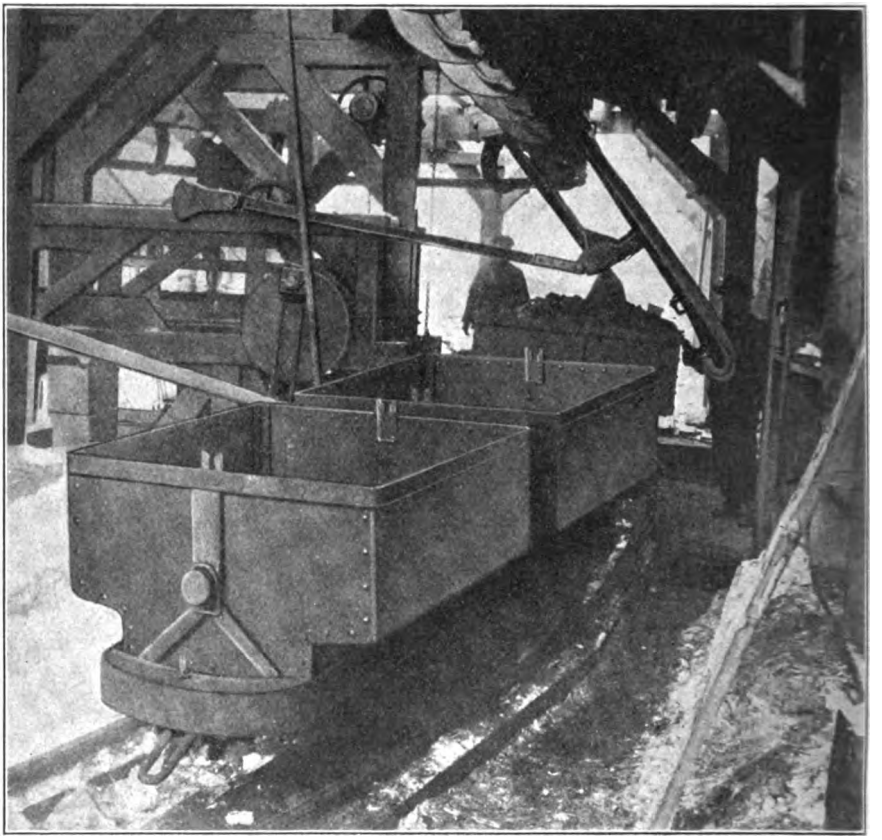

FIG. 3.—EMPTY BUCKETS ON TRUCK COASTING AWAY FROM TRANSFER AND LOADED BUCKETS APPROACHING. HANGERS PUSHED OUT TO CLEAR BUCKETS.

A large rock hotel has been erected with well-furnished rooms and an up-to-date culinary department. The store is of rock also and is conveniently situated in the midst of the greater number of cottages and away from the dust of the tipple and railroad yards.

The company office is in the second story of the store building and is a large roomy office with all the necessary equipment for an up-to-date mine office.

The power plant is located at the terminus of the railroad and near the

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tipple, this point being approximately 321 ft. lower in elevation than the portal of the mine some 3,000 ft. away.

The power house is built of rock with asbestos shingles carried upon steel trusses, making the building fireproof. Coal for steam is brought from the tipple, situated nearby, by a belt conveyor and dumped into a bin directly in front of the boilers.

Steam is generated in five Scotch marine boilers of 150 h.p. each. The power-house equipment consists of two 250 kilovolt-ampere sets and one 50 kilovolt-ampere set. The former are Buckeye engines connected to General Electric generators, while the smaller unit is driven by a Sturtevant vertical engine.

The electrical current generated is 2,300 volts, which in turn is transmitted to the mine and transformed by three motor-generator sets of 100 kw. to 250 volts direct current, which is used in the mine for all purposes, including coal-cutting machines, electric locomotives, fans, pumps, etc.

Three veins have been opened. The lower, known as No. 1, is a vein of about 8 ft. in thickness. The coal is a close-grained, glossy black, hard, bituminous coal, suitable for steam and domestic purposes especially, and on account of its hardness makes the best storage coal on the market today. This vein is known as a subvein, being 180 ft. lower geologically than any of the other workable veins in the region. The floor is of massive sandstone and the roof is a very hard shale, which does not need any timbering in ordinary entry work and very little in room and pillar work.

The next vein above is about 50 in. in thickness. This vein is approximately 15 ft. above No. 1. The hard shale roof of No. 1 extends up to and forms the floor of No. 2. The coal is very similar to that found in No. 1 but not quite as hard. This vein has not been developed to any great extent, due to the fact that it is too small for the large and specially constructed mine cars in use in the other veins.

The third vein, No. 3, is what is known in the region as the Castle Gate vein and is the most persistent of any of the veins in the locality, extending for a great many miles in all directions.

This vein has a thickness of from 3 to 14 ft. throughout the entire region, but where developed by the Spring Canyon Coal Co. workings, shows a thickness of from 6 to 12 ft., most of the workings being in 8 ft. of clean coal.

Although the general dip of the formation in the vicinity is about 7 per cent. all entries are driven sufficiently across the pitch to make the maximum grade not over 3 per cent., permitting the use of electric locomotives both on the main haulageway and cross entires.

Two 15-ton General Electric locomotives are used on the main haulage, and three 6-ton gathering locomotives of the same type are used in connection with the horses in the cross entries and rooms.

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The coal from No. 3 mine is brought along the side hill for a distance of 2,500 ft. on a 3 per cent. grade until it reaches the No. 1 vein.

Here it enters No. 1 mine through what is known as the first left entry, and thence is conveyed to the head house, or loading terminal, of the aerial tramway.

Eighteen cars are hauled in each trip by the 15-ton locomotive; 50-lb. rails are used on the main-haulage tracks and 25-lb. on the cross entries.

The motor-generator set referred to above is located in a central position with respect to the haulage system, the greatest distance the trolley line extends in any direction from the motor-generator set being about 2,900 ft.

Heavy copper feed lines are connected to the trolley system at central points in the mine, where the load is heavy, as it is essential to avoid any excessive drop in voltage at the working faces.

The coal-cutting machines in use are of three types and consist of seven Goodman, four Sullivan, and one Jeffrey, all of the shortwall continuous chain cutting variety. These machines are giving excellent service.

The topography of the country precluded the use of a surface incline and the exorbitant price of adjoining property prevented development of grades for motor haulage, or railroad to the mine portal.

The aerial tramway was feasible but had the objection of limiting the output of the mine, due to the small capacity of this system. The tramway was built for 60 tons hourly capacity, to be doubled by additional rolling stock. With the opening of the upper coal measures it was soon evident that this capacity would handicap the mine operations. After much negotiation with the manufacturers, they agreed to develop a capacity of 250 tons hourly on the line.

Our tramway has been so fully described in the *Salt Lake Mining Review* and the *Coal Age*, that the writer will merely outline the principal features, together with such changes as have been made to the original equipment to develop the increased tonnage.

In general, the tramway is 2,900 ft. long and has a fall of 321 ft. between the terminals. The track cables are $1\frac{5}{8}$ and $1\frac{1}{4}$ in. in diameter for loads and empties respectively, working under a tension of 35 and 15 tons each. The running rope is $\frac{7}{8}$ in. in diameter, lang lay. The grips are of the regular friction type, operated directly by weight of load, and hold very effectually.

The line operates at a speed of 500 ft. per minute when driven by motor, and 530 ft. when the motor is acting as a brake to take care of the surplus power developed by the loaded line.

In operation, two buckets are set in a truck frame of 6-in. angles, having a wheel base of 36 in. and an overall length of 11 ft. 3 in. The buckets

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are of 40 cu. ft. capacity, rated at 2,000 lb. but in actual service averaging about 2,200 lb.

The car with the two buckets is run on to a platform directly under the tram rail, upon which two hangers are positioned waiting for the buckets.

The car is stopped, the hangers swung under the trunnions of the buckets, and the platform lowered, carrying the truck with it, leaving the buckets suspended in the hangers. The hangers are then free to coast out on the cable. Two hangers with empty buckets are then brought

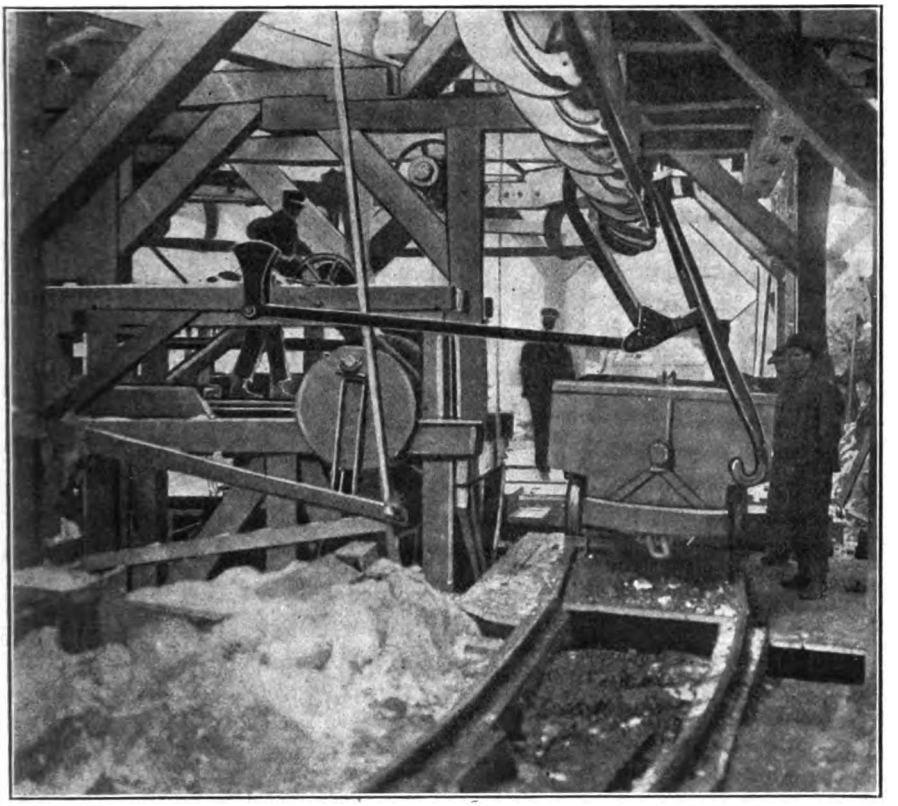

FIG. 4.—PLATFORM IN RAISED POSITION AND HANGERS BEING PUSHED FROM THE BUCKETS.

over the empty truck and the platform raised until the buckets rest on the truck, when the hangers are swung out from under the trunnions of the buckets. (Fig. 4.) The truck with empty buckets is allowed to coast through the kick-back and run into the mine while another load is brought forward for transferring to the hangers.

In developing 250 tons capacity several changes were necessary, as outlined below:

The strength of the cables being based on the weight of the individual

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load rather than the total loading of the line necessitated no change. As the line lowers the loads, more power was developed than the original 35-h.p. motor could handle so a 75-h.p. alternating-current motor was installed together with wider belts, clutch, and pulleys. Additional brake sheaves were put in for stopping the line. The speed of the motor is 900 rev. per minute and to reduce it to 22 rev. per minute at the main shaft two jack shafts are used, stepping down the speed three times, twice by belts and once by gearing. To allow proper tensioning of the individual belts adjustable sole plates were provided for the bearings on the countershafts.

Owing to interruptions of power circuits the motor could not be relied upon to control the speed of the line at all times and several expensive runaways were caused by failure of circuit breakers or fuses. To overcome this, the manufacturers worked out a very ingenious control to regulate automatically the speed of line without stoppage in event of any spurt above normal. Briefly stated, this device resembles a large fly ball engine governor and is gear connected to the main shaft to avoid possibility of failure due to belts or clutches. Instead of weights, as in a governor, large vanes are mounted on large pivoted arms which are extended as the speed rises above a predetermined point and effectually prevent the tramway traveling at a higher speed. It is entirely automatic and not affected by changes in temperature. A large dial with a pointer indicating the speed of the line is connected to an electric gong as an additional warning of excessive speed.

With increased loading of the line, the bending of cables over saddles was excessive and a new saddle was developed to overcome this trouble and prolong the life of the cables. (Fig. 5.) The new design consists of two saddles mounted on a 15-in. channel which is pivoted at the center. As the load approaches the tower, the first saddle is depressed and the second is elevated until load reaches the pivot point, when the reverse takes place. The smooth riding of the load over the towers since the installation of these saddles is very marked. To facilitate the turning of the cables with heavy loaded line a new swivel for terminals was designed which allows cables to turn freely under load.

When handling buckets on 60-sec. intervals the coal was dumped into a hopper 9 by 12 ft., but it was found necessary to replace this by one 20 by 22 ft. With this area to dump into there is no trouble in handling buckets on 15-sec. intervals.

Owing to constricted space on the ledge upon which the loading terminal is built the curve of the track for bucket carriers is of short radius. The settlement of part of the structure here caused by proximity of dump fire prevents proper grading of track and causes many derailments of buckets. The buckets derailed here drop several feet. This bends the hangers and breaks small parts and has been the cause of most of our repairs.

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The long overhang of the cars is an inherent weakness and a force of nen is constantly at work keeping them in proper repair. They are very

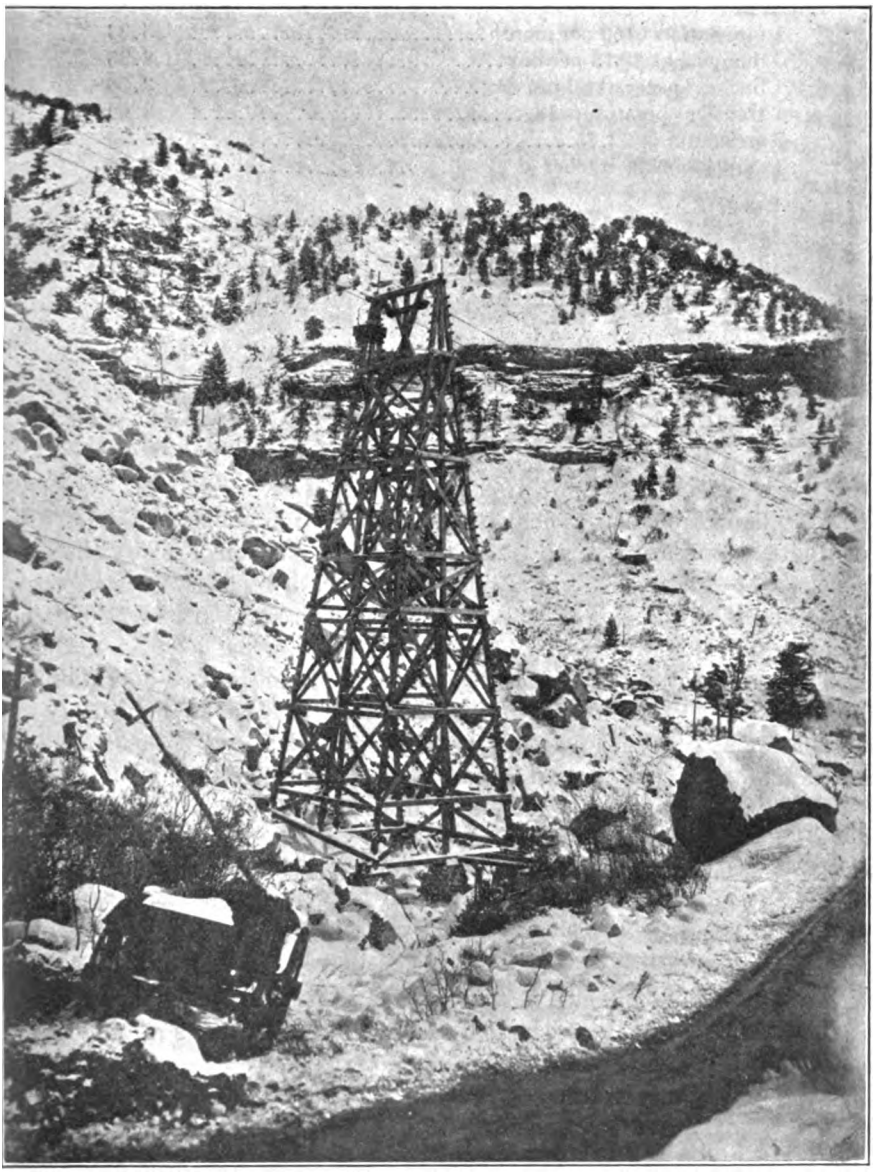

FIG. 5.—TYPE OF SUPPORTS FOR CABLE.

clumsy to handle when derailed. Our underground haulage cost is much greater than would be the case with the regular type of pit cars. The

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writer believes one bucket per truck would overcome these objections and make a nicer car to handle underground.

The tramway force and wages paid are as follows:

| | |
|---|---|
| 1 foreman at $160 per month | $5.33 |
| 3 dumpers at $2.75 per day | 8.25 |
| 1 bucket spacer at $3 per day | 3.00 |
| 1 transfer operator at $3 | 3.00 |
| 2 assistants at $2.75 | 5.50 |
| 1 empty-bucket catcher at $2.50 | 2.50 |
| Total | $27.58 |

In addition to the above, not actually engaged in handling the tramway, but in handling the cars outside the mine, are:

| | |
|---|---|
| 2 spraggers at $2.50 | $5.00 |
| 1 weighman at $85 per month | 2.83 |
| 1 oiler at $2 | 2.00 |
| | $9.83 |
| Gross total | $37.41 |

This figure remains unchanged regardless of tonnage, which is unfortunate as our cost per ton is much greater at 500 tons per day than at 1,500 tons. These costs do not include items of repairs, changes, etc., as the line is in the experimental stage and such costs are borne by the manufacturer.

The manufacturer has shown a commendable spirit of co-operation in the design of various changes and alterations to involve as slight delay in installation as possible and work in harmony with various other machinery, much of which was not furnished by him originally.

Many changes have been made and all seem to be an improvement. Two important drawbacks encountered are: First, the size of the force, whether on large or small tonnage, and second, the trouble encountered with trucks and cost of repairing them. With these points corrected and other improvements made so as to bring the tonnage up to 250 tons per hour, I see no reason why this tramway should not compare favorably with any system of haulage now in vogue in this coal field.

The most coal put over the line so far in a regular day's run has been at the rate of 200 tons per hour for short periods.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# A New Safety Detonating Fuse

BY HARRISON SOUDER, CORNWALL, PA.

(Pittsburg Meeting, October, 1914)

THE object of this paper is to bring to the notice of engineers a safety detonating fuse by the use of which misfires in blasting may be eliminated and safety in blasting operations promoted.

This new detonator is a French invention and is known as *Cordeau detonant* or detonating fuse and is sold under the name of Cordeau-Bickford. It consists of a lead tube 5 to 6 mm. in diameter, filled with trinitro-toluene.

While applicable to all classes of mining it will appeal especially to those who have to do with deep-hole blasting in open-cut mines or quarries, or any operations where a large number of holes are to be shot at one time.

Since the introduction of the deep-hole blasting in the Cornwall iron mines, much trouble has been experienced from misfires and many experiments were made to determine the best and safest method of blasting.

Before the introduction of the detonating fuse it was found that to insure the best results, two or more high-power electric exploders, not less than No. 6, with specially insulated conductors, should be placed in each hole; the holes wired in parallel, and for a large number of holes the current to be supplied by a live wire to the bus or lead wires at about the middle of the bench. The voltage in general use is 110 to 220, but as our material is a magnetic iron ore and the holes are generally damp, we found it advisable to reduce this to about 30 volts to prevent short circuits. A No. 7 or No. 8 special insulated detonator was adopted. But with the best of care and attention to the minutest detail, we were not able to eliminate occasional misfires and the consequent ever present danger of accidents.

In January, 1914, our attention was called to "Cordeau-Bickford" and we were invited to witness a trial blast with this material at the quarries of the Atlas Portland Cement Co. at Northampton, Pa. This was the first use of this detonating fuse in this country and as it marked

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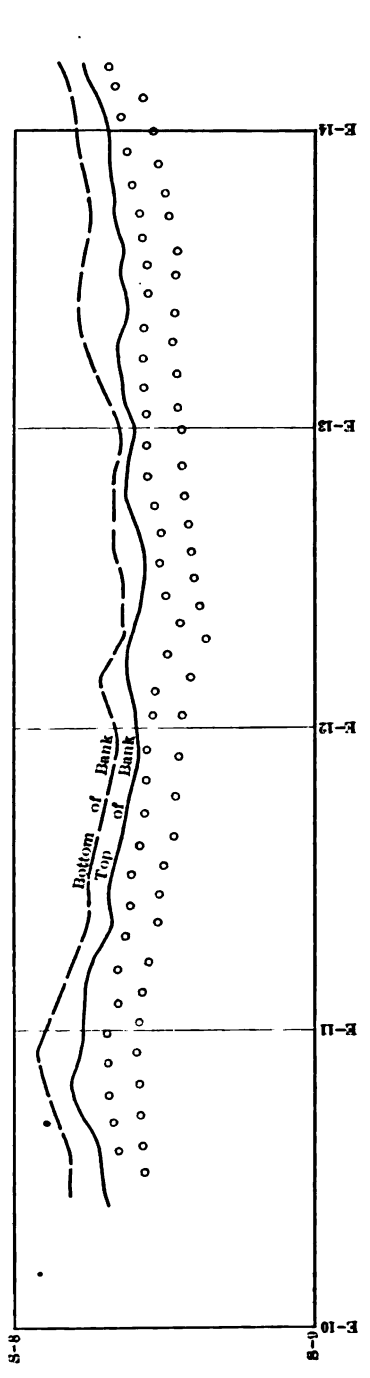


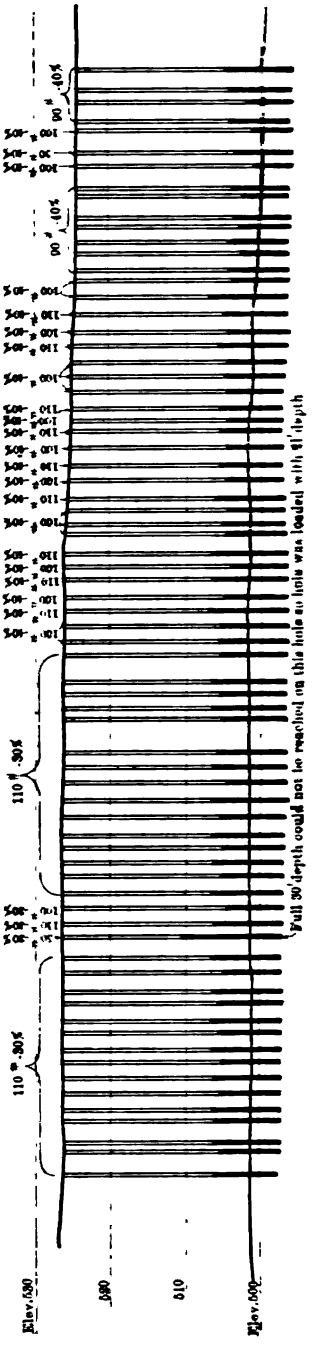


FIG 1.—BLASTING SHEET OF CORNWALL ORE BANK CO. (See opposite page.)

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## ORE WELL DRILL SHOT

April 2, 1914

### CORNWALL ORE BANK CO.

Shot Fired with Live Wire 2.55 p. m.

Holes were all pumped dry by means of a 2-h.p. gasoline engine and sand pump, thereby letting powder to the bottom of hole.

72 holes fired { 37 holes in front row. / 35 holes in back row.

Average depth of holes = 30 ft.

*Summary*

3,410 lb. 30 per cent.
3,870 lb. 40 per cent. } $830.07 = Total cost material.
1,000 M. Cordeau

*Summary Continued.*

9,129 cu. yd. ore moved or 25,561 tons.

Cost = 0.0909 per yd. or 0.0325 per ton.

Powder used = 0.8 lb. per yd. or 0.28 per ton.

East end of shot consisted of very bony ore and the holes were therefore loaded with 90 lb. 40 per cent., while the rest of the shot was rich heavy ore and 110 lb. 30 per cent. was used or 100 lb. 40 per cent.

All Cordeau and no exploders used in holes.

Holes were all very wet. Majority of them were over half full of water.

Better results were obtained by this shot than by any preceding one. Less powder was used. Bottom came out first.

FIG. 1A.—DESCRIPTIVE DETAIL OF BLAST AS FILED ON BLASTING SHEET OF CORNWALL ORE BANK CO. (See opposite page.)



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a great advance in safety and economy in quarrying operations a brief description of this blast will be of interest.

The rock blasted was a cement limestone shale bench 94 ft. high (perpendicular) with the bottom well cleaned up. There were eleven 5-in. diameter holes, 26 ft. back from the face of the bench and spaced 13 ft. apart, 100 ft. deep and extending 6 ft. below the bottom of the bench. The holes were cleared of water just before loading. The charge was made up of 50 per cent. gelatin and 40 per cent. low-freezing dynamite, with the detonating fuse, 6 mm. in diameter, running to the bottom of each hole.

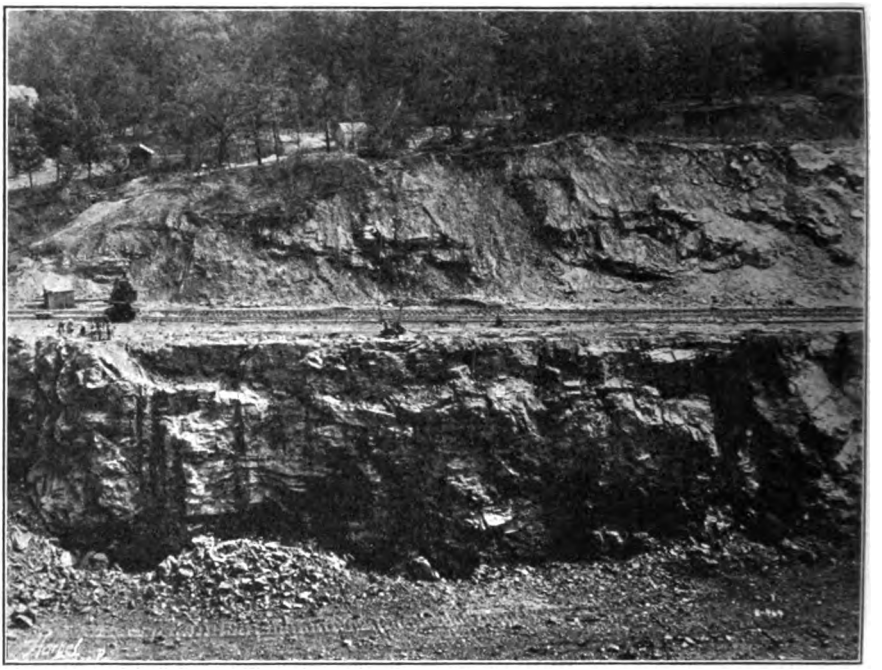

FIG. 2.—LOADING THE HOLES FOR THE BLAST.

All the holes were connected with a line of detonating fuse to the end of which a single No. 6 detonating cap was attached. This was fired by a 220-volt current from a live wire. Instantly there was a sharp crack as the fuse exploded, followed by a rumble of the crumbling bank. The rock was thrown well out, leaving a clean wall or face.

The results of the blast were so satisfactory that we immediately ordered 1,000 m. of the detonating fuse for trial in blasting down our ore benches. This was the first application of this fuse to ore mining.

A trial convinced us of the wonderful adaptibility of the detonating fuse to the purpose, and the simplicity and safety in handling brought a

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ense of relief and satisfaction. We, therefore, immediately adopted it or use in all our deep-hole blasting and stopped the use of the ordinary lectric fuse except in shallow holes. We have been using the detonating use for five months and in this time we have made a dozen big blasts.

The details of a shot made on Apr. 2, 1914, when 72 holes averaging 30 ft. deep were blasted at one time by means of the detonating fuse are given in Fig. 1. This is a copy of our standard blasting sheet and gives all information in compact form for future reference. Two No. 7 exploders were used and fired by current from live wire.

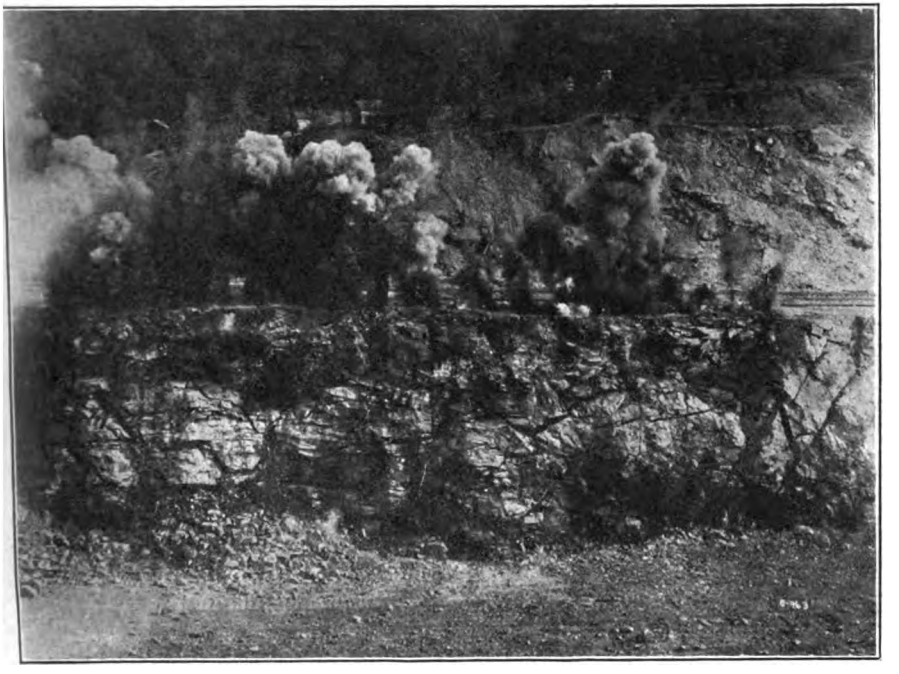

FIG. 3.—DURING THE BLAST.

On May 20, we blasted 26 holes varying from 80 to 85 ft. deep and using in all 7,800 lb. of 40 per cent. nitroglycerin dynamite with the detonating fuse, using two No. 7 exploders discharged by means of a battery. About 38,000 tons of ore were broken down.

Some views of the ore bench before, during, and after the blast are given in Figs. 2, 3, and 4.

The danger arising from the old method of blasting with electric exploders is exemplified by the only too extensive list of accidents due to premature explosions, misfires, etc. Such an accident as occurred during the construction of the Panama Canal in December, 1908, when 22 tons of dynamite exploded prematurely, killing 23 persons and injuring

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40 others, would have been practically impossible if this fuse had been used.[1]

In addition to greatly increasing the safety of blasting, the detonating fuse has the additional merit of increasing the efficiency of the explosive charge.

To summarize, we may state that this fuse has three very important qualities, viz.: 1. It is safe. 2. It is instantaneous. 3. It increases the efficiency of an explosive charge.

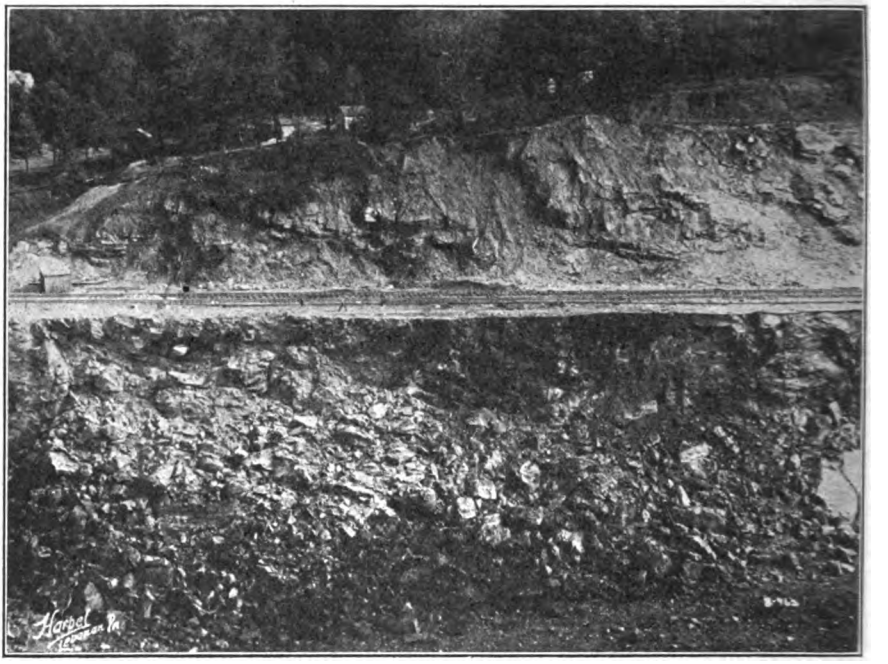

FIG. 4.—AFTER THE BLAST.

1. *Safety.*—There is no danger in the handling or storage of the fuse. It cannot be exploded by friction, fire, or ordinary shock. It requires the use of a strong blasting cap properly attached to explode it. In blasting charged holes, the cap or exploder can be applied outside the hole, thus avoiding the danger of burned powder caused by side spit from ordinary fuse; also any risk of accident while tamping and the risk from a portion of an unexploded charge accompanied by a cap remaining in the débris from a blast is entirely obviated.

2. *Speed.*—The average rate of speed of this fuse is estimated to be close to 17,000 ft. per second, so that when it is used the explosive

[1] See *The Panama Gateway*, p. 337.

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charge is detonated instantly throughout its entire length, instead of at one point as is the case with the blasting cap or electric exploder.

3. *Efficiency.*—It is known that the speed of an explosive decreases as the explosive wave travels away from the detonator. That the powder in a hole has the strongest explosive effect around the exploder is evident from an examination of the face of the bank after a shot. This can be demonstrated also by placing sticks of dynamite on the ground end for end, about 6 in. apart, with the cap in the first stick. The explosive force gradually lessens until it finally ceases to progress, leaving the farthest sticks unexploded.

By using this fuse the charge is detonated instantaneously throughout its entire length. This results in a saving of about 10 per cent. of explosives as determined by results obtained at Cornwall and elsewhere. It is not affected by heat, cold, or moisture, and lasts indefinitely without deterioration.

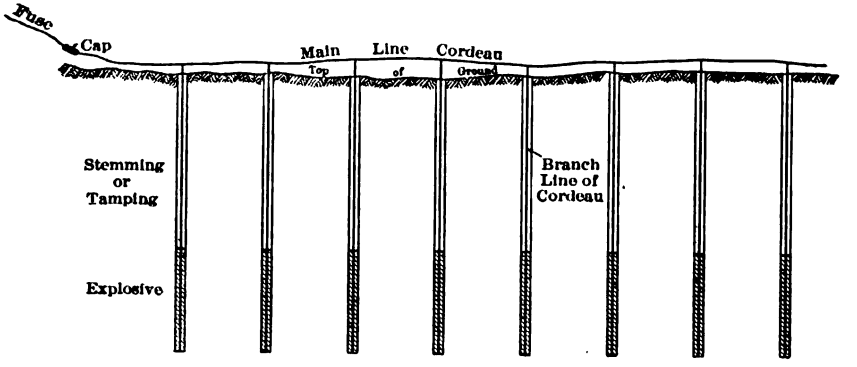


FIG. 5.—HOLES CONNECTED BY CORDEAU.

It is wound in continuous lengths on spools containing 100, 200, or 300 ft. each, and weighs about 7 lb. per 100 ft. It is accepted by transportation companies without restrictions except that it shall not be packed with other high explosives.

The method of applying the fuse is shown clearly by the accompanying sketches and instructions.

Fig. 5 shows how the holes are connected by the detonating fuse. The method of making connections between main and branch lines is shown in Fig. 6, and that of attaching the detonator to the fuse in Fig. 7. Either ordinary fuse and cap, or an electric exploder, can be used to set off the detonating fuse.

*How to Attach a Detonator to the Detonating Fuse.*—First, be sure the end of the detonating fuse is freshly and squarely cut off. If ordinary fuse is used, attach a cap by crimping it to the fuse in the usual manner. Next attach a union to the detonating fuse by crimping the end opposite

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the slit. Insert the cap in the slit end of the union till it comes firmly against the detonating fuse and then slip the ring over it to hold it firmly in place. Be sure the detonating fuse is cut off squarely and that the cap is firmly seated against it. A space of $\frac{1}{8}$ in. between cap and fuse may be sufficient to cause a misfire.

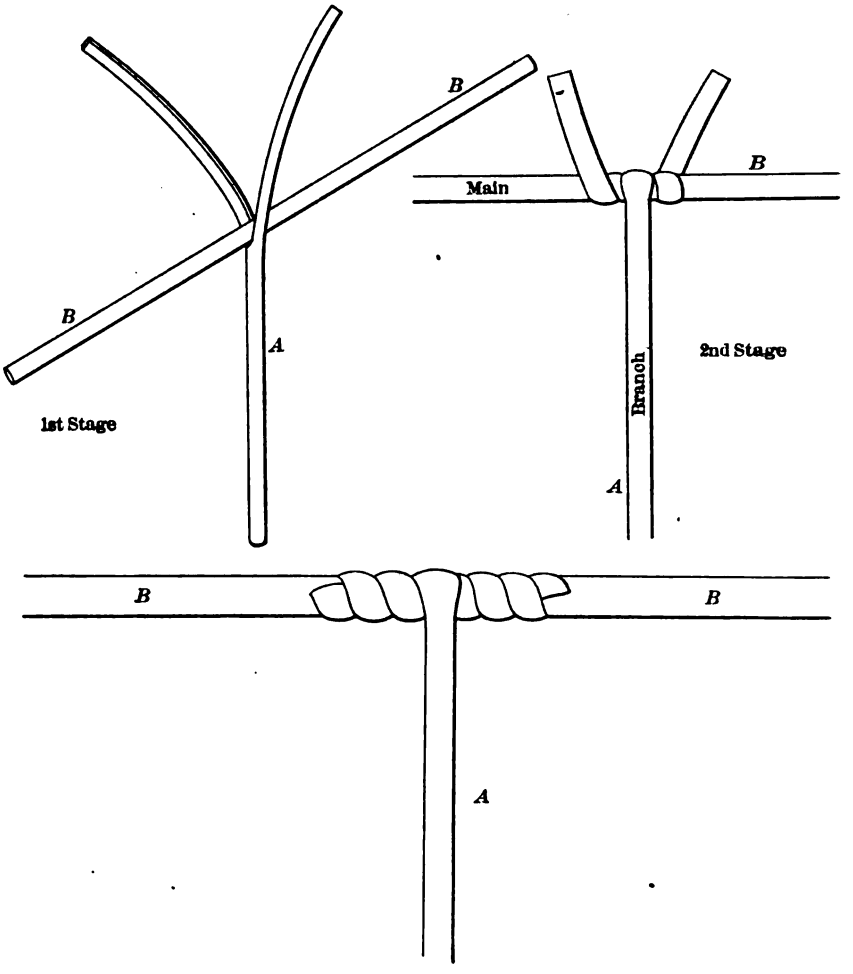


FIG. 6.—METHOD OF CONNECTING MAIN LINE AND BRANCHES.

Branch line *A* is joined to trunk line *B* by twisting the split ends, one to the right, the other to the left of the intersection. These joints may be protected from danger of loosening or breaking by winding with electric tape. This also protects them from moisture. This precaution is not necessary except where many connections have to be made or wet work is encountered.

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This operation is not difficult, but it is very important, and should be done carefully and in accordance with three simple rules.

1. Remember that it is the *branch* fuse that is to be slit and twisted about the *main* and never the opposite.

2. Be sure to make a firm joint.

3. At the point of intersection, be sure to lead the branch fuse away from the main at a *right angle*, at least for an inch or two. Beyond that curves and angles do no harm.

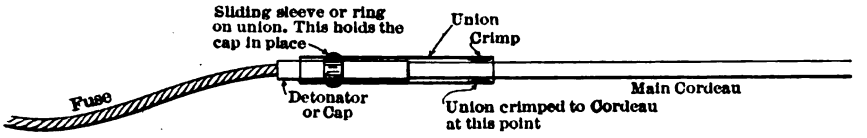


FIG. 7.—METHOD OF ATTACHING DETONATOR TO CORDEAU.

If the direction of the explosive wave is carefully borne in mind, there can be as many branches and sub-branches as are desired. To slit the end of the branch line an instrument called a "cordeau slitter" is provided. It can, however, be done easily with a sharp knife. After slitting it for a distance of 3 or 4 in., separate the legs and, after placing the main snugly in the crotch, wind the legs about it, one to the right and the other to the left. If some of the trinitro-toluene is lost from the split ends, no harm is done provided the main is pushed firmly against the

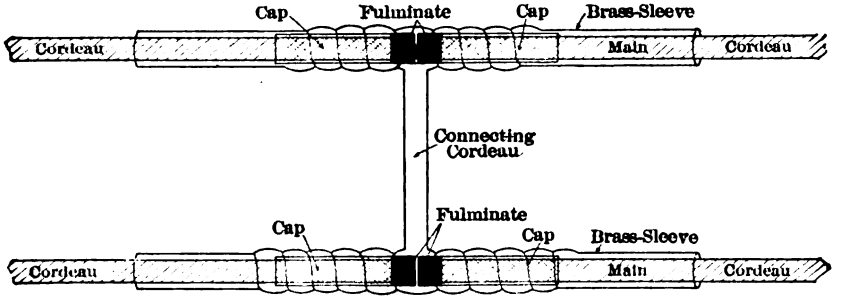


FIG. 8. —CONNECTING CORDEAU IS SPLIT AND WOUND AROUND THE BRASS SLEEVES IN THE SAME MANNER AS WHEN MAKING ORDINARY JOINTS.

exposed material in the crotch. If many connections have to be made, it is possible that the handling and moving of the fuse may displace some of the connections first made before all can be completed. To guard against such an occurrence it is better to wind each joint with electric tape. This will hold them firmly in place. Before a shot is fired, the line should be thoroughly inspected. See that the branches leave the main at right angles. Be sure that they are all made firm.

On very large and important work where more than one main is employed, these mains may be joined together as a measure of precaution.



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This is done in the manner shown in Fig. 8 by cutting the mains and putting caps on the ends and then uniting them again by means of a brass sleeve, the ends of which are firmly crimped to the detonating fuse. Around these brass sleeves the ends of the connecting line are wound in exactly the same manner as when attaching a branch to a main line.

Interesting details in regard to the use of this detonating fuse in France and Belgium may be obtained from the following articles, of which, for convenience, the writer has made a brief digest, viz.:

Commission du Grisou: Note sur l'amélioration de la sécurité dans les mines grisouteuses par emploi d'un nouveau dispositif d'amorçage des explosifs. *Annales des Mines*, 10th ser., vol. xii, p. 169 (1907). An investigation to determine whether the use of detonating fuse would increase safety in gaseous mines. Describes numerous tests and experiments and concludes with the following statement of the advantages of the detonating fuse:

1. No misfires, partial explosions, or burning of powder.
2. Allows reduction of powder charge. Estimated at 20 per cent.
3. Permits the use of explosives of very low sensitiveness.

Nouvelle Application du Cordeau Détonant aux travaux publics, by L. Barthélemy. *Bulletin de la Société des Ingénieurs civils de France*, November, 1910, p. 492. Describes the successful application of detonating fuse in the destruction of a 318.6-m. masonry tunnel on the Ypres canal in Belgium.

Shot 2,125 holes simultaneously. The charge was 3,250 kg. of melinite. The delicate part of the operation was to find a way to explode all of the holes at once. The first use of detonating fuse on a large scale.

Note sur le Cordeau détonant, by M. Fougerolles, Chief Engineer. *Comptes rendus mensuel de la Société de l'Industrie minérale*, February, 1908, p. 58. Description of tests made by Cia. des mines de Lens with permissible explosives and detonating fuse.

Conclusions: Compressed powders practical with easily observed precautions.

1. Be sure the hole is of large enough diameter to permit the charge to be forced tightly to the bottom.

2. Make a fresh clean cut at end of the detonating fuse, and place it in contact with the exploder so that it cannot slip away while loading the hole.

No unexploded powder was found and it is claimed 20 per cent. of the explosives by weight and 10 per cent. of expenses were saved. Detonating fuse is well liked by the miners.

L'Explosion du Rocher de Formery. *L'Illustration*, May 31, 1913. Illustrated description of the removal of a large mass of rock, overhanging the village of Formery in France, by blasting at one time 237 holes loaded with 92 per cent. nitroglycerin dynamite and connected with the detonating fuse. The whole was exploded with one detonating cap.

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# A Practical Wood-Burning Assay Furnace

BY E. A. H. TAYS, SAN BLAS, SINALOA, MEX.

(Pittsburg Meeting, October, 1914)

LAST fall, having a number of ore samples from mine-development work carried on in spite of the "Revolution," I was forced to do my own assaying again, after a lapse of many years. This gave me an opportunity to build a furnace for crucible assaying, using wood for fuel, along lines which had been developing in my mind since the time 20 years ago, when as general factotum at a little mining camp in western Chihuahua, Mexico, I had converted a crucible charcoal furnace into one using the reverberatory principle.

Wood-burning furnaces have been described in technical literature, but I believe the present application will prove helpful to those of the profession who are working in out-of-the-way places. The furnace is shown in detail in Figs. 1, 2 and 3.

Remembering the trouble caused by stoking my first charcoal furnace too heavily, I fired this one with but a few sticks of wood at a time. By this method I could run out a charge of 12 crucibles in 30 to 35 min., using 50 sticks of ordinary stove wood. Later I fill up the firebox, holding about 40 sticks, get everything ready, put the crucibles into the melting chamber and "fire up." Just as soon as the fire is well started the draft door is closed and the furnace allowed to heat up slowly for 5 min. Then the draft is opened (the door to the ash-pit) and the fire allowed to roar, which it does. In 20 min. from the time the fire is lighted the fire door is opened, the fire stirred along the grates, and 5 or 6 more sticks of wood thrown in. In 25 min. from the start, the draft is closed again, the cover of the melting chamber pushed back and the charges poured. The melts will be found to be perfect, and the crucibles to pour absolutely clean, if everything has gone right, as will usually be the case. A new charge may be put in, and fired with 8 to 10 sticks of wood at a time, and be ready to pour within 20 min.; but it is just as well to leave the crucibles in the melting chamber a full 25 min.

By using 10 sticks of wood to start the fire and adding 5 sticks as often as needed, which will be as soon as the last sticks fed in are in perfect combustion, the smelting can be completed in from 30 to 35 min., using from 35 to 40 sticks of wood. I consider this to be the best method of stoking.

I use the Battersea No. 10 crucible, having no others, but the base of

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these is so small that they are very apt to tip over in the strong draft; and often two and three will go over at a time, unless great care is used in setting them on the hearth. This precludes the putting in of a second charge, as, due to the heat, the crucibles cannot be set with the necessary care.

The crucible to use with this furance, whatever size be used, is the wide-based, flower-pot shaped crucible made by American manufacturers. With this class of crucible one can run as many charges as he cares to without the fear of a mishap of any nature.

In "drawing," one asbestos glove is necessary; and, when nails are used, they can be withdrawn easily with the cupel tongs.

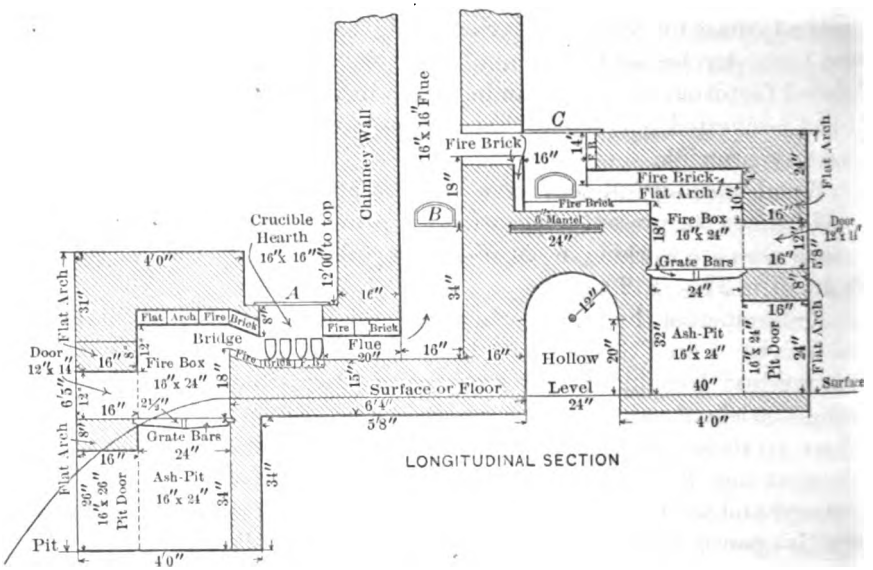


FIG. 1.—WOOD-BURNING CRUCIBLE AND MUFFLE ASSAY FURNACE.

When I built my furnace, I built at its back, and leading into the same chimney, an ordinary charcoal, muffle furnace. Later, as charcoal was hard to get, I decided to try wood with it also, and altered it to the shape shown in the drawing. It works splendidly, and once heated up, is easy to keep hot.

When firing the muffle furnace, burning charcoal, I noticed, when the crucible furnace was in use, that the flame from this passed up the chimney beyond the top of the muffle flue. I mentioned this to R. C. Sanford, of the Palmarito mine, who was examining my plant, and he suggested putting the muffle in the chimney. I could not do this very well, but he did, when he built a furnace after the plans I loaned him.

How well the furnace works, and the success attained by putting the

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ıuffle in the chimney of the crucible furnace can be judged from the fol-
›wing extract from a letter Mr. Sanford wrote me recently.

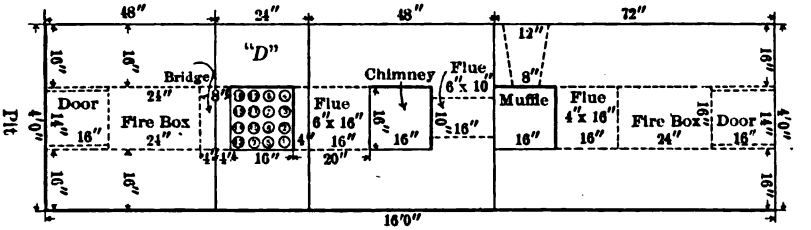
"On my return to Palmarito I at once started to build a furnace, and it has been in ›eration for some time now, and is a success in every way. I took great care in ıllowing the inside dimensions in your plans.

"In place of building a separate furnace for the muffle, I put the muffle in the ıimney of the crucible furnace (as an experiment), 34 in. from the floor of the fur-ıce and 1½ in. from the chimney wall. While the muffle is placed higher than need ›, it does perfect work.

"Mr. Price, of the Potrero mine, has also built a furnace from your plans and has ›t his muffle somewhat lower. When I last saw him he had not started using his ırnace, so I do not know what success he has had with it.

"The furnace has certainly been a wonderful help at the present time with gasolene › scarce and high priced; but without doubt we shall continue to use it even when ısolene is more plentiful and cheaper. It is as quick, is cheaper, and does the work › well as a gasolene furnace, so it is to our advantage to continue using it.

"You are to be congratulated on your success in devising it."




PLAN

FIG. 2

Since receiving this letter I have seen Mr. Price, and he was as enthusi-ıstic in the praise of the wood-burning furnace as Mr. Sanford.

The only disadvantage of putting the muffle in the chimney is that it uts the mouth of the muffle rather low for comfortable work, as the floor ıould not be over 24 in. below the top, *A*, of the melting chamber, which ·ould leave the muffle about 44 in. above the floor.

The whole furnace can be constructed of ordinary brick and mud mor-ır, but the arches, and all parts exposed to the greatest heat, should be ned with fire brick.

The chimney, built as shown, need not be over 12 ft. high; and if a amper were put in near the top it would do away with the need of a door › the ash pit. But, as there is no heat at this place, a piece of 2-in. plank, ·ide enough and high enough to cover the opening, is all that is needed, lthough a well-fitting door looks better and is more convenient.

The top to the melting chamber, *A*, should be a solid fire-brick piece 2 in. square and 1½ in. thick; but a piece of asbestos board 2 by 22 y 22 in. would do; or a piece of sheet steel ½ in. thick by 22 in. square.

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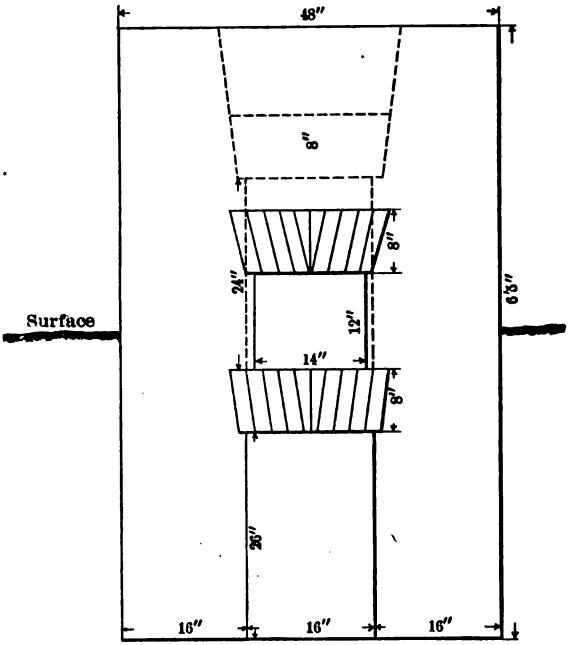
I use a piece of cast iron 1 in. thick, but it requires the aid of a helper to move, as it weighs over 125 lb. This heats, but not excessively, and makes a splendid place to warm up the molds.

The grate bars are shown longitudinally but may be placed crossways, and where regular grates are not to be had, 1-in. round iron cut in 18-in. lengths and 1 in. at each end flattened to $1\frac{1}{2}$ in. wide, will answer capitally.

An ordinary piece of $\frac{1}{8}$-in. sheet steel makes a good door, and this may be bolted to a 4-in. post set 2 ft. in the ground so that the door makes




FRONT

FIG. 3

a snug fit. Unless one has regular furnace fronts, this class of door is the very best that can be devised.

The furnace shown will hold sixteen 30-g. crucibles, but 12 work to perfection if set as indicated in the plan, and in the order shown, "drawing" from the side *D* and pouring No. 1 first, and so on to the end. This keeps the last crucible in the greatest heat until poured.

Should any of the charges seem to be slightly frozen, if the crucible be given a whirling motion in the flame for a moment, it will liquefy and pour readily, as a rule.

The top of the furnace, over the firebox, makes a handy place upon

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which to set the molds to receive the pour. A top finish of cement prepares it perfectly for this purpose.

This is a furnace that will give good results always, whether built with the utmost refinements of the art or as crudely as may be permitted.

The muffle, whether set in the furnace separately or in the chimney of the crucible furnace, should not be of a size less than 6½ by 10 by 16 in., outside measurements, and as nothing ever hits it, it will last for months, if care be taken with the bottom.

No draft is needed, as the heat can be controlled by stoking.

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# Shot Firing in Coal Mines by Electric Circuit From the Surface

BY GEORGE S. RICE AND H. H. CLARK,* PITTSBURG, PA.

(Pittsburg Meeting, October, 1914)

WHEN miners in the interior coal fields of the United States began the practice of blasting the coal without undercutting, or what is known as "shooting off the solid," many explosions resulted, some of them of great extent and violence, and attended with large loss of life. One of the means adopted for preventing this large loss of life, though not preventive of explosions, was to employ shot firers to discharge the shots when all the other employees had gone out of the mine at the end of the day's shift. This system still prevails in most of the Central and Southwestern mines.

Following the disaster at Winter Quarters mine, Utah, May 1, 1900, in which 200 lives were lost, a system of firing shots electrically from outside and when the men were out of the mine was adopted at the mines of the Utah Fuel Co.[1] This system has also been used in France in certain mines that are subject to instantaneous outbursts of carbonic acid gas. These outbursts generally occurred at shot-firing time, on account of the blasts releasing the gas held in crevices under high pressure.

The system was also adopted at a number of other mines in Utah, at the Cokedale mine, Colorado, at the Dawson mines, New Mexico, and in some mines of the Tennessee Coal, Iron & Railroad Co. in the Birmingham, Ala. district; and more recently in certain mines in Iowa and Kansas.

At first there was considerable trouble from misfires, and in a few instances from premature firing, while men were in the mine, through the shot-firing lines becoming crossed with power lines. Fortunately, however, no loss of life resulted from such premature firing within the knowledge of the writers, and latterly this danger has been reduced to a minimum by certain measures of precaution. The proportion of misfires, or failures of individual shots to fire, has also been reduced to a very small

* Non-member.

[1] Electrical Shot Firing in Coal Mines, *Engineering and Mining Journal*, vol. lxxxviii, No. 5, p. 243 (Jan. 30, 1909).

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number through better installation and arrangements of the shot-firing circuits, so that the system has proven very satisfactory, and has stood the test of a number of years' experience. The system and regulations employed in one mine are described in part in *Bulletin No.* 10 of the Bureau of Mines.

The need of the adoption of the system, under the conditions which prevail where coal is shot off the solid by black powder or dynamite, has recently been brought to the front by the large number of shot firers that have been killed, particularly in the Southwestern mines.

In some districts where there are restrictions on the quantity of black powder that may be used, and where a careful inspection is made of the holes before the shots are charged, there has been reasonable freedom from accidents; that is, only occasionally has there been an explosion, with the loss of a shot firer or two. This has been the situation in coal-mining States like Illinois and Iowa, but in Indiana, where larger charges of explosives are permitted by law, explosions resulting in the deaths of shot firers have been more frequent. In Kansas and Oklahoma, however, it is admitted on all sides that the methods of shot preparation have been extremely reckless, so that explosions have been very frequent. This has been due to the use of long drill holes, 6, 8 and even 10 ft. or more in depth, drilled straight into the solid, and sometimes loaded with both dynamite and black powder, such shots being used at the face of entries to crack and pulverize the coal in order that it may be easy to make a shearing. So many shot firers in this Southwestern district have been killed and with such regularity, that at some mines they have to be paid $10 per day for a shift of only 2 or 3 hr., and this price is none too high, as the risk is tremendous. One mine, for example, had three shot firers killed in as many explosions in one year and another two in a single month. Many of these explosions, particularly in the pitching beds of McAlester, Okla., which are often dusty and gaseous, are violent, resulting sometimes in wrecking the whole mine. As most of the mines in Oklahoma are on lands of the Civilized Tribes, who are the wards of the nation, the supervision of the leases is under the charge of the Indian Bureau, the U. S. Bureau of Mines acting in an advisory capacity; both bureaus are in the Department of the Interior.

In order to lessen the loss of life in the Indian lands from the use, or misuse, of explosives, and to lessen the number of mine fires and explosions caused by the use of black powder and dynamite, the Secretary of the Interior issued an order effective Aug. 1, 1914, afterward suspended to Jan. 1, 1915, as follows:

"On and after Aug. 1, 1914, except as hereinbefore provided, only such explosives as shall have passed the tests of the (United States) Bureau of Mines, and have been designated as 'permissible explosives,' shall be used in any coal or asphalt mine on

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the segregated coal and asphalt lands belonging to the Choctaw and Chickasaw Nations in Oklahoma.

"Permissible explosives shall be fired by detonators of a strength not less than No. 6 or otherwise as may be approved by the Bureau of Mines, and shall be used in quantities and under conditions approved by said bureau.

"Other kinds of explosives than 'permissible explosives' may beused in m ines in which the holes are loaded and the shots fired by special shot firers using an electric system from without the mine. The right is reserved to revoke this exception of the use of 'permissible explosives' if it shall be found that explosions of gas and dust are caused thereby resulting in loss of life or of coal through mine explosions or mine fires.

"During the months of June and July, 1914, and subsequently as may prove desirable the Bureau of Mines will supply on application, free of charge, the services of an expert to aid in demonstrating the best methods of using permissible explosives."

At a number of mines on Indian lands where the seam worked is flat, the operators are preparing to introduce undercutting machines and use permissible explosives, which are very effective where the coal is undercut, but which (with exceptions) cannot be used effectively in shooting off the solid, since they are quicker acting than black powder and their charge limit is 1½ lb. In hard or tough coal this is insufficient to throw down the coal without its having been undercut.

In pitching beds, under systems of mining now employed, there are difficulties in introducing the types of mining machines which are most popular. While it is probable that machines of equal efficiency may be developed which will be more suited to the conditions, and also that changes in the system of mining may be made which will permit some of the present semi long-wall or short-wall machines to be used, there are difficulties in agreeing upon labor scales to cover such new work, and therefore some of the operators on the Indian lands, to conform to the Department's order, have indicated their intention of introducing electric systems of shot-firing from outside the mine. It is on account of the prospective wider adoption of this system that it seems desirable to have the matter of specifications for the installation of the shot-firing system and the regulations governing it brought before the Institute, with a view to a discussion as to what such specifications and regulations should be, and with a view to making the system as efficient and safe as possible.

### *Electrical Shot-Firing Systems*

Electrical shot firing from the outside of a developed coal mine differs in general from similar shot firing from within a mine only in the number of shots fired at one time, and the greater distance to the shots from the point of firing. There are, however, several differences in the details of the equipment and its arrangement. Outside firing, as compared to inside firing, requires larger firing generators, larger conductors for distributing the current, better insulation on the conductors, and better insulators to support them.

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Since the main firing circuit of the outside firing system is a permanent one that extends over a large area and is exposed to various untoward influences and possible mishandling by many people, various switches and safety devices must be used in connection with the system.

It is also essential that there be provided effective means for insuring that no one is in the mine when the shots are fired from the outside.

*Factors Contributing to the Success of the System*

In order to insure safe and successful operation an outside system must be installed with careful attention to the following points:

1. Freedom from accidental connection with its source of power.
2. Freedom from stray currents from other sources of electricity.
3. Freedom from connection to the earth.
4. Freedom from short circuits between the two sides of the shot-firing line.

Freedom from accidental connection to its source of power requires that several switches located at different points should be connected in series with the line, and that these should be closed in predetermined order, and only by the man authorized to perform this duty.

Freedom from stray currents requires that only properly insulated conductors be used for the shot-firing circuits, and that such circuits be properly installed and protected from roof falls and where crossing or passing near other electric circuits.

Freedom from ground connections is desirable because such connections allow the firing current to be diminished by leakage, and at the same time provide a path by which stray currents may reach the firing circuits.

Freedom from short circuits on the firing line or from even moderate leakage between the two sides of the firing circuit is essential to the proper distribution of current to the detonators or fuses.

*Tentative Specification for an Outside Electrical Shot-Firing System*

There follows a tentative general specification for the various parts of a shot-firing system, which is subject to change whenever the wisdom of such modification is made manifest as a result of enlarged experience.

(1) *Generator.*—The generator should be a direct-current 250-volt compound-wound machine having a momentary capacity of about 30 kw.; 250 volts is recommended because this voltage will distribute the necessary current throughout the mine without necessitating the use of prohibitively large conductors and it is better than 550 volts because it will produce less strain upon the insulation of the line. However, 550-volt currents have been successfully used, and if there is a source of power of this voltage already installed in the mine power plant, it may be used with some saving in size of shot-firing conductors.

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(2) *Distributing Circuit.*—The size of the conductors used should be sufficient to pass at least 1.25 amperes through each room or heading circuit immediately upon closing the firing switch. This current will insure that three or four electric igniters or detonators (if perfect) will explode, even if they vary in resistance 25 per cent. above and below the average. With less current than 1.25 amperes, the Bureau of Mines has found by experiment that detonators 25 per cent. high in resistance will explode and open the circuit before detonators 25 per cent. low in resistance can become ignited.

(3) The conductors should be insulated with rubber if installed in damp places, but if installed in dry places may be insulated with triple-braid weatherproof insulation. However, as the humidity in coal mines is generally high it is advisable to use the rubber insulation.

(4) The conductors should be supported upon glass or porcelain insulators placed so near together that the wires do not touch anything. It is better not to install the circuits in the same entries with other electric wires, especially trolley lines, but preferably in the parallel entry, and when they have to cross other electric circuits, the shot-firing wires should be completely protected from contact with other circuits both under normal conditions and under conditions that might arise from roof falls and other contingencies. Where a trolley line must be crossed, the shot-firing lines should be protected by special timbering, or placed in a channel cut in the roof, if the roof is low.

(5) The wires that are used for connection between the room switch and the detonators will have to extend for a certain distance beyond the last insulator in order to allow for extensions without making frequent splices. These unsupported ends should be coiled up and hung on the post that supports the last insulator. From that post the wires will, of course, have to extend without support toward the face, but care should be taken that they do not make contact with the floor if it is wet. The ends of the wires should be scraped clean of insulation and the connection with the detonator leads made by cleaning the ends of the leads and twisting them tightly about the ends of the permanent room wires. The detonators in a room should be connected in series but the room and entry circuits should be connected in parallel.

(6) *Switches.*—There should be a double-pole single-throw switch at the entrance to each room or heading of entry. This switch should be mounted securely on a post with the hinges of the switch at the bottom. The room wires should be connected to the hinge or lower terminals of the switch and the power wires to the upper terminals.

(7) There should be at the mouth of each entry a double-pole single-throw 50-ampere switch mounted in the same manner as the room switch.

(8) At the foot of the shaft, slope, or bore hole where the shot-firing circuits enter the mine there should be provided two plugs with flexible

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lines not less than 5 ft. long to protect further the main circuit of the shot-firing system until all men in the mine have gone out. Provisions should be made for locking the plugs out of circuit.

(9) There should be located in the shot-firer's cabin, in a locked box, a switch that shall be thrown only by the authorized shot firer after all the men are out of the mine and after all other switches have been thrown in. This switch should be so designed that it can be thrown but once without operating a locked dog, tooth, cam latch, or other restraining device. The switch can be operated again only by unlocking the device. The authorized shot firer shall be the only person having a key to this lock.

(10) There should be located in the power house a switch in a locked box, used for connecting the shot-firing circuit to the generator or power line. This switch shall be thrown in only by the shot firer before closing the shot-firing switch, but not until all the men are out of the mine.

### *Explanation of Specifications*

Specifications (1) to (5) are self explanatory, but it may be advisable to make some further explanation of the paragraphs relating to switches and cutouts.

(6) With reference to the requirement of a switch at the mouth of each room or heading of entry, electrically it is of course not necessary to have such a switch, but to promote safety it is of the utmost importance. The miner in connecting up the shot-firing wires with the detonator or electric igniter leads at the end of his shift when he is about to go out of the mine, would, if there was any stray current through grounding on a power line in the entry, be exposed to the risk of premature firing.

It very much lessens the danger if there is a switch at the mouth of his room attached to a post in plain sight of the inspector or mine official going along the entry; and so placed that the miner is somewhat protected when the switch is thrown. The miner throws this switch in when he goes out of the mouth of his room or heading, after connecting up the shot-firing wires. The room and heading circuits are in parallel so that if by any inadvertence the miner fails to throw in the switch it does not cut out the other working places in the branch entry off which the rooms are turned.

As an additional precaution to prevent premature firing, switches are placed at the mouth of each branch entry, such branch switches being placed in a locked box, the key of which is in possession of the official in charge of the branch or district. This reduces the area affected by possible stray currents until these switches are thrown.

(8) The purpose of putting in the circuit a gap 5 ft. long, as specified above, at the foot of the shaft, slope, or bore hole where the shot-firing circuits enter the mine, is primarily to prevent the possibility of a dis-

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charge by lightning, which may strike the outer circuit entering the mine, and would jump an ordinary switch opening. This actually happened in a certain mine, evidently when the men had already left their working places, so that no accident resulted. This plan of using flexible lines 5 ft. long was then adopted at this mine.

(9) The reasons for the specifications for the switch in the shot-firer's cabin and power house are self evident, except that the reason for arranging the switch so that it may not be again accidentally closed, is to prevent a possible electric flash at the working face in case of grounding of the lines when dust and inflammable gas may have been liberated by the discharge of the shots resulting from the first switch closing.

### *Checking In-and-Out System*

An important part of an electric shot-firing system from outside the mine, is the method of keeping track of the employees entering the mine, to insure that the shot firing is not done until all of them have come out. At many mines, irrespective of the system of shot firing, the checking in-and-out system is used. Where such a system is employed, the check board should be kept in the shot-firer's cabin or adjacent to it.

One excellent method is to have the check board arranged so that every employee and visitor entering the mine may have a number; when the employee is working at the face his number will most conveniently be the same number as used for the checks which he places on mine cars he has loaded. These are generally iron checks; the man checks should be made of brass or other metal to distinguish them from the car checks. When no one is in the mine all of the man checks should be on the board, so that at a glance it can be observed whether or not there is any missing, which would mean one of two things, either that the man was in the mine, or had come out of the mine and gone home without leaving his check. There should be a penalty for non-delivery of the check.

The shot-firer's cabin should be so located that the men in coming out of the mine are compelled to pass the cabin, by means of fences, or by direction of officials stationed at the mouth of the mine. As a man enters the mine he is handed the brass check, and if it is a gaseous mine he may be given his safety lamp at the same time. When he comes out of the mine he gives up his check. When the shot-firing time comes the shot firer examines the board, then if there is any check missing a search is made for the man. There should be accessible for quick reference a book or list showing where each underground employee ordinarily works, if he has a regular station, which is the case with 90 per cent. of the employees.

The checking system, therefore, serves two purposes: One for the protection of the men from shot firing, and the other in case of an accident,

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such as a fall of roof upon a man working in an isolated place. When he does not appear at the end of the shift he will be sought for. By imposing suitable penalties on those who are careless either in coming out of the mine at the proper time, or in not giving up their checks when they have come out, it is very seldom, at mines where this or similar systems are used, that men fail to give up the check promptly.

When the shot firer has been informed by the underground officials that all the underground main switches have been closed, that the 5-ft. gap in the circuit where it enters the mine underground has been bridged by the flexible connections, and is sure that all the men are out, he proceeds to the power house, unlocks the switch box and closes the switch that connects the generator to the shot-firing lines; he then returns to the shot-firing cabin, opens the box containing the shot-firing switch and operates that but once. The reason for the mechanical locking of the switch in an open position after throwing, has already been explained.

In mines that use black powder or dynamite for shooting off the solid, fires are frequently started by the explosive, particularly if the mine makes a little inflammable gas at the face. It is, therefore, essential after shot firing that an inspection be made as rapidly as the state of ventilation permits, by a fire boss or other official, both to extinguish incipient fires, and to find out whether there have been any misfires. If there have been some shot failures it will be necessary to re-connect the shotsd the room after first having opened all the branch entry switches an that failed, switches on those branches where the shots failed. Then after reconnecting the switches leading to these shots, the balance being kept open, the officials come out of the mine and again try to fire those shots. In the event of a second failure it will probably be necessary to examine and test the shot-firing lines for grounds or other defects; and if none are found, to withdraw the explosive charges by the most approved means, and recharge, using fresh detonators or igniters.

Manifestly the success in handling such details will depend upon the efficiency of the mine organization; but as already stated, the percentage of failures in mines where the system has been tried for years is very small.

The object of having all the shots, which may be as many as three or four in a single room or heading, connected in series, is two fold:

(1) To prevent the amount of current falling below that which is necessary to discharge the detonators or igniters.

(2) To insure that either all of the shots in the room or heading be discharged, or none at all. This is because in shooting off the solid where a number of shots are likely to be prepared, especially if there are three or four, the success of some of these shots is dependent upon the success of the others; in other words, they are "dependent shots." In firing shot by shot, the order in which they go off can be assured; but

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while this undoubtedly is the best system, the next best method is that of having them all go off at once. It is true, it is possible to use delayed detonators or igniters. In this type, pieces of fuse of varying length are inserted, the electric current firing the fuse, and the latter in turn igniting the explosive; thus by regulating the length of fuse the shots may be put off in the order desired. However, the complications of this in the hands of the less experienced or intelligent miners make it somewhat dubious whether there is any material gain over firing simultaneously.

*Electric Shot Firing from Outside by Districts or Branch Entries*

A number of ingenious systems have been experimented with to fire shots in a mine from outside, sectionally. That is, first one pair of entries, then another pair, and so on, under the theory that this was less severe in its effects upon the mine. However, experience has indicated that the effect of discharging shots throughout the mine at one time is not any more severe than when the shots are fired by sections. In fact, at a large mine it is impossible to tell at the mouth when the shots have been fired, there being so little effect in the way of setting up an air wave. It would seem that the local pressures set up by the shots in the different rooms and entries neutralize one another so that there is no general concussion wave started. Further, there does not appear to be any serious difficulty with the roofs (apart from causing falls of roof), or blowing out of ventilating doors, stoppings, etc., as is liable to take place in the ordinary shot firing, shot by shot.

The expense of a new installation, apart from the generator, is from $1,000, in a mine of moderate size, to about $3,000 in a large mine.



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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Iron Industry in Brazil

BY E. C. HARDER, MILWAUKEE, WIS.

(Pittsburg Meeting, October, 1914)

## INTRODUCTION

FEW mineral deposits have in recent years attracted such general and widespread attention as the Brazilian iron-ore deposits, due mainly to the quantities of rich ore occurring here, in contrast to the ever-decreasing grade of ores shipped from many large producing iron-ore districts of the world. The general average grade of the iron ores shipped from the Lake Superior district[1] has decreased to approximately 50 per cent., natural, in recent years. The Rubio ore of Bilbao, Spain,[2] in 1890 averaged 55.5 per cent. metallic iron, dried; in 1902, 52.8 per cent., while at the present time a large proportion of the ores shipped from Bilbao are obtained by reworking the old dumps. In northern Sweden, where large magnetite deposits have only in recent years been extensively developed, selective mining is even now necessary in order to obtain high-grade Bessemer ores. During the last two years the iron ores of Chile have attracted much attention on account of the activities of the Bethlehem Steel Co. and others in the Chilean iron-ore region. Many of these ores are of good grade, yielding 67 and even 68 per cent. metallic iron, but judging from the writer's observations in this region a large proportion of them will be found to be of non-Bessemer grade.

The most important variety of iron ore in Brazil is hard, dense to specular hematite, occurring in Minas Geraes. A large number (89) of analyses of this type of ore have been averaged,[3] giving the following composition:

| | Per Cent. |
|---|---|
| Iron | 69.65 |
| Phosphorus | 0.0125 |
| Silica | 0.24 |
| Combined $H_2O$ | 0.38 |

[1] Personal communication from Prof. C. K. Leith.

[2] Whitwell, William: Presidential address, *Journal of the Iron and Steel Institute*, vol. lix, pp. 24 to 50 (1901).

[3] Communicated to the writer by H. K. Shearer.

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In very few places in the world has iron ore been found in marketable quantities which even approaches this ore in grade, while in Brazil several hundred million tons of such ore are in sight, occurring practically on the surface, while besides this rich ore there is an almost incredible amount of lower-grade ores, *i.e.*, ores of 60 per cent. metallic iron and over. It may be found impracticable, when mining operations com-

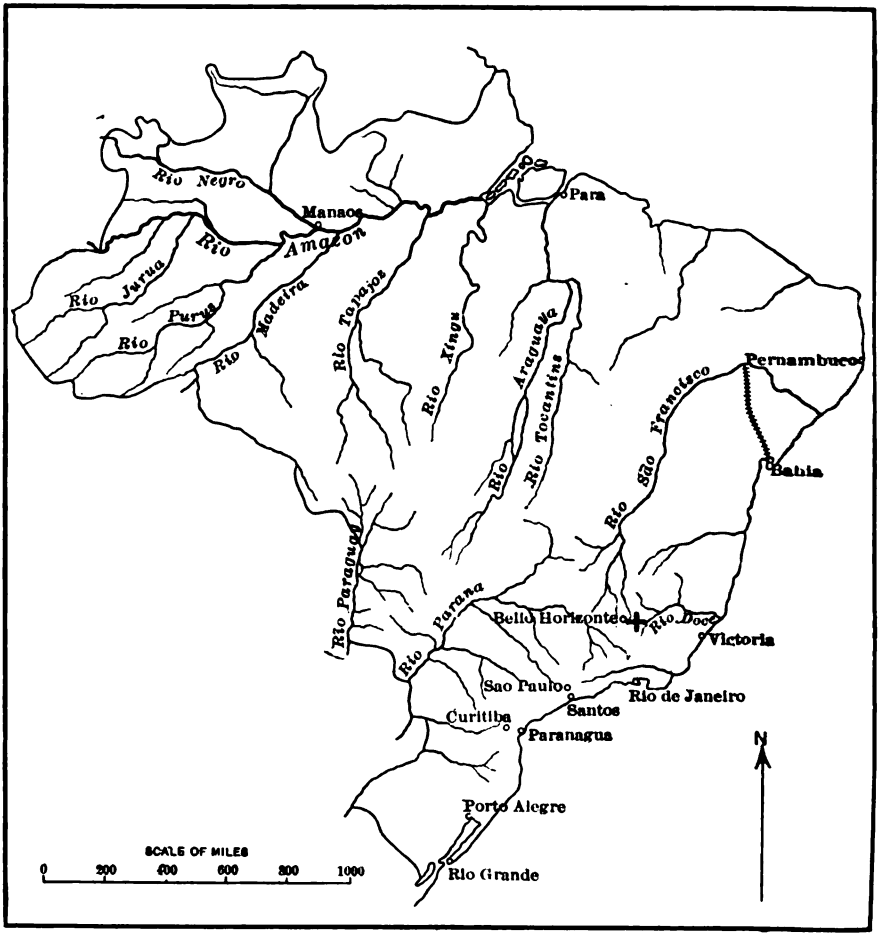


FIG. 1.—MAP OF BRAZIL, SHOWING LOCATION OF MINAS GERAES IRON-ORE DISTRICT.

mence, to mine exclusively the high-grade ores, and that admixtures of other ores will somewhat lower the general quality of marketed material; still it is safe to say that Brazil will be able to furnish for many years ores of Bessemer quality which will average more than 68 per cent. metallic iron.

Up to the present time no iron ore has been exported from Brazil, and

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only a very insignificant quantity has been used to supply the small domestic furnaces. Plans have been made to ship a small quantity of ore from one of the mines in Minas Geraes during 1915, but it will probably be several years before any important quantities are exported. Conditions are not yet ripe in Brazil for the establishment of an iron-smelting industry of importance within the country. As there is no coal at present known in Brazil fit for smelting purposes, coal would have to be imported from Europe or North America, and since Brazil cannot afford a market for any great quantity of manufactured articles of iron, it would be necessary to export the greater part of the iron to other countries, thus necessitating double transportation, the bringing in of fuel and the taking out of manufactured iron. For this reason it is proposed to export the iron ore to foreign smelters where the fuel supply and the markets for the manufactured products are close at hand. Eventually, as the demand for iron in South America increases, smelting plants will doubtless be established in Brazil to supply the local market.

## History of the Iron Industry

The history of the Brazilian iron industry is centered around two regions: (1) Ipanema, in the State of São Paulo, where small deposits of titaniferous magnetite have been operated intermittently since the beginning of the seventeenth century; and (2) Minas Geraes, where the extensive bodies of hematite occur which now seem destined to play an important part in the iron-ore industry of the world.

Dr. Derby[4] gives the following interesting history of the iron industry in São Paulo:

"The colonial records of Brazil register the fact that about 1590 an exploring party that set out from the town of São Paulo, founded about 40 years before, reported the finding of iron ore in a mountain situated 100 kilometers to the southward. Gold and silver were also reported from the same region, and acting on this information the Portugese Government took measures to promote the mining industry in the colony by sending out, in 1597, officials especially charged with this mission. The inclusion of an iron founder in the party indicates a special interest in the discovery of iron ore. One or two small forges were set up which commenced to produce iron probably about the year 1600, and continued in activity to about 1629. The place subsequently took the name of Ipanema which has ever since been inseparably connected with the long, though not brilliant, history of the iron industry in Brazil. There is a reasonable probability that the iron produced here was the first to be manufactured on the American continent.

"About the same time a forge was established close to the town of São Paulo in order to work the argillaceous ore that abounds in the vicinity, but it does not seem to have had a prolonged existence. . . . . .

In 1765 the production of iron was resumed at Ipanema in São Paulo, but again

[4] Derby, O. A.: The Iron Ores of Brazil, *Iron Ore Resources of the World*, p. 813 (Stockholm, 1910).

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abandoned after a few years. An attempt made in 1800 to revive the industry, this time by means of a high furnace, was unsuccessful, and in 1810 a Swedish metallurgist, under contract with the Government, constructed four direct-process furnaces which continued in operation until 1818, when two high furnaces constructed by the German engineer officer, Frederic von Varnhagen, then in the service of the Portugese Government, were put into operation. These continued in blast, under Government administration and with a daily production of 3 to 4 tons until 1895."

In Minas Geraes we have the first definite information of the production of iron about the beginning of the nineteenth century, although it is reasonable to suppose iron was manufactured previous to this date by primitive African methods, with which many of the slaves imported for the purpose of working the gold mines in this region were probably familiar.[5] Dr. Derby[6] states that when Eschwege[7] arrived in the district in 1811 most of the smithies then existing produced their own iron, either in a small furnace constructed for the purpose, or by the spoonful in the ordinary backsmith's forge.

Toward the end of 1812 a company formed under the direction of Eschwege began to produce iron at the rate of about 1 cwt. per day in a direct-process plant erected near Ouro Preto. After this the process spread rapidly, and before long the entire gold-mining district of Minas Geraes was dotted with little iron furnaces. In 1853, John Monlevade[8] a French engineer, then operating in Minas Geraes, estimated that 84 plants producing iron were located in the region between Ouro Preto and Itabira de Matto Dentro. These employed at least 2,000 persons, free and captive, and produced from 145,000 to 150,000 *arrobas* (2,175 to 2,250 tons) of iron yearly. Derby states that the total number of plants in the district in 1864 was estimated at 120.

Blast-furnace operations have not been extensive in Minas Geraes. Dr. Derby[9] states that an attempt was made to erect a high furnace at Morro de Pilar between 1809 and 1814 but it was unsuccessful.

In 1888, a charcoal furnace about 30 ft. in height and capable of producing about 5 tons of pig iron per day was erected at Esperanca[10] near Itabira do Campo on the Central Railroad of Brazil, and a few years later a similar furnace with a foundry in connection was built at Miguel Burnier, some distance south on the same railway. The Miguel Burnier furnace has been idle for many years but the foundry has been operating

[5] Derby, O. A.: *Op. cit.*

[6] Derby, O. A.: *Op. cit.*, p. 314.

[7] Baron W. L. von Eschwege, a Prussian military engineer, employed by the Portuguese government during the years 1810 to 1822 to develop the mining industry of Brazil. See *Trans.*, xxxiii, 407 (1902).

[8] Monlevade, João Antonio de: Letter to President of Province of Minas Geraes, Dec. 12, 1853.

[9] Derby, O. A.: *Op. cit.*

[10] Scott, H. K.: The Iron Ores of Brazil. *Journal of the Iron and Steel Institute*, vol. lxi, p. 248 (1902).

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with pig iron produced at Esperanca. The Esperanca furnace has been operated almost continuously to the present, using principally rubble ore obtained from the hills near Miguel Burnier. At the present time it is the only blast furnace in South America actively producing iron.

## Direct Process of Iron Manufacture

There are at the present time about a dozen small plants in the Minas Geraes iron-ore district engaged in manufacturing iron from the ores by the direct process, that is, without the addition of fluxing material.

For smelting the ore by the direct process two types of furnaces are used: (1) the closed or crucible furnace, and (2) the open or Italian furnace.

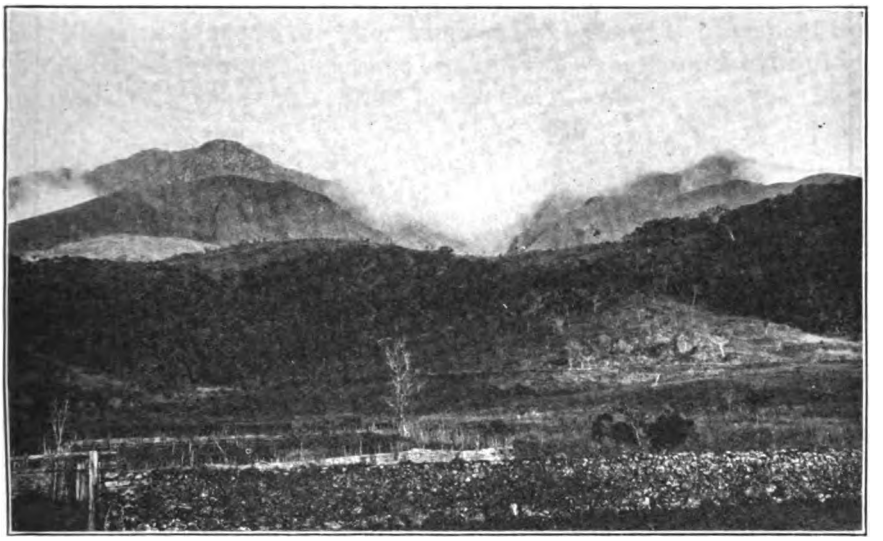

*Photo by H. K. Shearer*

Fig. 2.—Serra do Caraca at Alegria. The ridge on the skyline consists of quartzite; the bare ridge in the middle background is iron formation.

While nearly all the plants have furnaces of both types, the crucible furnaces are more extensively used. These are generally arranged in series of four, six, or eight. A series of four furnaces will have about the following outside dimensions: 4 ft. high, 5 ft. wide and 10 ft. long. The opening or crucible itself is a vertical, cylindrical cavity about 2 to 3 ft. in depth and about a foot in diameter. From the base of the crucible an opening extends through the front wall through which the bloom is extracted after firing. In the rear wall of the crucible is a small opening which admits the blast, produced by water power.

The smelting operations consist in filling the crucible with successive layers of charcoal and crushed or pulverized iron ore, slightly dampened, then igniting and applying the blast. More ore and charcoal are gen-

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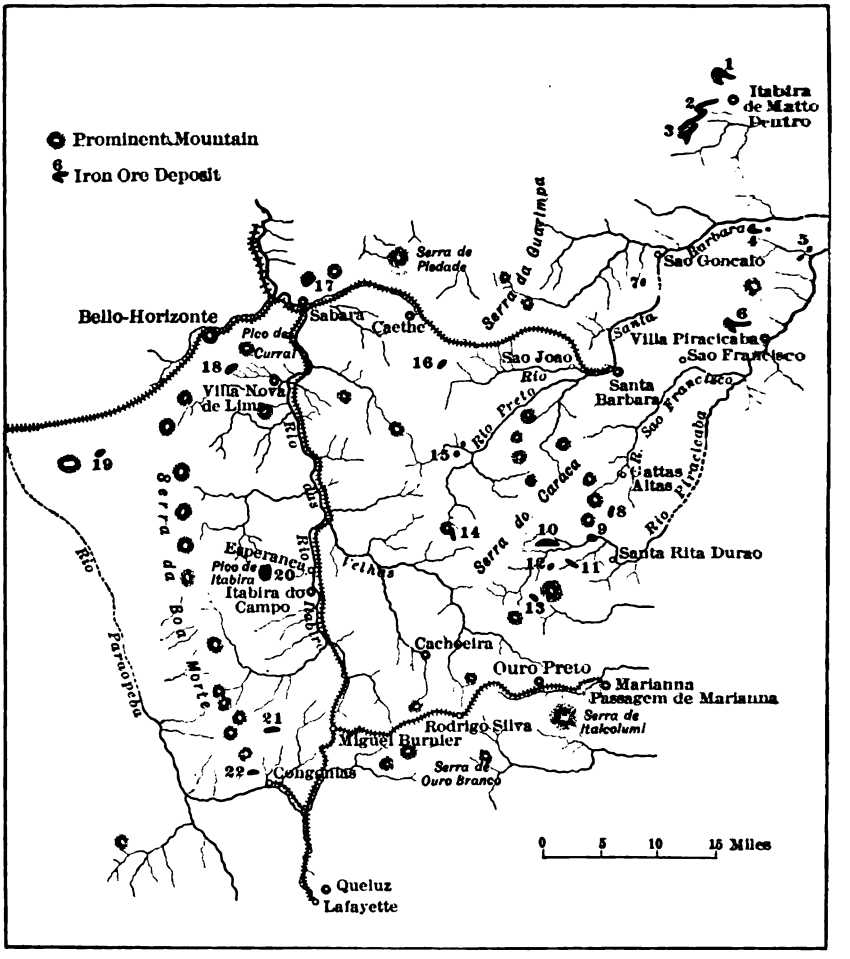
erally added as the charge settles until a bloom of upward of 25 lb. is produced, the size depending more or less upon the size of the hammer with which it is to be worked. The bloom consists of a mixture of metal-




1. Caue. 2. Esmeril. 3. Conceicao. 4. Andrade. 5. Monlevade. 6. Morro Agudo. 7. Cocaes. 8. Bananal. 9. Morro da Mina. 10. Alegria. 11. Fabrica Nova. 12. Germano. 13. Timbopeba. 14. Capanema. 15. Mutuca. 16. Gongo Soco. 17. Gaya. 18. Aguas Claras. 19. Jangada. 20. Catta Branca. 21. Fabrica. 22. Casa de Pedra.

Fig. 3.—Map Showing the Location of the Principal Iron-Ore Deposits in Minas Geraes.

(Base adapted from a map by Gonzaga de Campos.)

lic iron, carbon, and slag. This mass, in a semi-molten condition, is pounded with a trip hammer, operated by an overshot water wheel, for the purpose of eliminating the slag. Very frequently it is necessary to reheat and

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hammer a bar of bloom two or three times in order to make it fairly free from slag, the reheating being done in an ordinary forge.

The Italian furnace differs from the crucible furnace in being open, having only a floor and back wall. In the floor is a hollow or basin into which ore and charcoal are fed and in which the bloom is formed. The blast enters through the rear wall.

An ordinary direct-process furnace will produce about four bars of bloom daily, each 25 lb. and upward, or between 100 and 150 lb. of iron per day. One or two of the existing plants have about 20 lb., which makes the output of such a plant, when in continuous operation, nearly a ton of iron per day. As a rule, however, only a limited number of furnaces are operated at the same time.

The bars of bloom are later hammered into long, thin wrought iron strips and these are manufactured into horse shoes and various agricultural implements, such as hoes, brush hooks, etc.

## Fuel

Good fuel for smelting purposes is scarce in Brazil. Charcoal is usually made of poor wood which remains after the hardwoods and timber resistant to decay have been cut for building and other purposes. The wood remaining after this valuable timber has been removed is usually soft and yields a poor quality of charcoal, which serves the purpose for direct-process smelting but is unsatisfactory for blast-furnace operation.

Coal occurs in the States of Sao Paulo, Parana, and Santa Catharina in stratified rocks of probable Permo-Carboniferous age. It is said[11] to be of very poor quality, however, containing a large percentage of pyrite which makes it entirely unfit for smelting purposes. Other discoveries of coal are reported from the Amazon basin but these have not yet been proved to be of economic importance.

## The Iron-Ore Deposits

The iron ores of Brazil are of two distinct types: (1) magnetite ores associated with igneous rocks occurring in the States of Minas Geraes, Sao Paulo, Parana, and Santa Catharina; and (2) hematite ores associated with metamorphosed sedimentary rocks in the State of Minas Geraes.

### *Magnetite Deposits*

The magnetite ores are of relatively little importance, occurring only as small scattered deposits. The best known of these are found at Sao

[11] Communicated to the writer by Dr. O. A. Derby.

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Joao Baptista, Minas Geraes; Ipanema and Jacupiranga, Sao Paulo; Antonina, Parana; and Joinville, Santa Catharina. Other deposits are reported elsewhere in Parana and Santa Catharina and also at localities in the States Rio de Janeiro, Minas Geraes, and Espirito Santo.

The Sao Joao Baptista deposits contain some magnetite of very high grade and are saidto be extensive. They occur west of Oliveira station on the West of Minas Railroad in south-central Minas Geraes. A specimen of ore gave the following analysis.[12]

| | Per Cent. |
|---|---|
| Fe | 72.24 |
| P | 0.017 |
| Siliceous matter | 0.52 |
| $TiO_2$ | nil. |

The Ipanema and Jacupiranga deposits consist of titaniferous magnetite occurring in pyroxenite associated with nephelinite and apatite-pyrone rocks.[13] As is the case with most ores of this class every gradation exists between pure pyroxenite rock and iron ore. The content of titanic oxide in the Ipanema ore is said to be relatively low, but that in some of the Jacupiranga ores is said to be as high as 20 per cent. The Jacupiranga deposits are found in the southeastern corner of Sao Paulo near the sea coast, while those of Ipanema are located in the interior in the south-central part of the State.

The Antonina deposits occur in northeastern Parana close to the shore of a bay which extends northward from the port of Antonina. Eight or 10 different orebodies occur arranged at intervals along an approximately northeast-southwest line. The bodies are in the form of rough lenses inclosed in a basic igneous rock resembling diorite. The ore varies in grade; locally, masses of pure magnetite occur, but generally the ore consists of mixed magnetite and rock. A sample of fairly good-looking ore from one of the larger orebodies gave the following analysis.[14]

| | Per Cent. |
|---|---|
| Fe | 64.00 |
| P | 0.031 |
| $SiO_2$ | 6.94 |
| $H_2O$ (combined) | 0.63 |
| Mn | 0.20 |

Some of the Antonina orebodies consist of manganiferous iron ore, samples taken running as high as 25 per cent. manganese.[15]

The Joinville magnetite deposits are found about 16 miles southwest of Joinville and apparently consist of a number of scattered lenses in an

[12] Analysis by T. H. Lee, Servico Geologico e Mineralogico do Brazil, Rio de Janeiro.

[14] Analysis by H. K. Shearer, Brazilian Iron & Steel Co.

[15] Analyses by T. H. Lee, Servico Geologico e Mineralogico do Brazil, Rio de Janeiro

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area underlain by granite and gneiss. The outcrops occur in the lower foot hills of the Serra do Mar, and are very near the coast, being on the edge of the belt of flat lowlands running parallel to the coast line. A sample taken over one of the outcrops gave the following analysis.[16]

| | Per Cent. |
|---|---|
| Fe | 71.30 |
| P | 0.041 |
| $SiO_2$ | 0.45 |
| $H_2O$ (combined) | 0.19 |
| Mn | 0.43 |

### *Hematite Deposits*

The hematite deposits associated with the metamorphosed sedimentary rocks in Minas Geraes are the only known iron ores in Brazil to which any considerable importance is attached. They occupy a large area in the south-central part of Minas Geraes, being practically coextensive with the sedimentary formations in this region. The area within which the principal deposits are found is roughly 100 miles long from northeast to southwest, and about 60 miles in width. Within this region the orebodies are more or less segregated at certain localities.

The sedimentary series in which the iron ores occur is of probable pre-Cambrian age and rests unconformably on a basement of igneous rocks consisting of gneiss, granite and various crystalline schists. The lowest member of the sedimentary series is a massive quartzite of great thickness which is overlain by a thin discontinuous argillaceous schist. Above this schist is the remarkable Minas Geraes iron formation, which consists largely of a laminated, iron oxide-bearing quartzite known as itabirite, but which contains also beds of ferruginous schist and layers and lenses of iron ore. The iron formation is overlain by an extensive formation of schist and quartzite which contains iron formation and limestone beds in the lower part and which grades up into a formation consisting of quartzite with some schist, the highest known member of the sedimentary series.

Itabirite, the principal constituent of the iron formation, generally consists of alternating layers of quartz sand and iron oxide, but locally it consists of a granular mixture of quartz sand and iron oxide. It probably makes up more than 95 per cent. of the iron formation, the remaining 5 per cent. consisting of ferruginous schist and iron ore. Itabirite rarely contains less than 30 per cent. metallic iron and from this it grades up, with diminishing quartz sand, into ore, the lower limit of which may be taken as 50 per cent. metallic iron. Occasionally itabirite, very low in iron, apparently grades down into ferruginous sandstone or quartzite.

[16] Analysis by H. K. Shearer.

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The iron formation varies greatly in thickness, in places being less than 50 ft. and elsewhere more than 4,000 ft. thick. The iron ore and ferruginous schist occur interstratified with the itabirite as beds or lenses varying in length and thickness. The iron ore appears to be a true sedimentary formation,[17] laid down at the same time as the inclosing rocks and later metamorphosed with them. The iron-ore beds have the same strike and dip as the inclosing rocks and outcrop with them at the surface. In many places hard-ore layers form the tops of hills or ridges or form conspicuous cliffs along the hillsides, due to their greater resistance to erosive processes.

When the iron formation weathers at the surface a blanket consisting of a mixture of itabirite and ore fragments cemented by limonite is formed. This blanket may vary in thickness from a few inches to more than 50 ft., and spreads over the surface of the iron formation as well as over the surface of adjacent formations such as schist or granite. This is the iron-ore conglomerate or breccia termed *canga*.

The so-called "itabirite ores" of Brazil are therefore of two distinct classes: (1) those occurring as original beds or lenses in the iron formation and (2) those resulting from the weathering and concentration of the iron formation. They may be classed as follows: Original or bedded ores: (1) hard massive ore; (2) soft powdery ore; (3) laminated or thin-bedded ore. Concentration ores: (1) canga (iron-ore conglomerate); (2) stream sand and gravel ores; (3) rubble ore; (4) enriched itabirite; (5) leached carbonate.

Of the above classes the bedded ores are by far the most important, especially the hard massive ore. This is of high grade and is the only type of ore being considered at present in plans dealing with the development of the Brazilian iron industry. The soft powdery ore is also of high grade, but on account of its consistency it is at present considered undesirable. The laminated ore is of lower grade than either of the other types of bedded ore. However, there occur in different parts of the district enormous deposits of this ore which will ultimately be of great importance.

Of the concentration ores the canga is the only type of importance. While the canga blanket covers many square kilometers, its grade is such that it cannot be considered desirable for the iron industry for many years to come.

Below are given the average range in iron and phosphorus content of the principal types of ore, assuming 50 per cent. metallic iron as the lower limit for iron ore. These figures are based on extensive sampling, much of it under the direction of the writer.

[17] Leith, C. K., and Harder, E. C.: The Hematite Ores of Brazil and a Comparison with the Hematite Ores of Lake Superior, *Economic Geology*, vol. vi, No.7, pp. 670 to 686 (Oct.-Nov., 1911).

Harder, E. C.: The Itabirite Iron Ores of Brazil, *Economic Geology*, vol. ix, No. 2, pp. 101 to 111 (March, 1914).

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| | Iron, Per Cent. | Phosphorus, Per Cent. |
|---|---|---|
| Hard massive ore | 69 to 70 | 0.003 to 0.020 |
| Soft powdery ore | 50 to 69½ | 0.004 to 0.05 |
| Laminated ore | 50 to 68 | 0.003 to 0.07 |
| Laminated ore (hydrated) | 63 to 67 | 0.05 to 0.3 |
| Canga | 50 to 65 | 0.1 to 0.3 |

The hard massive ore is hematite, generally dense or finely specular, but occasionally, where strongly metamorphosed, it is coarsely crystalline and has an admixture of magnetite. It occurs in beds varying up to more than 450 ft. in thickness and to more than two-thirds of a mile in length. Hard ore rarely contains more than 1 per cent. of silica and its metallic iron content is remarkably constant.

The soft powdery ore is specular hematite in a fine friable form, so that most of it when dried crumbles to dust which will pass through a 100-mesh screen. Hard ore and soft ore frequently occur intermixed with each other, lenses or irregular bunches of soft ore occurring in hard-ore beds or irregular masses of hard ore occurring in soft-ore deposits. Soft-ore deposits are in general much smaller and more irregular than hard-ore deposits. Both occur as lenses or beds interlayered with itabirite or laminated ore.

Laminated ore consists of earthy to specular hematite. It is friable and breaks up into thin plates parallel to the lamination. It is also quite porous, allowing free circulation of ground water, on account of which large portions of these deposits are hydrated, especially near the surface, and consist of a mixture of hematite and limonite. Unhydrated, laminated and soft powdery ores grade from almost pure iron oxide, with little or no quartz sand, down to itabirite with more than 30 per cent. silica. The line between itabirite and ore in these cases is purely artificial. In the hydrated portions of the laminated-ore deposits the silica has generally, in large part, been removed, so that, although somewhat lowered in grade by the hydration, its silica content is generally small and the ore is fairly constant in grade.

Unhydrated ores of various types and even itabirite contain a very small percentage of phosphorus. It appears that phosphorus becomes concentrated during the hydration, either being brought in from the outside or remaining while other materials are being removed. Ore hydrated to beyond 2 per cent. combined water increases rapidly in phosphorus content. The hydrated portions of laminated-ore deposits rarely go to a greater depth than 50 or 75 ft. except along occasional more porous layers.

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Surface weathering has but little effect on hard ore and soft powdery ore. The former is sometimes slightly hydrated along cracks, for a few inches from the surface, while the latter is generally altered to canga to a depth of 10 to 15 ft. and below this somewhat hydrated for an additional 4 or 5 ft. The laminated ore is affected to a greater depth by surface agencies, and along with the hydration there is removal of silica. This, however, rarely is sufficient to make an ore out of an itabirite, so that but a small percentage of this material can be classed with the concentration ores.

Canga consists of material derived by weathering from the iron formation. The canga blanket is generally thin along the tops of the iron-formation ridges and thick on the lower slopes, due to material being transported down the slopes and deposited near the base. Where iron-formation hills rise above the bordering schist or granite areas the canga blanket extends out over these areas. The best grade of canga occurs on the iron-formation areas, especially where ore occurs with the itabirite. Frequently such canga is made up, to a large extent, of irregular fragments of ore cemented by limonite and hydrated hematite. The canga overlying areas of other rocks is of lower grade than that overlying iron-formation areas, containing a large proportion of impurities such as clay (perhaps containing bauxite, etc.) and silica. The farther away from the iron formation the poorer is the canga, until finally it becomes merely an ocherous clay. Over and close to iron-formation areas the fragments in the canga are usually angular, farther away they are small and rounded and finally they disappear. Thus there is every gradation of canga, from ferruginous clay to iron ore with 65 per cent. metallic iron.

Of the other classes of concentration ores the only ones of importance are rubble ores and leached itabirite.

Rubble-ore deposits occur as talus under extensive outcrops of hard ore. They are simply masses of hard-ore fragments broken off from the outcrops, and are of the same grade as the bedded hard ore from which they are derived.

Where canga overlies compact thin-bedded itabirite it frequently retains the itabirite texture though most of the silica is removed. This canga *in situ*, exhibiting original textures, is rarely more than a few feet in thickness, being underlain by sandy itabirite. It is leached itabirite.

Stream sand and gravel consisting of hematite occur along some of the streams cutting through the iron-formation belts. They are rarely pure enough to be called ore.

Leached carbonate is an earthy material consisting of hydrated iron or manganese oxides, or a mixture of the two. It is derived from the decomposition of impure limestone beds occurring in the upper part of the iron formation and in the lower part of the overlying schist. It is seldom rich enough in iron to be called ore.

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## TRANSPORTATION

In relation to such enormous quantities of high-grade iron ores as exist in Brazil, the principal problem is, of course, that of transportation. For more than 25 years the Central of Brazil Ry., owned and operated by the Brazilian government, has run northward from Rio de Janeiro through the center of the iron-ore district. At the present time it continues far to the north to Pirapora on Rio Sao Francisco, from which point it is possible to travel by steamboat and railway to Bahia. The Central of Brazil is a broad-gauge line (1.6 m.) from Rio de Janeiro as far as Miguel Burnier in the southern part of the iron-ore district, a distance of about 310 miles, and beyond this point it continues northward with a narrow gauge (1 m.). From this point also a branch line runs eastward to Ouro Preto and Marianna while another branch line runs eastward from Sabara to Caethe and Santa Barbara, both in the iron-ore district. The latter branch is being continued to Itabira de Matto Dentro. From General Carneiro, north of Sabara, a branch line runs southwestward to Bello Horizonte, the capital and principal city in Minas Geraes. A broad-gauge line is now being constructed from Lafayette, a station south of the iron-ore district, to Bello Horizonte by way of Rio Paraopeba, west of the iron-ore district. This will make a direct broad-gauge connection between Rio de Janeiro and Bello Horizonte.

The Leopoldina Ry. has a network of lines in the southeastern part of Minas Geraes and in the neighboring States of Espirito Santa and Rio de Janeiro. Saude, the terminus of one of its northernmost branches, lies about 20 to 25 miles east of the eastern border of the iron-ore district. The Leopoldina Ry. has a gauge of 1 m.

The West of Minas Ry. has a branch line of 1-m. gauge which runs into Bello Horizonte from the west. The main line of this road, however, is some distance west of the iron-ore district.

The Central of Brazil Ry. was not planned as an ore-carrying road. It crosses two divides between the iron-ore district and Rio de Janeiro, one south of Barbacena, known as Serra da Mantiqueira, and the other north of Rio de Janeiro, known as Serra dos Orgaos. Heavy grades and sharp curves abound over large portions of the line. The Brazilian government, however, has taken several contracts for hauling iron ore over this road to Rio de Janeiro from points in the western part of the iron-ore district at the rate of $1.50 per ton. Small shipments of manganese ore are being transported over the road at the present time, and judging from difficulties encountered in this traffic, it seems probable that much reconstruction will be necessary before this railroad can handle even a small fraction of the iron ores which must ultimately be transported from the district.

The Leopoldina Ry. and the West of Minas Ry., on account of their

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sharp curves and prohibitive grades, are not likely to enter into the ore-carrying traffic.

It seems probable, therefore, that by far the greater part of the ore is destined to leave the district by another route than those mentioned, viz., by the Victoria to Minas Ry., which leaves the sea coast at Victoria and runs westward up the Rio Doce. This railroad, which is now under construction, will enter the iron-ore district from the northeast and will tap first those deposits which are now farthest from lines of transportation. The distance to Victoria from the eastern part of the iron-ore district over this route is about 375 miles, the entire distance being on a down grade following the course of the Rio Doce. About 275 miles of the line has been constructed, but the greater part of this will probably need reconstruction since it was not originally planned to make this an ore-carrying road.

In connection with the rapid development of the district it may be of interest to note that previous to 1910 most of the iron-ore deposits were in the hands of Brazilian owners, many of whom were descendants of persons to whom the original land grants were given more than 100 years ago for purposes of gold mining, while a few deposits were owned by the gold-mining companies operating in the district.

At the present time the principal deposits in the eastern part of the district, that is, in the part tributary to the Victoria to Minas Ry., are controlled by English, French, and American interests, who have done extensive exploration and development work. In the western part of the district some deposits are in the hands of Brazilians, as are also a few minor deposits in the eastern part, but here foreign interests, including the German, have made extensive purchases as well and have done much development work. It is from this part of the district that small shipments of ore are expected to be made in 1915 over the Central of Brazil Ry.

## Labor and Wages

A serious difficulty to be contended with in extensive mining operations in Brazil, is that of labor. The wages are low, ranging from about 75c. to $1.25 per day for ordinary labor and about twice as much for skilled labor, but the difficulty is not only in obtaining laborers, but in keeping them. The gold mines in the district are in constant difficulty in this respect, and unsuccessful attempts have been made to import Spanish labor, while recently one of the mining companies imported a number of Japanese laborers. The native Brazilian laborer is averse to working underground and prefers agricultural work with lower wages to mining work. As the iron-mining operations will, however, be largely of the nature of quarrying, there should be less difficulty in obtaining and retaining laborers than is experienced in the gold mines.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the ittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, ien discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 9th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Pittsburg Coal Field in Western Pennsylvania

BY H. A. KUHN, PITTSBURG, PA.

(Pittsburg Meeting, October, 1914)

THE Pittsburg coal field in western Pennsylvania is conceded to be the most important in the world. To measure its importance it is necessary to understand the extent of its service in the various industries of the country. Probably 90 per cent. of the pig iron manufactured in the United States up to the present time has been made by using coke manufactured from the Pittsburg coal seam in western Pennsylvania. This coal field is the foundation on which the city of Pittsburg rests and is the reason for the great growth of the iron industry in the Pittsburg district. Iron ore is brought to this district, not because Pittsburg is a natural location, over other locations, for the iron and steel industry—as its position places the manufacturer at the mercy of the railroads—but the ore is brought 1,100 miles to meet the fuel. It can be said that the illuminating-gas industry in the United States has used this coal exclusively to the same extent that the pig-iron maker has used it. It may be also said that 20 to 25 per cent. of the fuel used on railroads in the United States comes from this coal field. The Pittsburg coal field is unquestionably the center of the industrial population of the United States, for in addition to the industries of the district and those closely adjoining, it has tributary to it all the cities and industries along the Great Lakes and practically all of Canada with the exception of the extreme western and eastern ends. It supplies the industries and population west of Duluth and Superior many hundreds of miles. This coal is floated down the Ohio and Mississippi rivers, supplying the towns *en route*, and is delivered in New Orleans, a distance of 2,200 miles, for approximately 80 to 90 c. per ton transportation cost. It is delivered on the docks of Superior and Duluth at a cost of transportation 50 c. a ton less than the cost of transporting the same coal from Pittsburg to a local consumer in Philadelphia. With other Appalachian coals it has large markets east and along the seaboard, especially for byproduct-coke making and the illuminating-gas industry. It is considered the premier railroad fuel of the world on account of the fact that this coal in a given-sized locomotive will probably haul more cars than any other coal in the



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world. Tests made at Altoona by the motive-power department of the Pennsylvania Railroad Co. show that Pittsburg gas coal evaporates as high as 18.9 lb. of water per square foot of heating surface. It is stated that the lower volatile coals, with a theoretically higher heat value per pound of fuel, do not evaporate more than 12 or 13 lb. of water per square foot of heating surface. For this reason this important railroad has adopted this coal as its standard fuel. By its use with the same crew and engine a maximum number of cars may be hauled.

These statements explain somewhat the reason for the enormous production of coal from this field and seam.

The production of coal from the Pittsburg seam in western Pennsylvania in 1913 was in round figures approximately 100,000,000 tons; in Ohio from the same seam of coal approximately 20,000,000 tons; in West Virginia 10,000,000 tons, making a total tonnage of approximately 130,000,000 out of a total production of 470,000,000 tons of bituminous coal produced in the United States in the same year. Of the 170,000,000 tons of bituminous coal produced in Pennsylvania approximately 100,000,000 tons comes from the Pittsburg seam. Fayette and Westmoreland counties with only a small acreage left of unmined coal are now producing approximately 60,000,000 tons of coal annually; Allegheny and Washington counties produce about 20,000,000 tons each; Greene county has only recently started as a producer of coal but a capacity is now established in this county which is upward of 2,000,000 tons a year.

The geology of the Pittsburg coal measure is so well understood that it is not worth while discussing this field from that standpoint. Some practical features, however, which relate to its geology, are interesting. The main feature of this seam from a geological standpoint is its persistency and integrity over the area which it covers. Whether the coal has been opened or not, those who are familiar with the coal in each zone or subdistrict (and the slow but uniform change from one basin to the next) can state unreservedly how thick the coal is, how much sulphur it will probably contain within a small percentage and about how many tons per acre may be recovered. Thousands of miles of entries have been driven through this coal field in mining operations of the district and thousands of oil wells and gas wells have been drilled through it where it is not being mined. Several facts touching the deposit of this coal seam are worth mentioning. It is evident from an inspection of the geological maps of Pennsylvania, Maryland, and West Virginia that the plant life which formed this field of coal was extremely luxuriant, increasing in depth or density toward the east, and that the purest part and largest part of this coal field has been lost through erosion. This seam in Georges Creek field near Cumberland, Md., measures as much as 16 ft.; in its western outcrop in Ohio about 3½ to 4 ft.; in the northwestern end

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of Allegheny county in western Pennsylvania near the State line about 4½ ft. of the main or working bench of coal and 2½ to 3 ft. of roof coal measures, the working bed being separated from the roof measures by what is known as the "draw slate" of the district, which averages from 8 to 14 in. In the Connellsville field the coal at places attains a thickness of 8 ft. and on the southern boundary of Greene county near the Monongahela river it measures 9 to 10 ft. in thickness. In the southern extension of this seam in West Virginia the roof coal and main or working bed of the Pittsburg district become practically one solid bench of coal with only thin laminations of slate marking the divisions which are so noticeable in the northern part of the field. It may be said from an inspection of the geological map and from a knowledge of the quality of the coal that the plant growth that formed this coal was much more luxuriant in the eastern end and that the coal was deposited with less impurities and under quieter conditions. The coal thins going west, but the extreme purity of the coal extends westward to the Pinhook anticlinal running southwest from Pittsburg through Allegheny and Washington counties; and west of the Pinhook anticlinal the change is gradual, the coal becoming more impure, higher in ash and sulphur with the distance westward, and the working bed or bench becomes thinner. The coal containing the least impurities from this seam in the Pittsburg district is mined from what is known as the "Gas Coal Basin," which lies between the Pinhook anticlinal and the Murraysville anticlinal on the west and the Waynesburg and Saltsburg anticlinal on the east. The working bed of coal in this basin averages about 6 ft. thick. Above this bed and separated from it by the draw slate, 2 to 4 ft. of roof coal is deposited. The main working bed in this basin is prepared for market containing about 6 per cent. non-combustible matter, and in some cases less than this, and the average coal contains not over 1 per cent. sulphur. Coal in the Connellsville basin is equally pure, but on account of the fact that more of the seam is mined in this field the ash is usually higher than in the coal produced in the gas-coal field, the sulphur being about the same. In recent years it has been discovered that the lower or southwestern portion of the gas-coal zone makes a standard coke, the difference between it and the Connellsville coke being that the coal in this basin on account of its hardness has to be crushed before it is put into the coke ovens, or when not crushed only the slack coal is used. This hardness of the coal in the southwestern portion of the gas-coal basin, however, is an advantage on account of the fact that the lump coal from the gas-coal basin is sold at the mine for shipment at a normal price of $1.40 per ton, which is equivalent to coke (including its cost of manufacture based on beehive oven practice) at $2.50 per ton at the ovens.

The Pittsburg coal field in western Pennsylvania west of the Monongahela river and within the State lines comprises about 850,000 acres of

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unmined coal today. In order to understand the available tonnage per acre in this field of coal and the values in various parts of the coal field it is necessary to subdivide it into zones of quality and thickness; and in order to understand the value from all standpoints it is necessary to further subdivide it into zones of thickness, zones of quality, zones of cose with reference to the cost of mining and of transportation. The thickness of the various parts of the field determines the available tons per acre which in turn becomes a factor in the valuation of the land. To measure the value of the land of the different subdivisions or zones, all of the factors, quality, tons of recoverable coal to the acre, the cost of mining and the cost of transportation to market, must be considered.

The author in considering this coal field finds it more interesting from the utilitarian point of view, for after all that is the true test of all fuel resources. Its past and present service to the industries of the country, the future uses to which it may be put, its conservation, its probable life, now susceptible of definite measurement, are the principal factors which give interest to its consideration. The value and virtue of the different subdistricts unfold from a study of them under these subjects. Another practical feature that today attracts attention to this or any other coal field is the railroad tariffs for the transportation of the product, which will from the present time be adjusted by the Interstate Commerce Commission from the standpoint of cost and service of the railroads, the welfare of the public being properly considered.

To measure the life of this coal field is an easy matter. In the Connellsville basin, the Lower Connellsville or Klondike basin, and other areas of coal east of the Monongahela river there are approximately 100,000 acres of the Pittsburg seam available and unmined at this time. In 1912, I determined the unmined acreage of coal in the Connellsville basin and also the Lower Connellsville basin and the ovens attached to the lande by copying from the assessors' books in each township of these two fields the sworn statements of the various owners of the coal lands and coke plants. From these sworn statements I made the following summaries:

*Connellsville Field. Ovens and Acreage in* 1912

| | Number of Ovens | | Acres |
|---|---|---|---|
| | Tax List | Connellsville Courier | |
| Old Basin | 22,985 | 20,361 | 32,461 |
| Klondike | 12,784 | 12,377 | 33,029 |
| Total | 35,769 | 32,738 | 65,490 |

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*Connellsville Field. Ovens going Out of Blast Due to Exhausted Coal Acreage*

| | Old Basin | Klondike | Total |
|---|---|---|---|
| In next 3 years | 5,393 | 1,310 | 6,703 |
| In next 5 years | 7,097 | 2,052 | 9,139 |
| In next 10 years | 11,940 | 3,814 | 15,744 |
| In next 15 years | 14,529 | 5,270 | 19,789 |
| In next 17 years | 17,491 | 6,130 | 23,621 |

After 17 years there will be, of all ovens shown, 5,494 ovens in the Old Basin with 5,881 acres of coal left, with an average life of over 9 years; and 6,654 ovens in the Klondike with 9,501 acres of coal left with an average life of over 13 years.

*Connellsville Field. Ovens going Out of Blast Due to Exhausted Acreage*
(Eliminating the H. C. Frick Coke Co.)

MERCHANTS OVENS

| | Old Basin | Klondike | Total |
|---|---|---|---|
| In next 3 years | 2,575 | 1,310 | 3,885 |
| In next 5 years | 3,135 | 2,052 | 5,187 |
| In next 10 years | 4,691 | 3,814 | 8,505 |
| In next 15 years | 6,455 | 4,760 | 11,215 |

After 15 years there will be of the present oven plants shown 1,039 ovens in the Old Basin, with 1,653 acres of coal left, with an average life of over 14 years; and 3,842 ovens in the Klondike with 5,781 acres of coal left, with an average life of over 13 years.

*Number and Life of Ovens of H. C. Frick Coke Co. in Connellsville Field*

| | Number of Ovens | | | Average Life, Years |
|---|---|---|---|---|
| | Tax List | Connellsville Courier | Acres | |
| Old Basin | 15,491 | 14,199 | 23,896 | 14+ |
| Klondike | 4,182 | 3,664 | 14,271 | 31+ |

In addition to the above the Frick Coke Co. has 2,935 acres in Redstone township, 1,043 acres in Menallen township and 4,423 acres in Luzerne

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township on which there are apparently no ovens. This coal together with considerable acreage in adjoining townships will probably be shipped to byproduct ovens, making the total life of the Steel Corporation's coal lands in the Klondike district approximately 20 to 25 years, assuming that they draw on these lands to make up the decline in the Old Basin.

It is clear from this accurate information that the original coking coal fields of western Pennsylvania comprising these two basins of Pittsburg coal seam, which were thought to be inexhaustible a few years ago, have only a short life. More astounding than this, however, is the short life of what is known as the Youghiogheny or Westmoreland gas coal, which comprises the coal in the gas-coal basin extending from Irwin on the main line of the Pennsylvania Railroad, east of Pittsburg, in a southwest direction, crossing the Youghiogheny and Monongahela rivers and extending into the northern end of Greene county. The coal from this basin has supplied probably 90 per cent. of the coal-gas illuminating plants in the United States up to the present time. With the disuse of oil in gas making, steel heating and melting and in many other industries, the demand for this grade of coal will probably double within the next five years. Including the Westmoreland field and extending into the northern end of Greene county there is unmined and available in this basin approximately 150,000 acres. On account of the fact that the southwestern portion of this basin makes a standard low-sulphur coke, various steel corporations have purchased large bodies of this land and there is now only available for the open market the product from about 70,000 acres of this coal. The short life of the anthracite field is usually commented upon, but it can be stated that the commercial coal in this particular field and grade of coal will be exhausted in one-seventh of the time necessary for the exhaustion of the anthracite coal, assuming that the peak of production in the anthracite field has been reached. The basin of coal lying west and northwest of this basin between the Pinhook and Washington anticlinals will be called upon to supply the gas coal from the Pittsburg district when the gas-coal basin proper has been exhausted. In this discussion, however, all lands west of the Pinhook and Murraysville anticlinals are included under the name of "Panhandle" coal.

Fig. 1 shows the number of acres unmined in each subdistrict, the number of tons of recoverable coal per acre based on the past practice of the district in mining, and the divisions of quality in the district. Fig. 2 shows the divisions in the cost of mining and the freight-rate divisions, all of which go toward making the values of the various lands of the district. Fig. 3 shows the estimated life of the various divisions of the Pittsburg coal field. The coking-coal lands are divided into those available and unmined containing 1 per cent. sulphur and under. This

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includes the Connellsville basin, the greater part of the Klondike field and the eastern portion of Washington county east of the Pinhook anticlinal and southwest of Monongahela City, and the field in the northeastern portion of Greene county. The coking coal higher than 1 per cent. in sulphur and which must be washed in order to make standard coke includes the upper portion of the Connellsville basin, the southern portion of the Klondike field and the southeastern portion of Greene county. The extreme thickness of the coal and large recovery of coal

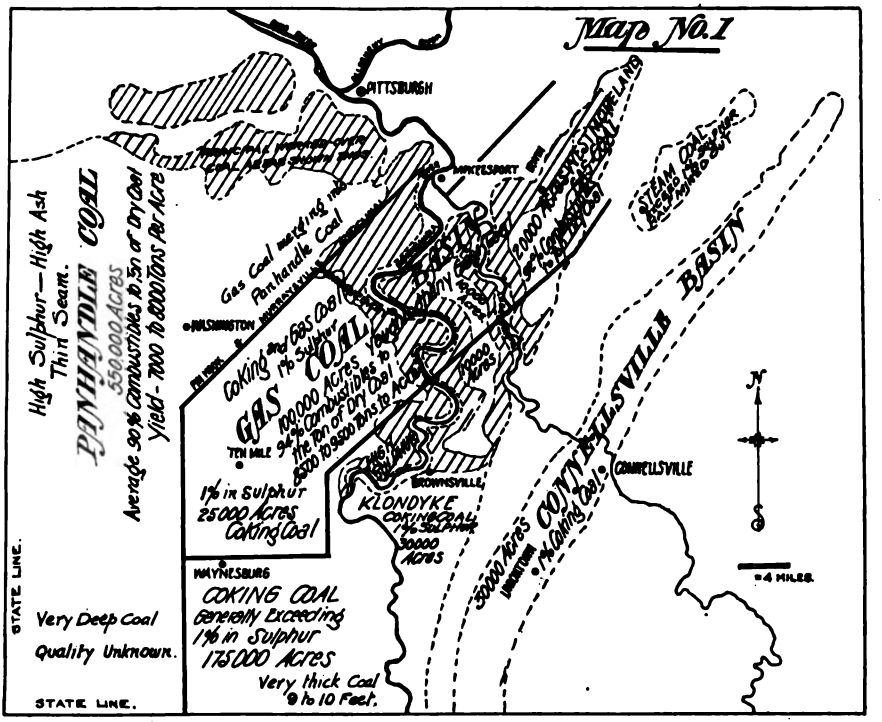


FIG. 1.—SHOWING NUMBER OF ACRES UNMINED AND NUMBER OF TONS OF RECOVERABLE COAL PER ACRE. ALSO DIVISIONS OF QUALITY.

to the acre in the southeastern portion of Greene county and the low mining cost offset the cost of washing.

In calculating the life of the various fields of coking coal, the pig-iron production tributary to this field of coal up to October, 1912, as given by the *Iron Age*, with a forecast showing the normal increase, is used as a basis. It is also assumed that the new fields will replace the declining production of the Connellsville basin and will also take on the burden of the increased consumption due to increased production of pig iron geographically tributary to these coal fields.

The average flat rate of increase of pig-iron production, based on

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figures for 1900 to 1910, was approximately 1,171,334 tons. For the next 10 years 1,400,000 tons may be assumed as the average flat rate of annual increase. The productions from 1912 to 1920 would, therefore, be as shown below. This forecast allows little for increase in population or increased use of iron per capita.

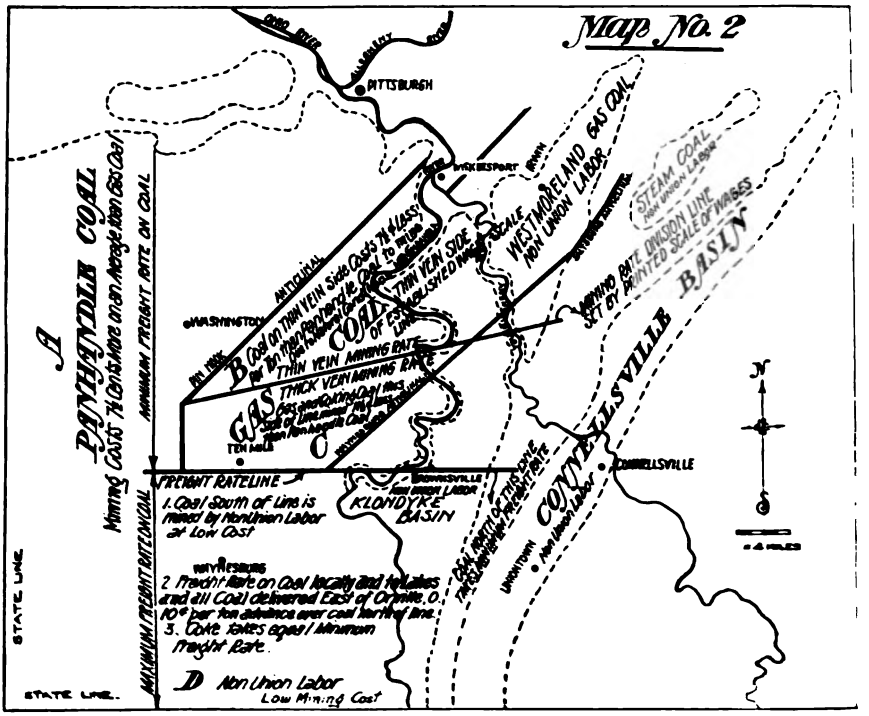
Map No. 2
PITTSBURGH
PANHANDLE COAL
Mining Costs 7¢ Cents More on an Average than Gas Coal
MINIMUM FREIGHT RATE ON COAL
STATE LINE
STATE LINE
WASHINGTON
TEN MILES
FREIGHT RATELINE
1. Coal South of Line is mined by Non Union Labor at Low Cost
2. Freight Rate on Coal locally and Points and all Coal delivered East of Orrville, O. 10¢ per ton advance over coal North of line.
3. Coke Rates equal Minimum Freight Rate.
D Non Union Labor Low Mining Cost
KLONDYKE BASIN
CONNELLSVILLE BASIN
CONNELLSVILLE
UNIONTOWN
WESTMORELAND GAS COAL
NON UNION LABOR
STEAM COAL
NON UNION LABOR
WAYNESBURG
BUCKEYE PIPE LINE
SCALE OF MILES

FIG. 2.—DIVISIONS IN COST OF MINING AND FREIGHT RATES.

| Year | Average Pig Iron Production<br>Tons | Coke Required 1.56 Tons Coke to 1 Ton Pig Iron<br>Tons |
|---|---|---|
| 1912 | 29,115,000 | 45,419,000 |
| 1913 | 30,515,000 | 47,603,000 |
| 1914 | 31,915,000 | 49,787,000 |
| 1915 | 33,315,000 | 51,971,000 |
| 1916 | 34,715,000 | 54,155,000 |
| 1917 | 36,115,000 | 56,339,000 |
| 1918 | 37,515,000 | 58,523,000 |
| 1919 | 38,915,000 | 60,707,000 |
| 1920 | 40,315,000 | 62,891,000 |

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This following statement shows: 1. That 75 to 80 per cent. of all the iron production of the United States is located tributary to the coking coal of the Pittsburg district.

2. The quantity of coke required by the different iron-producing districts. This is arrived at by taking the pig-iron production of the United States for the last five years and the coke production for a like period. The pig-iron production of the United States from 1907 to 1911, inclusive, was 117,445,261 tons, and the coke production from 1907 to 1911, inclusive, was 183,388,466 tons. Coke required (for blast furnaces, foundries, smelters, and gas production) in terms of 1 ton of pig-iron production equals 1.56+ tons of coke production.

| District | No. of Stacks | | Month of October, 1912. Pig-Iron Production, Tons | | Coke Required, Tons | |
|---|---|---|---|---|---|---|
| Buffalo, New York | 16 | | | | | |
| Rest of New York | 2 | | 179,726 | | 280,372 | |
| Pittsburg district | 51 | | 621,813 | | 970,028 | |
| Eastern Pennsylvania (small stacks) | 35 | | 227,106 | | 354,285 | |
| Western Pennsylvania | 18 | | 157,027 | | 244,962 | |
| Shenango valley (Penn.) | 16 | | 143,115 | | 223,259 | |
| Mahoning (Penn.) | 20 | | 256,711 | | 400,469 | |
| Central and Northern Ohio | 21 | | 237,506 | | 370,509 | |
| Wheeling district (Ohio and W. Va.) | 10 | | 118,037 | | 184,137 | |
| | | 189 | | 1,941,041 | | 3,028,012 |
| Chicago district (Mich., Wis. and Minn.) | 38 | | 442,091 | | 689,661 | |
| | | 38 | | 442,091 | | 689,661 |
| Southern Ohio | 9 | | 38,419 | | 59,933 | |
| New Jersey | 1 | | 5,370 | | 8,377 | |
| Southern States | 41 | | 239,686 | | 373,910 | |
| Western States | 3 | | 23,326 | | 36,390 | |
| | | 54 | | 306,801 | | 478,610 |
| | | 281 | | 2,689,933 | | 4,196,292 |

This analysis shows that the net demand on the Pittsburg district by the different iron-producing districts, is approximately 35,000,000 tons of coke per year. This figure is arrived at by taking the iron production for October, 1912, of the States of Pennsylvania, New York, and Northern Ohio, with one-half that of the Chicago and the northern Lake district. These requirements show an annual demand for 40,485,212 tons of coke, from which must be subtracted 4,136,000 tons of coke manufactured in by-product coke ovens in Pennsylvania, New York, and Ohio, and also subtracting 1,000,000 tons of hard coal used in iron smelting in Eastern Pennsylvania.

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Using the foregoing forecast as a basis and assuming that the same percentage of pig-iron production tributary to the Pittsburg coal field will be maintained out of the whole production of the country, the life of the various subdistricts of coking coal may be stated approximately as follows: Connellsville basin, 15 years; Lower Connellsville or Klondike, 20 years; eastern Washington county, 25 years, it being assumed that one-half of this coal will be used for smelting iron ore and the other half used for gas making, steel melting and other purposes; 175,000 acres in eastern Greene county, 40 years.

Merging all the acreage together the life of the coking coal in the Pittsburg district based on a normal increase in the use of this coal in the

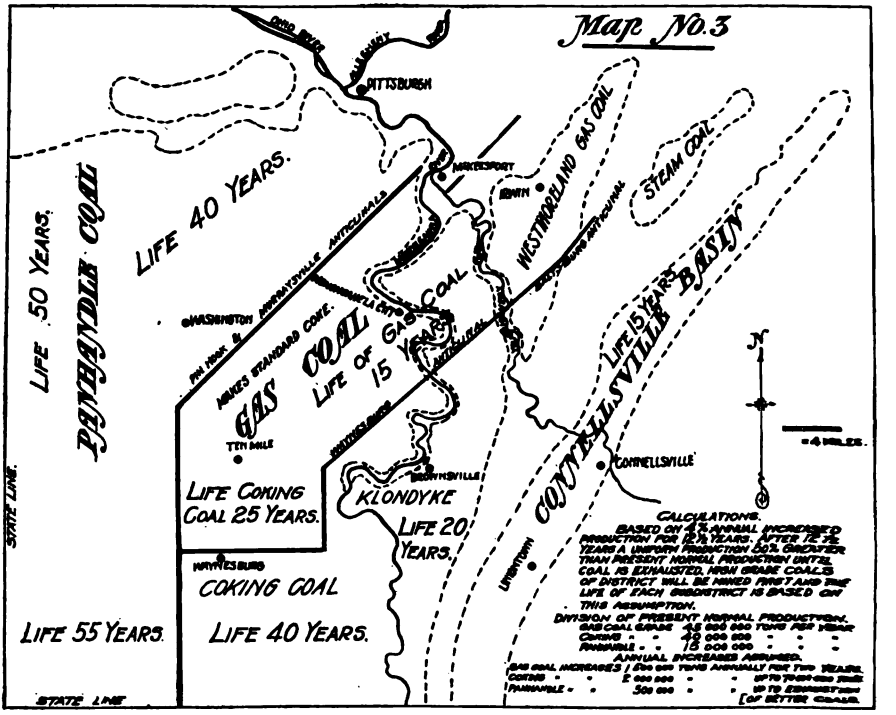


FIG. 3.—ESTIMATED LIFE OF VARIOUS DIVISIONS OF PITTSBURG COAL FIELD.

iron and steel industry and as a domestic fuel to replace hard coal will be approximately 40 years. It may be assumed that the present type of ovens will be largely replaced by byproduct oven plants on the field and at the point of manufacture.

The conditions governing the value of the Pittsburg coal field both by the ton and by the acre may be stated as follows:

The Pittsburg coal field has been so largely operated and the coals from the different subdistricts tested that it is not only possible to determine approximately the number of tons of recoverable coal per

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acre but it is possible to determine the value of the land based on the number of heat units delivered per pound of coal. The pure gas coal in the gas-coal basin as compared with the average Panhandle coal as a steam-generating fuel (the noncombustible matter including ash, sulphur and water) may be stated as follows: The purest gas coal is prepared for market with about 94 per cent. combustible matter and 6 per cent. noncombustible matter, the Panhandle coal is prepared for market with an average of 90 per cent. combustible matter and 10 per cent. noncombustible matter, the average difference being 4 per cent. The average delivered cost of a ton of Pittsburg coal as it is now delivered to consumers, including freight rates, approximates $2.50 per ton. (The lowest delivered cost in the Pittsburg district is about $1.50 per ton. Coal shipped by way of the Great Lakes and into Canada and the West is delivered to the consumer at from $3.50 to $4.50 per ton, the average, giving due weight to the tonnage, is approximately $2.50 per ton.) A difference of 4 per cent. in heat value on a basis of $2.50 delivered cost per ton shows a difference in delivered money value of 10 c. per ton in favor of gas coal, and reduced to an acreage basis for gas coal amounts to an excess value of between $800 and $900 an acre. As has been stated before, the gas coal merges gradually from the Pinhook anticlinal westward into Panhandle coal and there are points and locations immediately west of the gas-coal basin where the difference in value of heat units per pound of coal delivered would be much less than the average difference.

The value of the coal in the Pittsburg district based on quality as applied to any special industry may be reduced to a value per acre of land. It has been stated by managers of illuminating-gas plants that the Pittsburg gas-coal product has an ultimate excess value of 25 to 30 c. per ton of coal over gas coals from other districts. The coking coal of the Pittsburg seam in western Pennsylvania, based on physical and chemical properties and geographical location and freight rates, is given possession of approximately 80 per cent. of the coke demand of the United States and Canada. Judge Gary, of the United States Steel Corporation, is credited with a public statement placing $2,000 an acre as the value of coking coal from the Pittsburg seam in western Pennsylvania. This checks up with a statement made to me by one of the Pig Iron Committee of a large steel corporation that a coke with the best physical structure containing 1 per cent. or less of sulphur is worth about 30 c. a ton more in the manufacture of pig iron as compared with higher sulphur coke, as additional coke and flux must be placed in the furnace in order to remove the surplus sulphur.

The geographical value of the Pittsburg coking coal based on freight rates to distant markets has not been fully grasped either by the steel companies or the public. For byproduct-coke making or gas making along the Great Lakes, at any point, it is possible to ship low-sulphur

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Pittsburg coking coal to the lake front for a delivered freight cost placed on board vessel of 83 c. a ton, the vessel rate from Lake Erie ports to any point on the Great Lakes ranges from 25 to 35 c. a ton, making a total freight rate to Lake Superior points of $1.13 and to Lake Michigan points $1.18 per ton. The minimum railroad freight rate on coal from West Virginia and Kentucky to Chicago or Lake Michigan points is $1.90 per ton. A freight rate from southern Illinois based on about 3 mills per ton-mile would be $1.05 per ton to Gary, Ind. Coal shipped from the Pittsburg coal field via lake to Lake Michigan points therefore shuts out all West Virginia and Kentucky coals by rail and can be put into the storage yard at the byproduct coke oven plant in the Chicago district along the lake front (based on available carbon per unit of pig iron produced) at about an equal cost with southern Illinois coal. Southern Illinois coal, such as Franklin County coal, according to a paper on Illinois coals,[1] contains from 18 to 21 per cent. noncombustible matter as against 6 to 7 per cent. noncombustible matter in Pittsburg byproduct coal. I believe that recent developments in southern Illinois indicate that these analyses of southern Illinois coals are too high in noncombustible matter, but taking this coal as now produced in the railroad car, 16 per cent. may be assumed as noncombustible matter, divided approximately into 9 per cent. ash and 7 per cent. water, or 10 per cent. ash and 6 per cent. water. There would, of course, be certain losses in heat units in the coking of southern Illinois coal on account of the water that could not be charged to Pittsburg coal, and excess sulphur in Illinois coal would also reduce the amount of available carbon per unit in the blast furnace. The foregoing facts when taken in connection with the superior coking qualities of Pittsburg coking coal would indicate that southern Illinois coal for byproduct use in the Chicago district would not be as cheap ultimately as Pittsburg coal shipped there by lake, based on equal cost of coal at the mines. A careful study of the situation indicates also that the pig-iron furnaces bordering on the Great Lakes, almost all the furnaces in northern and central Ohio, western Pennsylvania, furnaces in New York State, a large percentage of the furnaces in eastern Pennsylvania, and the furnaces in Canada will draw their fuel supply from the Pittsburg coal field; as that industry increases in volume, the coal production will have to increase in the Pittsburg coking-coal field and the life of the field will be shortened accordingly.

Another factor that governs the value and life of the subdistricts in the Pittsburg coal field is involved in the cost of mining (see Fig. 2). The wage scale divides the field into two districts (known locally as the thin- and thick-vein mining-scale districts), the dividing line running from Port Royal on the Youghiogheny river west through Lock 4 on the Monongahela river, and extending still farther west through a point

[1] *Trans.* xl, 4, *et seq.* (1909).

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north of Bentleyville, Washington county. The coal north of this line has the thin-vein wage scale, which reduced on the face of the scale to a run-of-mine basis amounts to about 7 c. per ton excess cost for cutting and loading, as compared with coal south of this line. This, however, is not a true measure of the cost of mining coal in this district, as natural conditions in various parts of the field modify the cost of the coal as it is finally loaded for market.

The Pittsburgh Coal Co. made sworn statements before the Interstate Commerce Commission at Washington on the cost of mining 18,000,000 tons of coal from Apr. 1, 1910, to Aug. 31, 1911, as follows:

| | Production, Tons | Average Cost Mine Run Coal. (Does not include interest or dividends) |
|---|---|---|
| Panhandle coal | 8,174,880 | $1.0434 |
| Gas coal | 5,070,600 | 0.9686 |
| Thick-vein coal (coal south of wage-scale line). | 5,052,887 | 0.9283 |

This statement, on account of the fact that the tonnage is extremely large and covers a long period of time, and the further fact that the same accounting system was applied to the whole tonnage, and in like manner to each district, makes it an accurate measure of the cost of mining Pittsburg coal in the various subdistricts of the field, and for purposes of current operation places an accurate value on the land in the different districts. From the standpoint of cost it will be seen that the gas coal marked area *B* in Fig. 2 was produced on board railroad cars for practically 7½ c. per ton less than Panhandle coal. Reduced to an acreage basis this indicates that for current operating purposes gas coal costs $600 to $700 an acre less to mine than Panhandle coal. As a steam-generating fuel it has been noted before that it is actually worth, at a delivered cost price of $2.50 a ton, 10 c. per ton more to the consumer, which makes its excess value above that of the average Panhandle coal about $1,400 an acre, assuming that the consumer is willing to pay for the coal based on the heat units delivered. The thick-vein coal was mined for 11½ c. a ton less than the Panhandle coal, or when reduced to an acreage basis was mined for $1,000 an acre less than Panhandle coal. As a large part of ths thick-vein coal comes under the head of coking coal and gas coal, ite value, based on delivered results, is almost equal to the value placed on coking coal by Judge Gary, as compared with average Panhandle coal, assuming the average Panhandle coal value at $200 to $300 per acre.

Another light may be thrown on the value of Pittsburg coal lands by

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the sworn testimony of the Pittsburgh Coal Co. before the Interstate Commerce Commission involving the same period of time, by combining the foregoing statement with Comptroller Hornberger's Exhibit No. 5. Exhibit No. 5 gives an average market value of all grades of coal at the mine for lake shipment during 1910 of $1.0708 for run-of-mine coal. For purposes of calculation it may be assumed that all three grades of coal were sold at the same price, and based on this assumption, the 8,174,800 tons of Panhandle coal earned the Pittsburgh Coal Co. $224,000, the 10,123,487 tons of gas coal and thick-vein gas and coking coals earned this company approximately $1,338,000, provided the low-grade Panhandle coal was sold at equal prices with the higher-quality coal, otherwise all the profits must have been made out of gas coal and the adjoining thick-vein gas and coking coal.

With all these accurate data in hand it is very easy to place values on the different subdistricts of the Pittsburg coal field when measured by governing conditions within the district itself. When compared with other coal fields and districts in West Virginia and Kentucky it is found that the excess freight rate now charged on coal from the Fairmont district in West Virginia in the same seam of coal to the lake front is 12 c. per ton. As compared with Kanawha coals, southern West Virginia and Kentucky coals, the excess freight rate to the lake front paid by these coals amounts to 19 c. per ton and upward. As compared with all-rail shipments as far west as Chicago the freight rate on West Virginia and Kentucky coals lying nearest to Chicago is the same as the Pittsburg rate, but by way of the lake the Pittsburg field has an advantage of 12 to 19 c. per ton over these coals together with a higher market value based on delivered results. The tendency of freight-rate adjustment will add additional value to the Pittsburg coal field in western Pennsylvania, as is illustrated by the attempt of the C., H. & D. R. R. to haul coal to the lake front for approximately 2 mills per ton-mile as compared with 5 mills per ton-mile paid by Pittsburg coal. It is reasonable to believe that changes in freight rates from the Pittsburg district will be in reductions and the changes in freight rates from southern West Virginia and Kentucky will be necessary increases sufficient to maintain the solvency of the railroads carrying the coal.

The Interstate Commerce Commission will find it necessary to fix freight rates more nearly on the basis of cost of transportation in order to maintain the solvency of the railroads themselves and to conserve the natural resources of the country, all of which will tend to conserve the Pittsburg coal field and probably extend its life over the estimates made. With the exception of the Connellsville field it may be stated that the Pittsburg coal areas worked over up to the present time have not shown a recovery of more than 60 or 65 per cent. In the old days of mining only the breast coal was mined and only cheaply mined portions

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of this bench were hauled to the tipple. In recent years the more modern operations in the Pittsburg district have adopted systems of extraction which permit a recovery of 85 per cent. and in some cases 90 per cent. may be secured, but this percentage of recovery only represents the last few years and only a portion of the district. The destruction of the Pittsburg coal seam in past mining operations in this district was compulsory on account of the fact that the railroads of the district placed an extremely high freight rate on coal shipped, in order to divide the market with southern railroads, the result being that with non-union labor, in combination with extremely high freight rates per ton-mile from the Pittsburg district, and extremely low freight rates from West Virginia and Kentucky, it has been possible for operators in the Southern fields to enter markets, to a limited extent, which naturally belong to the Pittsburg coal field. It is obvious that it is bad public policy to permit the issue of railroad securities for transportation equipment, the effect of which is to destroy natural resources in the Appalachian coal fields, in order to secure an unnatural coal market for a distant coal field, and to increase the gross business of a railroad at no profit to the stockholders. As stated before, the present condition of the C., H. & D. R. R. illustrates the fallacy of this policy. The resources of the Pittsburg coal field and those of the United States cannot be conserved unless the cost of service on railroads is a measure of freight rate. With gradual adjustments made in freight rates to cure this evil it may be assumed that 90 to 94 per cent. of the Pittsburg coal measures may be recovered based on present mining practices. The Connellsville field proper, under the supervision of Thomas Lynch, President of the Frick Coke Co., has employed methods in mining which are the most efficient in the United States. The recovery of coal in the Connellsville field through the system established by Mr. Lynch is about 93 to 94 per cent. The operations in this field from the standpoint of percentage of recovery, safety measures provided for guarding against death and accidents to miners, and improved living conditions around the mines, are an example for all coal-mining operations of the country. If the remainder of the Pittsburg coal field, through adjustment in freight rates and mining costs, would adopt a modified system of the Connellsville plan of extraction, the life of the Pittsburg coal field would be increased materially and the wealth of the district proportionately.

The author of this paper has over a period of 10 or 15 years developed a system of mechanical mining to be applied to a modified system of extraction as practiced in the Connellsville field by the Frick Coke Co. (Figs. 4 to 5). After spending much time and considerable money in this period of time machines and methods were produced which have apparently reached a stage of perfection. Several of these machines

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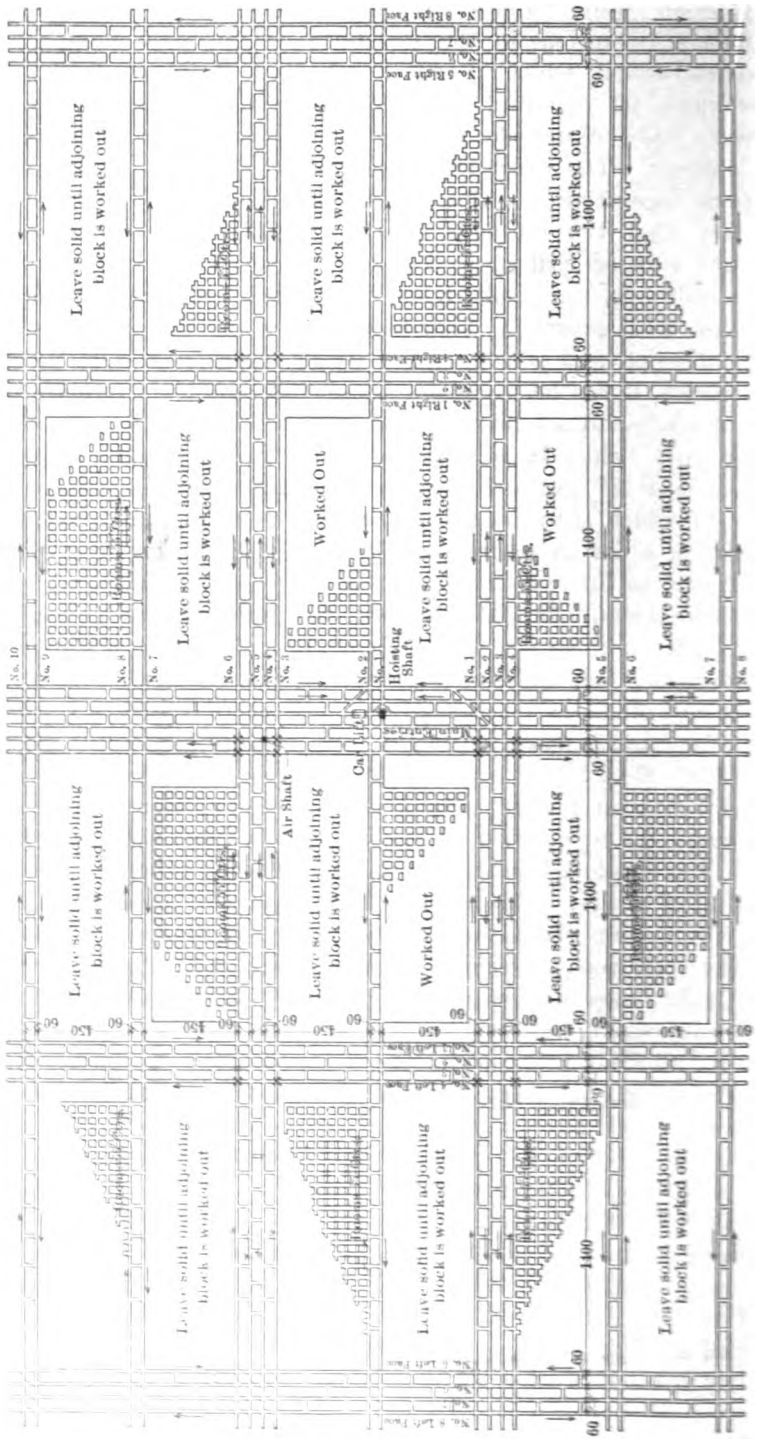


FIG. 4. PLAN OF MINING, SHOWING EXTRACTION OF COAL BY MECHANICAL MEANS.

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have operated continuously for many months in the Pittsburg coal field with small repairs and without any delays in operation. The results are so convincing, that it can be announced without reservation that mechanical mining is established and that by means of machines now perfected and operating daily, and by adapting the modified plan of extraction equivalent to the methods in the Connellsville field, the following results may be obtained:

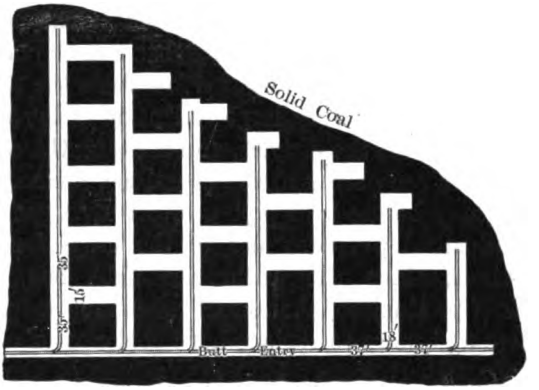


FIG. 5.—PLAN SHOWING ADVANCE.

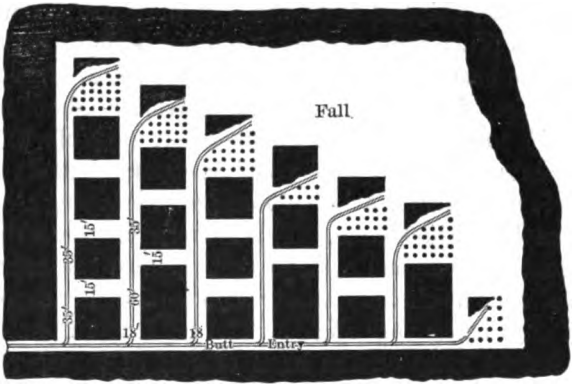


FIG. 6.—PLAN SHOWING RETREAT.

1. That 90 to 95 per cent. of the coal measure may be recovered mechanically.

2. That the cost of mining in thick seams averaging over 5 ft. will be reduced from one-third to one-half.

3. That one man operating the new machines, and using the methods now perfected and working, will be able to produce as much coal as four



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to six men produce under the present methods by the use of the present machinery.

With all this in view the future of the Pittsburg coal field is extremely bright and the wealth of the district is increased largely.

It may be stated that the present beehive ovens or open ovens of the district will be used to exhaust areas of coal lands attached, but that future coke plants supplying the iron and steel industry tributary to this coal field will, to a large extent, be the retort type of oven with byproduct recovery. Some of these plants will be placed at the point where the iron and steel is manufactured and other plants will be placed in the coal field. The Pittsburg district is facing in a few years the total exhaustion of natural gas, or at least a very large reduction in the supply. There are established throughout the district, and crossing the coal field, many large pipe lines which are now used as natural-gas mains. It is obvious that these mains may be used for piping retort-oven gas to the various industries for industrial purposes and to the cities for domestic use. It is also obvious, since the anthracite field has approximately reached its peak of production, that small coke made at byproduct ovens will be used more and more to replace hard coal west of a line north and south running through Buffalo. It is the opinion of the author of this paper that the majority of the retort-oven plants will be placed in the coal field on account of the fact that the pipe lines are already established and on account of the further fact that byproducts such as tar, ammonia, and benzol may be shipped from the Pittsburg district with as much economy as from practically any other point. Freight rates throughout the country have been established so that the cost of transporting a ton of coke to market from the Pittsburg district is about equivalent to transporting 1½ tons of coal. This makes it possible to ship coke to the various points as cheaply as the coal necessary to make a ton of coke, so that there is no transportation advantage in shipping coal to a distant point to make byproduct coke. The modern construction of railroad coke equipment has reached a stage that enables the railroads now to haul a train of coke as cheaply as a train of coal of the same weight, the difference, if any, being very slight. The tendency, therefore, will be to adjust coke freight rates on the basis of cost of transportation, which will be in favor of establishing coke plants in the coal field in the Pittsburg district.

With this adjustment in view, it would be more profitable to establish the byproduct-coke plants in the coal field, especially if the gas could be sold to the pipe lines now established. Or if the railroad lines are equipped for electric traction, largely, it may be that it would pay to put the byproduct-coke plants in the coal field and by the installation of gas engines at the coke plants generate electric power which would be transmitted and sold to the railroads at low prices. The question of where to

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establish the byproduct-coke plant will depend also largely on whether the owner of the coke plant uses his own products or whether the coke plant is established for the general market. It would seem, however, that a commercial coke plant would have a larger market both for gas and coke if established in the Pittsburg coal field than if placed elsewhere, as the established freight rates on coke would allow shipments of coke to any point now available, and the local market for gas would absorb at fair prices the surplus gas of the plant.

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# Gasoline Locomotives in Relation to the Health of Miners

BY O. P. HOOD, PITTSBURG, PA.

(Pittsburg Meeting, October, 1914)

NONE of the methods now in use for the transportation of materials underground is entirely free from more or less serious objection. The great flexibility, ease of control and economy of operation of electric tramming are accompanied by the serious menace of a trolley-wire distributing system.

As the gasoline locomotive has even greater flexibility of application and requires no similar dangerous distributing system, it might be considered a safety device which would make possible the elimination of the dangerous trolley system were it not possessed of other objectionable qualities peculiar to itself. It is found that the exhaust gases from the engine may be injurious to the health of those breathing the air in which the locomotive has been operating. While electric shock may kill the individual who makes contact with uninsulated parts, the gasoline engine may be detrimental to the health of all those who have to work within the atmosphere corrupted by exhaust gases. The degree of pollution measures the magnitude of the menace. This may be negligible at times, but with careless operation it becomes serious. The exhaust gases from an engine are composed of nitrogen, a little free oxygen, hydrocarbons, hydrogen, carbon monoxide, and carbon dioxide, the last two being considered dangerous.

The presence of carbon dioxide as a product of combustion of the gasoline was recognized as an objection from the beginning of the use of these machines, but attention was called to the fact that the amount produced was relatively small as compared to that from other sources of this gas, and it was not likely to be made in dangerous quantities. The effect of carbon dioxide, except in relatively large percentages, is confined to the effect it may have in reducing the oxygen content of the air that is breathed.

The presence of carbon monoxide in the exhaust gases in injurious quantities was less apparent, but it appears from what is now known that this is the limiting factor in the use of these locomotives if they are not to be injurious to health. The presence of carbon monoxide in the air in relatively small quantities has been shown to have a marked effect upon the blood, producing sickness and, if inhaled in sufficient quantity, pro-

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duces death. A discussion of the effects of these gases is to be found in *Technical Paper No. 62, U. S. Bureau of Mines*, by G. A. Burrell, F. M. Seibert, and I. W. Robertson.

It is difficult to get those most conversant with the effects of carbon monoxide to assign a definite limit to the amount which may be in the air without detriment to health. There does not exist a sharply defined line on one side of which is to be found safety, with danger on the other side. It has been shown that the physiological effects vary with different people, and with the state of health and degree of activity required of the individual, so that a condition which may be considered safe may prove to be unsafe under conditions of harder work and unfavorable physical state. After careful inquiry, the best that can be stated at this time is that, without injury to health, no more than 0.1 per cent. of carbon monoxide can be breathed and that for short and infrequent intervals. It is probable that one-half of this percentage could be allowed for a considerable period of time without noticeable effect.

The percentage of carbon monoxide in the mine air depends upon the amount made by the engine and on the quantity of air with which it is mixed. It will be necessary to provide ventilation for the worst possible combination of gases which such engines can make under unskillful handling, or else to become informed as to the actual amount of carbon monoxide produced and provide air accordingly. To provide for the worst conditions it becomes necessary, then, to consider the gasoline locomotive as a carbon monoxide producer, and to discover the maximum quantity of this gas which any engine is capable of making. It is not sufficient to consider the average amount produced as distributed over the whole time of running such a machine. The total quantity of gasoline burned in any one day may have produced but a small quantity of carbon monoxide, but if this has been confined to a relatively short period during bad carburetor adjustment, and in some poorly ventilated space, the momentary percentage may be very high and the consequences may be fatal. It is evident that to be entirely safe the ventilation must be sufficient to keep the percentage of carbon monoxide below the assigned limit when the engine is producing the maximum quantity possible. If this maximum quantity is provided for by proper ventilation, the chance of injury to health can be considered to be quite remote. Certain peculiarities of gasoline engines make the percentage of carbon monoxide generated vary between rather wide limits, but the maximum is fairly constant. No other constituent of the exhaust gases varies so much or so rapidly with slight changes of adjustment as does the carbon monoxide.

Extended experiments have been made by the Bureau of Mines to determine under what conditions of running such locomotives give a maximum quantity of carbon monoxide. This maximum quantity is found when the engine is running at full load and full speed, and when

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burning the greatest quantity of gasoline which will maintain these conditions. Any further enriching of the mixture results in weaker explosions, misfires, a reduction in speed of the locomotive, and thus a reduced total quantity of gases. The weight of gasoline used per stroke may vary nearly 100 per cent. without varying the speed or power. As the mixture is enriched the combustion is less complete, reducing the efficiency but affecting the power but little until the upper limit of explosibility is approached, when the power falls rapidly. Under these conditions a maximum quantity of carbon monoxide equivalent to 5¾ per cent. of the piston displacement may be generated and delivered into the surrounding air. When the engine is throttled down to a slow speed and with a light load it is possible to produce exhaust gases containing a greater percentage of carbon monoxide but the total quantity is less because of the slow speed of the engine. It can therefore be stated that the amount of carbon monoxide possible to generate with a gasoline locomotive is a function of its piston displacement.

TABLE I.

| Engine Cylinder Size, In. | No. Cylinders | Speed Rev. per Min. | Piston Displacement *a* (Cu. Ft. per Min.) | Maximum Probable Amount of Noxious Gases (Cu. Ft. per Min. at 60° F. and 30 in. Barometer) Produced with | | | | Amount of Air (Cu. Ft. per Min.) Required to Dilute Exhaust Gases to 1 Part CO per 1000 Parts of Air *b* | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | Good Carburation | | Bad Carburation | | Good Carburation | Bad Carburation |
| | | | | CO | $CO_2$ | CO | $CO_2$ | | |
| 4.75 by 5.25 | 4 | 800 | 172 | 2.61 | 6.80 | 9.91 | 3.65 | 2,610 | 9,910 |
| 5 by 5 | 4 | 600 | 136 | 2.06 | 5.37 | 7.84 | 2.88 | 2,060 | 7,840 |
| 5 by 5 | 4 | 800 | 182 | 2.76 | 7.18 | 10.48 | 3.86 | 2,760 | 10,480 |
| 5 by 6 | 4 | 800 | 218 | 3.30 | 8.60 | 12.56 | 4.62 | 3,300 | 12,560 |
| 5.5 by 5 | 4 | 600 | 165 | 2.50 | 6.51 | 9.50 | 3.50 | 2,500 | 9,500 |
| 6 by 6 | 4 | 700 | 275 | 4.17 | 10.86 | 15.85 | 5.82 | 4,170 | 15,850 |
| 6 by 7 | 4 | 500 | 229 | 3.47 | 9.04 | 13.19 | 4.85 | 3,470 | 13,190 |
| 6.5 by 7 | 4 | 500 | 269 | 4.07 | 10.63 | 15.50 | 5.70 | 4,070 | 15,500 |
| 6.5 by 8 | 4 | 650 | 399 | 6.04 | 15.76 | 23.00 | 8.46 | 6,040 | 23,000 |
| 7 by 7 | 4 | 500 | 312 | 4.73 | 12.33 | 17.97 | 6.62 | 4,730 | 17,970 |
| 7 by 7 | 6 | 500 | 468 | 7.08 | 18.49 | 26.97 | 9.92 | 7,080 | 26,970 |
| 8 by 7 | 4 | 500 | 407 | 6.16 | 16.08 | 23.45 | 8.62 | 6,160 | 23,450 |
| 8 by 7 | 6 | 500 | 610 | 9.24 | 24.10 | 35.14 | 12.93 | 9,240 | 35,140 |

*a* Area piston in square feet × stroke in feet × number of cylinders × revolutions per minute.

*b* Maximum amount of carbon monoxide which can be breathed for short and infrequent intervals without injurious effects.

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The sizes of locomotives in common use are given in the accompanying table, together with the maximum quantity of injurious gases which they may generate.

The size of a locomotive that may with safety be introduced into a mine depends upon the amount of air that can be mixed with the exhaust gases in the most unfavorable portion of the run of the locomotive. For each cubic foot of carbon monoxide possible to generate in the engine there should be available 2,000 cu. ft. of air to mix with the exhaust gases if this air is for continued breathing, while for short and infrequent intervals the proportion may rise to one part in one thousand.

The air with which the exhaust gases are mixed is the air passed through by the locomotive. If the speed of the air current and that of the locomotive are the same there would be a concentration of exhaust gases immediately surrounding the engine, which would very soon reach the danger limit. The engine runner would be exposed to this excessive amount. Or, the air may be stationary, and if the locomotive passes though this air at such a rate as to distribute the exhaust gases into the required quantity of air the dilution may be below the danger limit. It becomes necessary, therefore, to investigate the relative velocity of the locomotive and the air current, together with the cross section of this air current, in order to determine whether a sufficient quantity of air is passed to insure a proper dilution. The mixing due to the movement of the cars and the diffusion of the gases is expected to distribute the gases through the air. An investigation of this kind will frequently disclose some portion of the run of the locomotive where the air and the locomotive are traveling together and the dilution is insufficient, forming the basis for just complaint. By a change in the speed of the locomotive, or more careful running, perhaps with lighter loads, these difficulties can frequently be overcome. It is evident from the foregoing that it is a dangerous practice to allow locomotives to stand idle for any length of time in locations where there is not ample movement of air. The introduction of a self-starter on locomotives which are compelled frequently to wait for cars would be a great aid. The engines on these locomotives are usually difficult to start, and to stop the engine whenever the locomotive is compelled to wait for a few minutes would not be practicable unless some easy means of re-starting was provided, but this practice would greatly reduce the chance of air pollution.

The requirement that not less than 1,000 cu. ft. of air shall pass the locomotive for each cubic foot of carbon monoxide possible for the engine to generate will limit the size of the locomotive in many cases to sizes smaller than are now in use. It may also confine the use of these engines to main entries where there is ample ventilating current.

This maximum quantity of noxious gases which is here provided for should not actually be realized with improved carburetor construction

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and skillful manipulation. The percentage of carbon monoxide in the exhaust gases under the worst conditions is about 13½ per cent., while under normal running conditions the amount seldom exceeds 6 per cent. It is this great variation in the possible amount of carbon monoxide due to unskillful operation that makes the experience of different users of these machines vary through the wide limits noted. If a locomotive is installed in a place where the quantity of air is not sufficient properly to dilute the maximum quantity of carbon monoxide, analyses of the exhaust gases should be made to determine just what percentage is actually produced with the carburetor in use and the methods employed in running these machines. The ventilating current may then be regulated to conform to the actual amount of carbon monoxide as found. It is proposed that this be done by a method somewhat as follows:

Suppose, upon analysis, that the actual amount of carbon monoxide is found to be not over 4 per cent.; allowing a factor of safety of 2, and assuming that this percentage might on occasion mount to 8 per cent., and, since the maximum percentage possible is about 13½ per cent., the ventilating current required can be regulated in the proportion of 8 to 13½.

If a careful study is made of the amount of air available for dilution throughout the run of a locomotive, and the quantities are kept within the limits that are here proposed, a wide field of usefulness for these locomotives remains, where it is believed there would be no injurious effects upon health.

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# A Test of Centrifugal Motor-Driven Pumps

BY S. S. RUMSEY* AND W. F. SCHWEDES,† DULUTH, MINN.

## *Introduction*

IN order to realize the economies which would result in operating the mine pumps at the Chapin mine with electricity generated by water power, the Oliver Iron Mining Co. recently installed a hydro-electric plant at the upper Quinnesec falls on the Menominee river, and constructed a transmission line to the mine.

On account of the large flow of water in the Chapin mine, and the great depth from which it has to be handled, it was necessary to make a thorough investigation of the merits of both the motor-driven centrifugal and motor-driven plunger pumps, in order that the most economical results as regards power, cost of installation, maintenance, and operation, be obtained. This investigation resulted in the selection of motor-driven centrifugal pumps, as the cost of the pumps and motors, installation, and maintenance was considerably lower, and the space required in the underground workings very much smaller, than for the plunger pumps. The overall efficiencies guaranteed by several of the centrifugal-pump manufacturers were so nearly those guaranteed by plunger-pump manufacturers that the saving in power which would result from the installation of plunger pumps did not justify their increased first cost and the increased cost of installation under the conditions prevailing at the Chapin mine.

In order to substantiate the representations of the centrifugal-pump builders it was deemed necessary to insist upon a factory test of the completed units before shipment, the test to be conducted under conditions of head and capacity approximating as nearly as possible those under which the pumps would operate in service. Furthermore, it was considered necessary to demonstrate, by test, that there were no deficiencies in design or capacity of the pumps or motors before their final installation in the mine, as it would be extremely difficult to make alterations in the pumps or motors after they were installed.

* Chief Engineer, Oliver Iron Mining Co.
† Non-member. Electrical Engineer, Oliver Iron Mining Co.

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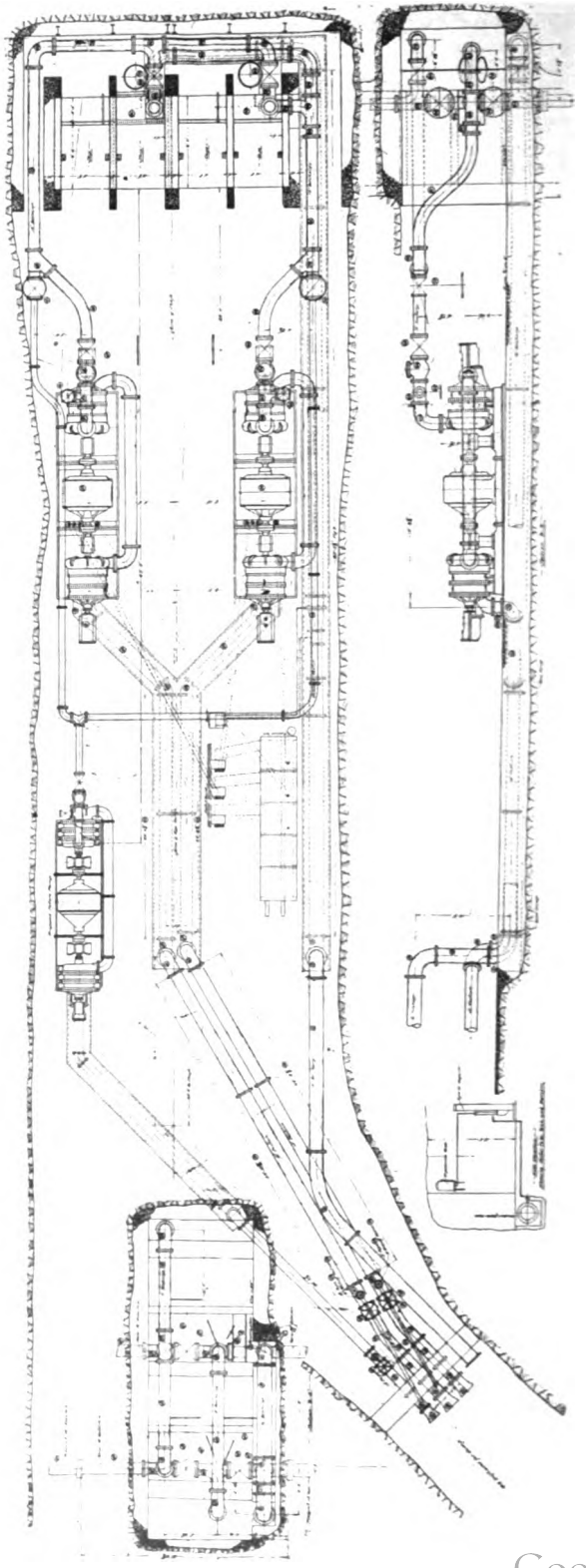

FIG 1.—PLAN AND SECTIONS OF THE 12TH LEVEL PUMP STATION, NO. 2 HAMILTON SHAFT, CHAPIN MINE.

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The general arrangement of the pumps, piping, and electrical control for the 12th level is shown in Fig. 1, and for the 16th level in Fig. 2.

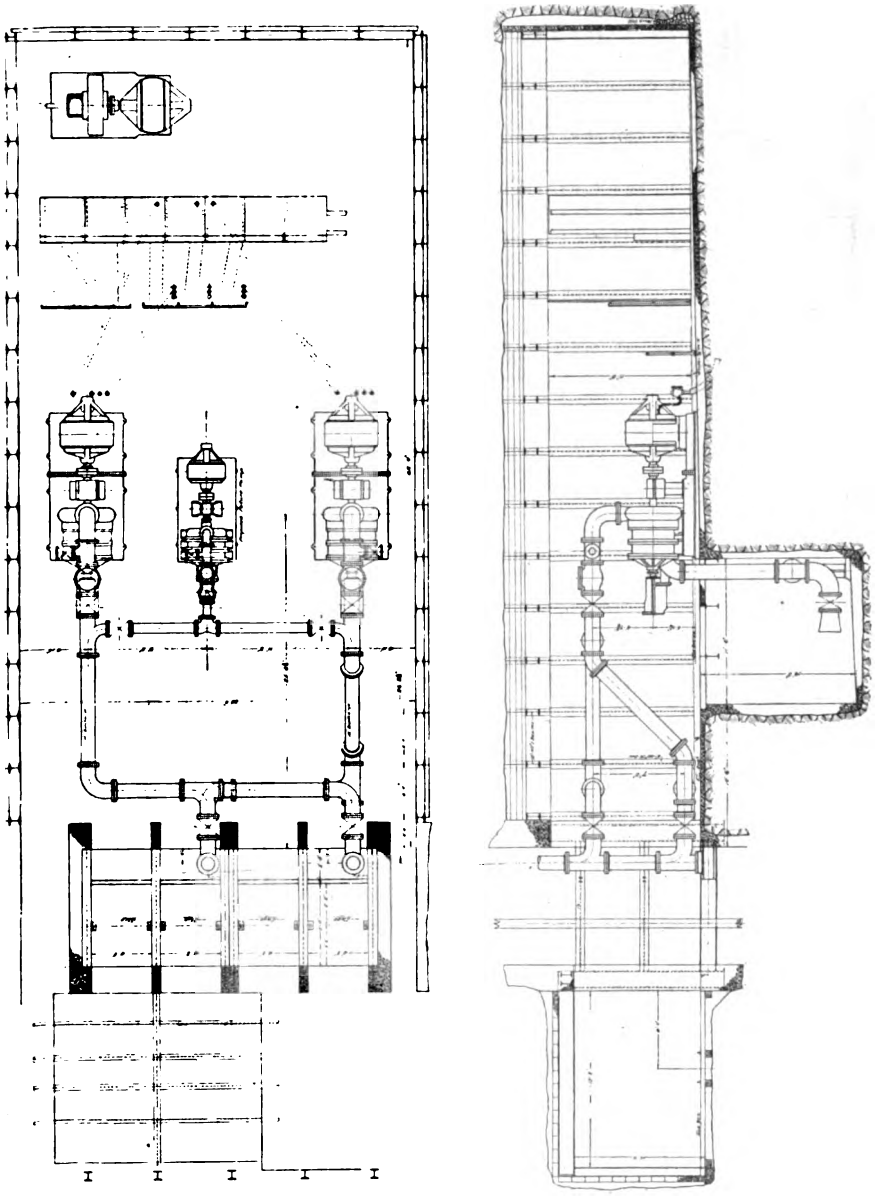

FIG. 2.—PLAN AND SECTION OF THE 16TH LEVEL PUMP STATION, NO. 2 HAMILTON SHAFT, CHAPIN MINE.

Power is transmitted from the surface to 12th-level pumps through two 250,000-circular mil, three-conductor, 30 per cent. Para rubber-insulated, lead-covered, steel-armored cables; and from the surface to 16th

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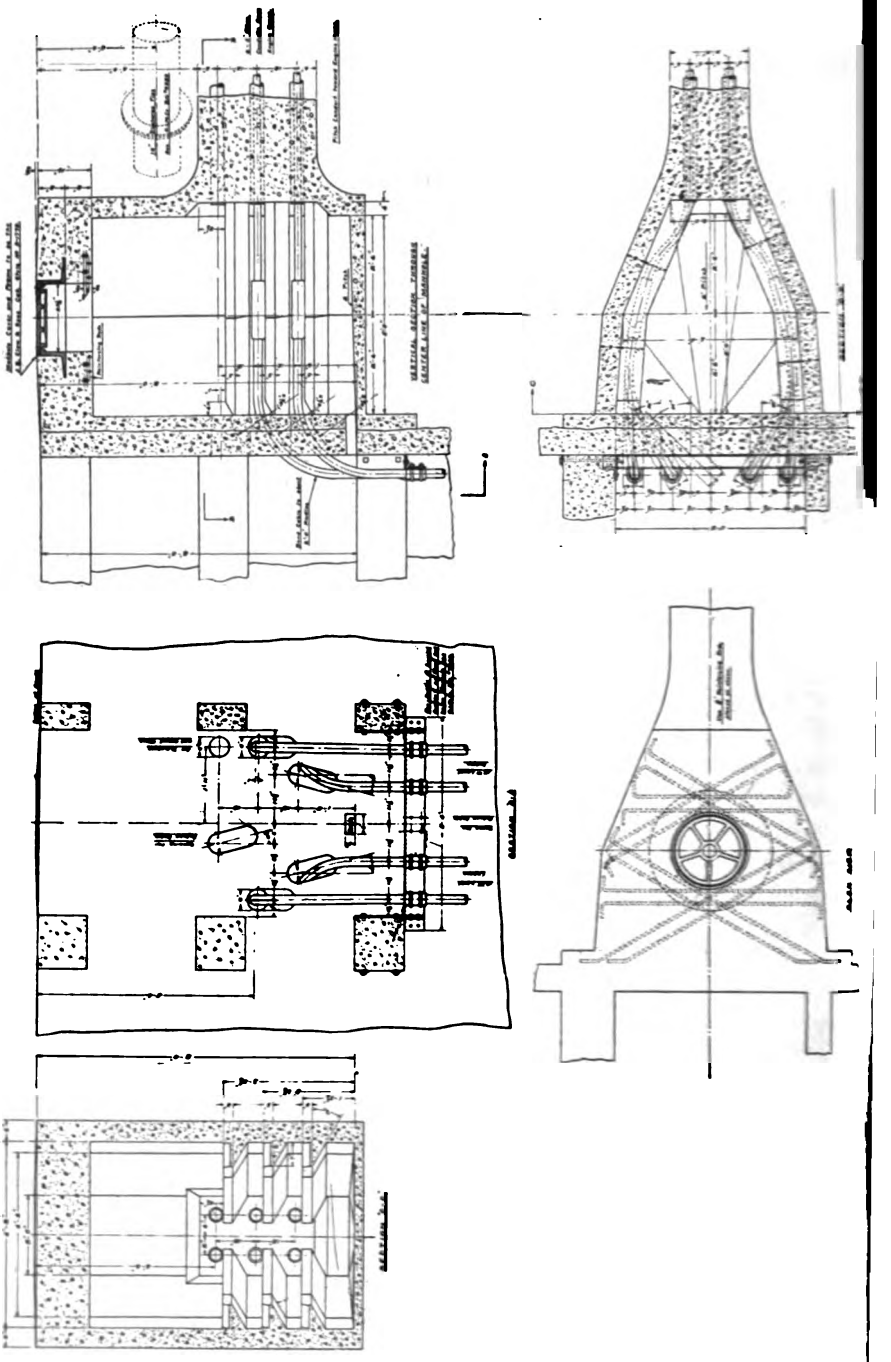
FIG. 3.—DETAILS OF CABLE MANHOLE FOR 2,300-VOLT CABLES.



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level through two 3-0 three-conductor, 30 per cent. Para rubber-insulated, lead-covered, steel-armored cables. These cables were manufactured by the American Steel & Wire Co., in conformance with specifications which were especially suited to the conditions.

The method of carrying these cables down the vertical mine shaft, and supporting them by the concrete shaft timbers, is shown in Fig. 3.

Specifications for all equipment required for the power and pumping plants, and all plans for the installation of same, were prepared in the office of the Chief Engineer of the Oliver Iron Mining Co., at Duluth, Minn.

### *Report of Test*

In accordance with the contract with Henry R. Worthington Co., tests were run on the four 3,000-gal., motor-driven centrifugal pumps built by them for service on the 16th and 12th levels, No. 2 Hamilton shaft, Chapin Mine, Iron Mountain, Mich. The complete erection and testing of these four motor-pump units was witnessed by the Electrical Engineer of the Oliver Iron Mining Co. The first pump was turned over Dec. 13, 1913, and the test completed on the fourth pump Jan. 13, 1914.

### *Description of Motor-Driven Pumps*

The water to be pumped from the Chapin mine is encountered on the 16th level, and is approximately 3,000 gal. per minute. The vertical lift from the sump on the 16th level to surface will be 1,358 ft., and the water will be pumped to the surface in two lifts, the pumps being located as described below:

Two 3-stage pumps will be installed on the 16th level, each having a capacity sufficient to handle the entire mine flow to a sump on the 12th level, a vertical lift of 392 ft. The estimated dynamic head against which each 16th-level pump will operate when throwing 3,000 gal. of water per minute is 412 ft.

Two 6-stage pumps will be installed on the 12th level, each having a capacity sufficient to handle the entire mine flow to surface, a vertical lift of 966 ft. The estimated dynamic head against which each 12th-level pump will operate when throwing 3,000 gal. of water per minute is 1,000 ft.

Under normal conditions only one pump on the 16th level and one pump on the 12th level will operate simultaneously. There are, however, two independent discharge column pipes, making it possible to operate both pumps on both levels simultaneously, thereby providing a maximum capacity of 6,000 gal. per minute from the 16th level to the surface.

Fig. 4 shows the general arrangement of present steam and air pumps and new electrical pumps.

The pumps are of the multistage turbine type, built in units of three

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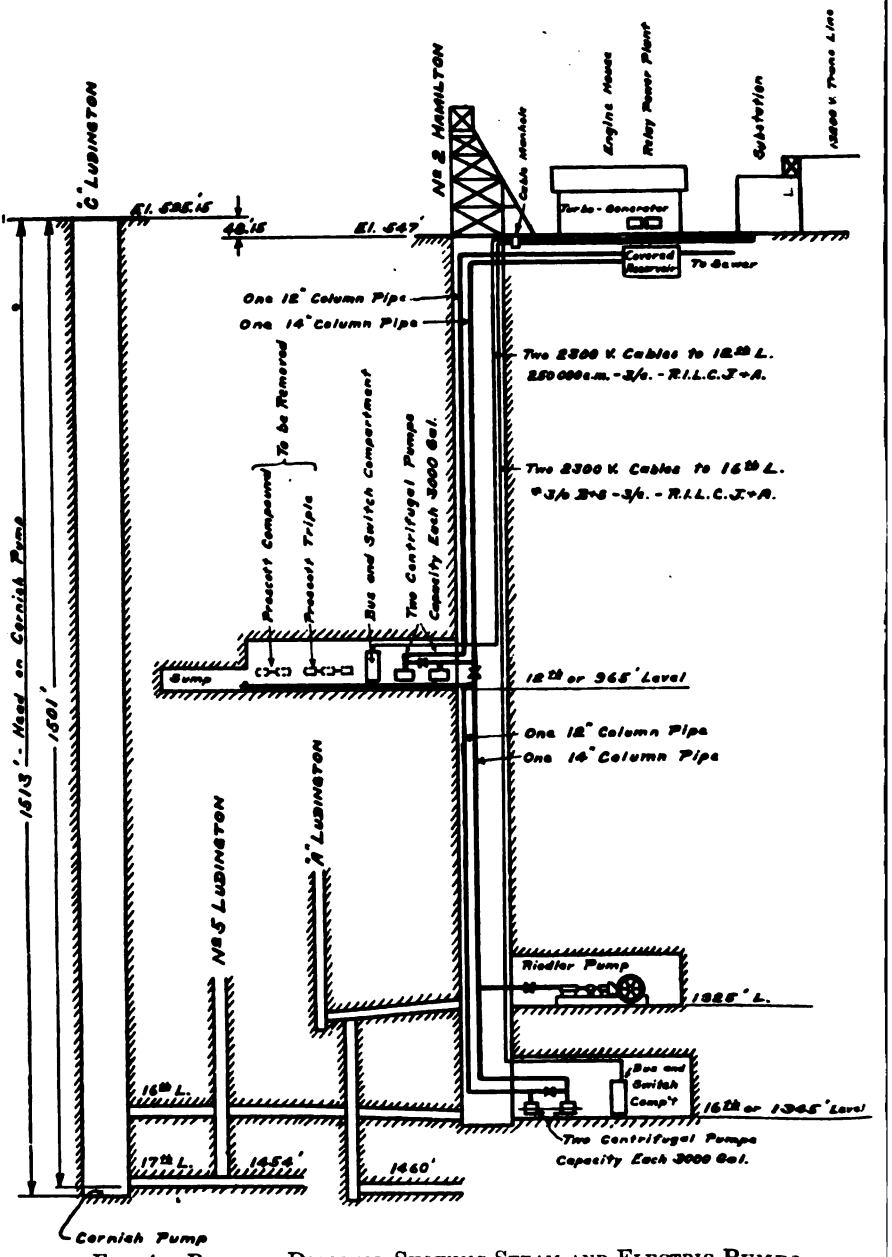


FIG. 4.—PUMPING DIAGRAM, SHOWING STEAM AND ELECTRIC PUMPS.

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stages each. The two 16th-level pumps each consist of one unit of three stages, mounted on a common cast-iron bedplate, with a 450-h.p. 1,200 rev. per minute induction motor direct connected through a flexible coupling. The 12th-level pumps each consist of two three-stage units, mounted on a common cast-iron bedplate, and direct connected through a flexible coupling to each end of a 1,050-h.p. 1,200 rev. per minute induction motor, the discharge of the low-pressure unit being connected through a 12-in. pipe to the suction end of the high-pressure unit.

The three-stage units that make up the four pumps are identical in design and dimensions so that all parts are interchangeable, excepting the impellers in the six-stage pumps, which have a trifle larger diameter as their operating head is more than twice that of the three-stage pumps.

The pumps are designed to allow lowering through a vertical mine shaft 3 ft. 10 in. wide by 6 ft. 4 in. long. All glands are water sealed and all bearings are water cooled; casings are cast steel; impellers and diffusion vanes are bronze.

Each pump is driven by a General Electric 2,200-volt, three-phase, 60-cycle, 1,200 rev. per minute, wound secondary induction motor, with provision for short-circuiting secondary windings at the control board by means of an oil switch which is so interlocked with the drum controller and primary circuit breaker that the entire control equipment is as "fool-proof" as is possible to make it.

All the 2,200-volt control equipment will be mounted in concrete compartments and all wiring around motors will be so protected that all possible danger to operator is eliminated.

These centrifugal pumps will replace the present steam pumping equipment. The rating of the centrifugal pumps is shown in Table I.

TABLE I

| | 12 in. 3-Stage 16th Level | 12 in. 6-Stage 12th Level |
|---|---|---|
| Capacity in U. S. gallons per minute | 3,000 | 3,000 |
| Total dynamic head in feet | 412 | 1,000 |
| Rev. per minute at full load | 1,178 | 1,170 |
| Diameter of suction in inches | 12 | 12 |
| Diameter of discharge in inches | 12 | 12 |
| Horsepower of motors | 450 | 1,050 |
| Electrical characteristics of motors: 2,200 volts, three-phase, 60 cycles. | | |

*Purpose of Test*

Each pump with its motor was tested to determine: (1) Head-capacity curves of pumps; (2) overall efficiency, wire to water; (3) degree



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with which builder's guarantees were met; (4) mechanical operation of pumps and motors for vibration, thrust, heating, and noise.

### *Method of Testing*

The four pumps were shipped from the Worthington factory at Harrison, N. J., to the factory of the General Electric Co. at Schenectady, N. Y., and each complete unit assembled and aligned on a steel floor plate.

Power supply for pumps Nos. 1, 3, and 4 was taken from a 12,500-kva., 2,200-volt turbo-generator which was run especially for this test.

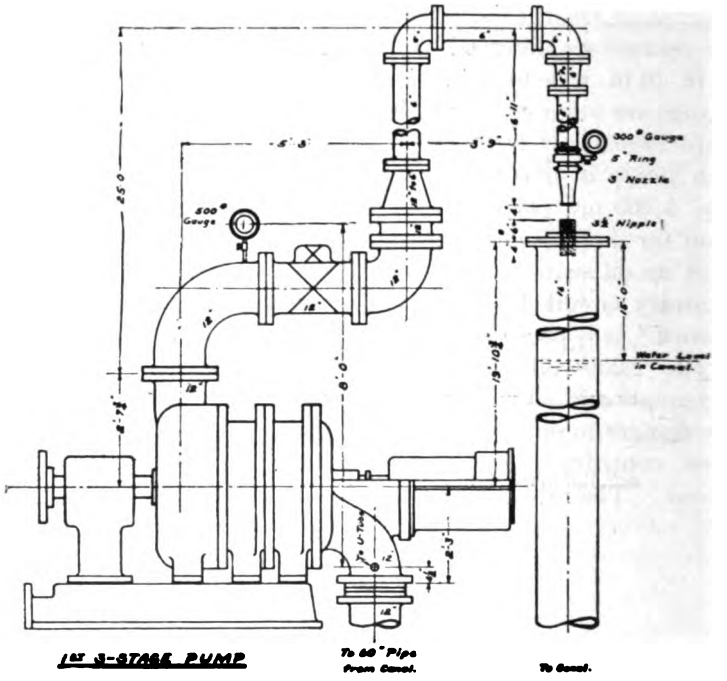


FIG. 5A.—ARRANGEMENT OF DISCHARGE WATER PIPING OF FIRST THREE-STAGE PUMP.

Pump No. 2 was supplied from a 750-kw. direct-current motor-driven alternating-current generator. Constant frequency was available from the turbo-generator, but the frequency of the motor-generator set fluctuated and test readings were taken as frequency became reasonably steady at 60 cycles. Power input to motors was measured with previously calibrated electric instruments. The head on pump was obtained by means of a pressure gauge between pump and discharge valve and a U-tube in pump suction. The pump discharge in gallons per minute was measured with a calibrated 3-in. nozzle. Fig. 5 shows the piping around

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pumps, as set up for each test, and the location of pressure gauges and U-tube. All pressure gauges were carefully calibrated and all pressure readings in the following data are corrected.

The 3-in. nozzle was calibrated over a rectangular weir up to 46½ lb. nozzle pressure, and Table II gives readings taken on this calibration.

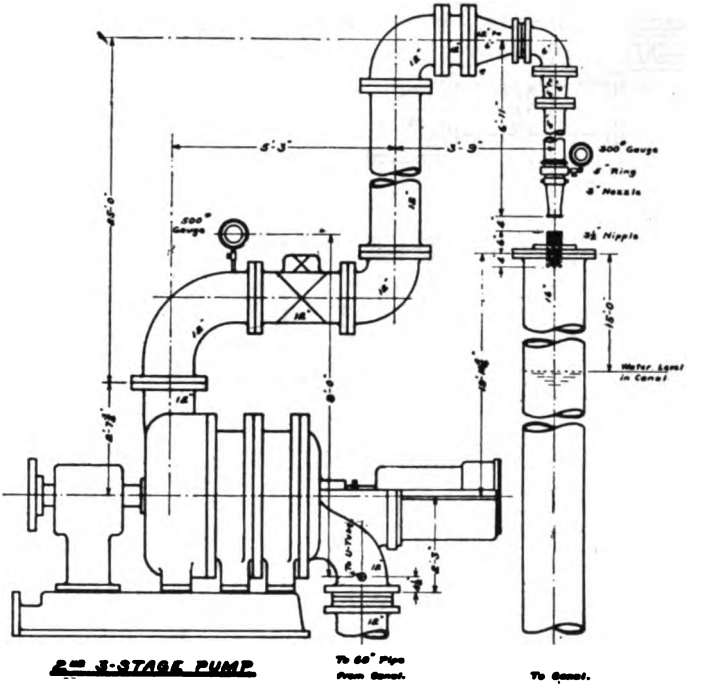


FIG. 5B.—ARRANGEMENT OF DISCHARGE WATER PIPING OF SECOND THREE-STAGE PUMP.

TABLE II

| Pressure on Nozzle *P* | Gallons per Minute over Weir *C* | Constant *K* |
|---|---|---|
| 7⅛ | 820 | 91,200 |
| 13⅛ | 1,105 | 91,500 |
| 19½ | 1,330 | 90,750 |
| 26 | 1,520 | 89,000 |
| 29½ | 1,610 | 89,000 |
| 41½ | 1,900 | 87,100 |
| 46½ | 2,040 | 89,400 |

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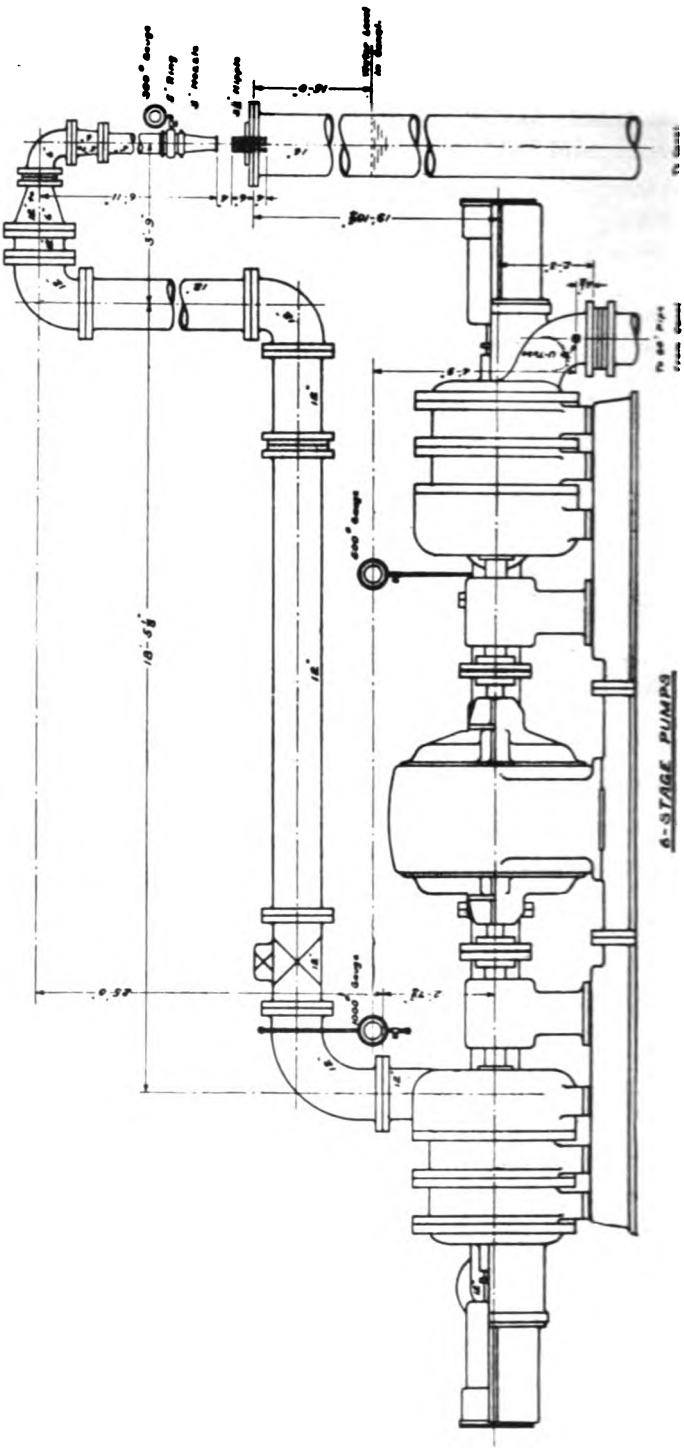


FIG. 5C.—ARRANGEMENT OF DISCHARGE WATER PIPING OF SIX-STAGE PUMPS.

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From the readings in Table II, the average value of constant $K$ = 90,000. Substituting in equation $C^2 = KP$, the pressure-capacity curve shown in Fig. 6 was prepared. Although there is a slight variation in the value of constant $K$ for different nozzle pressures, as shown in Table II, this does not appreciably affect values for gallons per minute discharge, as $C$ varies as the square root of $K$.

Water was pumped from a 60-in. pipe-line connection from the Erie Canal and the 3-in. nozzle discharged into a 16-in. stand pipe back to canal. As the level of water in the canal was higher than the suction

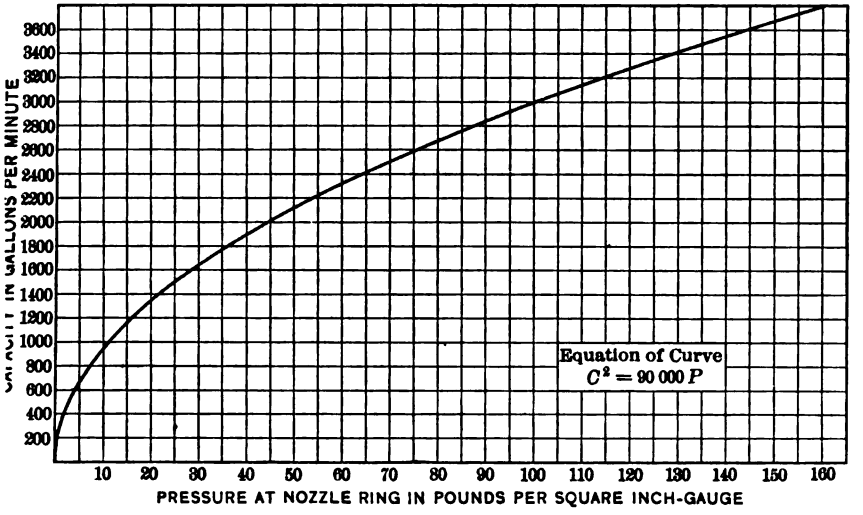


FIG. 6.—PRESSURE-DISCHARGE CURVE FOR 3-IN. NOZZLE AND 5-IN. RING.

connection on pump, the pressure on suction varied from 2 to 6 in. of mercury and was measured with the mercury U-tube. (See Fig. 5 for location of tube connection in pump suction.)

In taking the test readings on each pump, no preliminary test runs were made or readings taken, and each pump was operated before the test for only a sufficient time to inspect piping and assure proper lubrication, tight glands, and true alignment.

## *Test Data*

In the following tables, pumps are numbered as follows: No. 1, 16th level, 12 in., 3-stage, 450 h.p.; No. 2, 16th level, 12 in., 3-stage, 450 h.p.; No. 3, 12th level, 12 in., 3-stage, 1,050 h.p.; No. 4, 12th level, 12 in., 6-stage, 1,050 h.p.

Table III gives the starting current for each pump. These were taken with the 12-in. discharge valve closed and the 2-in. bypass valve open, which will approximate conditions of starting when installed.

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The motors were started with their respective resistance grids and controllers. The starting duty is so light that no trouble should be expected with the drum controllers.

It will be possible to start these pumps within 25 sec. without exceeding full-load current, by using a greater rate of acceleration on the first number of notches on the controller and a decreasing rate of acceleration on the last few notches, as the pump takes its load.

Motor efficiencies and power factors for one 1,050-h.p. and one 450-h.p. motor were obtained by separation of losses.

In Table IV are shown the guaranteed and obtained efficiencies and power factors for the two motors.

Tables V, VI, VII, and VIII for pumps 1, 2, 3, and 4, respectively, give obtained data from which curve sheets (Figs. 7, 8, 9, and 10) were prepared, giving head-capacity curves, horse-power input curves, and overall efficiency curves. The pump-efficiency curves were obtained by dividing overall efficiency by motor efficiency.

### *Test Results*

Table IX was prepared from the curve sheets to show the degree with which the builder's guarantees were met.

### *Motor Slip*

On pumps Nos. 3 and 4, the motor slip was measured directly by means of a 5-volt voltmeter connected across the slip rings, which gave actual values. The percentages of slip given in Tables VII and VIII are calculated, assuming 60-cycle power supply. Inconsistency in slip values is due to slight variations in speed of the prime mover, which it was impossible to check within the limits of error, in the values of slip.

Slip for pumps Nos. 1 and 2 was obtained by means of speed counter and tachometer and the figures are not accurate, but indicate maximum values that can be expected.

When in actual service, the slip for all motors will be less than the values given in data sheets, because of the long secondary leads used on test.

### *Temperatures, Motors and Bearings*

Because of the large amount of power and time required to obtain final full-load temperatures on motors and bearings, no attempt was made to obtain these readings further than those taken during test, which in each case covered a period of about 2 hr.

The highest temperatures recorded at the end of test runs were the end turns of stator windings given by thermometers Nos. 3 and

Table X shows maximum temperature rises above 25° C. at e

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TABLE III—*Starting Currents*

| Controller Notches | Pump No. 1 Amperes | Pump No. 2 Amperes | Pump No. 3 Amperes | Pump No. 4 Amperes |
|---|---|---|---|---|
| S | 43 | 79 | 150 | 157 |
| 1 | 43 | 49 | 160 | 143 |
| 2 | 47 | 49 | 150 | 126 |
| 3 | 51 | 51 | 110 | 110 |
| 4 | 61 | 57 | 110 | 118 |
| 5 | 65 | 53 | 110 | 118 |
| 6 | 67 | 61 | 140 | 118 |
| 7 | 79 | 69 | 140 | 118 |
| 8 | 69 | 79 | 140 | 118 |
| 9 | 77 | 79 | 140 | 118 |
| 10 | 79 | 83 | 140 | 126 |
| 11 | 75 | 84 | 120 | 126 |
| 12 | 89 | 96 | 160 | 157 |
| 13 | 98 | 118 | 190 | 197 |
| 14 | 89 | 79 | 150 | 118 |
| 'otal time, seconds | 30 | 25 | 25 | 30 |
| Full-load motor a nperes. | 101 | 101 | 229 | 229 |

TABLE IV

| | Loads | | | |
|---|---|---|---|---|
| | ½ | ¾ | Full | 1¼ |
| uaranteed efficiency, 450-h.p. motor..... | 86.5 | 91. | 92. | 92.5 |
| btained efficiency, 450-h.p. motor..... | 92.4 | 93.9 | 94.65 | 94.9 |
| uaranteed efficiency, 1,050-h.p. motor..... | 90.5 | 92.5 | 94. | 94. |
| ɔtained efficiency, 1,050-h.p. motor..... | 93. | 94.4 | 95. | 95.2 |
| ıaranteed power factor, 450-h.p. motor..... | 82. | 88.5 | 92. | 93. |
| ɔtained power factor, 450-h.p. motor..... | 86.1 | 90.3 | 91.3 | 91.5 |
| ıaranteed power factor, 1,050-h.p. motor... | 90. | 92. | 93.5 | 93.5 |
| ɔtained power factor, 1,050-h.p. motor..... | 90.8 | 93.2 | 93.3 | 93.3 |

test runs and gives an idea of what can be expected of these motors ıder continuous operation.

Being high-speed motors, a relatively small amount of material entered to their construction, giving them a small thermal capacity, so that the ove temperature rises are nearly final. The guaranteed full-load nperature rises for 1,050-h.p. and 450-h.p. motors is 35° C.

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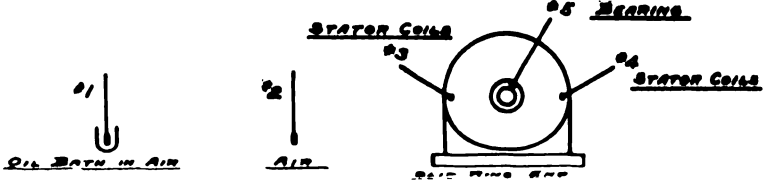
TABLE V.—*No. 1 Pump Unit for 16th Level Station*

| Time | Motor | | | | | | | Pump | | | | | | | Efficiencies | | | | | *Temperatures | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | Head-feet | | | | | Capacity | | | | | | | | | | | |
| | Volts | Amperes | Kv.A. | Kw. | Power Factor | R.P.M. | Per Cent. Slip | Mercury Column Suction. | Ft. S | Gauge Pressure Discharge | Ft. D | D−S+8 ft. Total | Gauge Pressure Nozzle | Gal. per Min. | Water Hp. | Motor Hp. Input | Wire to Water Efficiency. | Motor Efficiency | Pump Efficiency | 1 | 2 | 3 | 4 | 5 |
| A.M. | | | | | | | | | | | | | | | | | | | | | | | | |
| 10.55 | 2,180 | 64.8 | 244 | 224 | 91.8 | ..... | .... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 24.7 | 24.0 | 26.0 | 25.0 | 28.0 |
| 11.10 | 2,190 | 102.0 | 387 | 356 | 92.0 | ..... | .... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 25.0 | 24.0 | 34.0 | 33.0 | 44.0 |
| 11.25 | 2,190 | 101.0 | 384 | 351 | 91.5 | 1,190 | 0.83 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 24.5 | 23.5 | 41.0 | 39.4 | 48.5 |
| 11.40 | 2,190 | 101.0 | 384 | 351 | 91.5 | 1,190 | 0.83 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 24.5 | 24.5 | 44.0 | 42.0 | 51.2 |
| 11.55 | 2,190 | 101.0 | 384 | 351 | 91.5 | 1,190 | 0.83 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 25.0 | 22.0 | 45.5 | 43.0 | 53.0 |
| P.M. | | | | | | | | | | | | | | | | | | | | | | | | |
| 12.10 | 2,190 | 101.0 | 384 | 351 | 91.5 | 1,190 | 0.83 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 24.0 | 22.5 | 46.0 | 43.0 | 54.0 |
| 12.25 | 2,185 | 108.0 | 409 | 379 | 92.7 | 1,185 | 1.25 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 24.0 | 22.0 | 47.0 | 44.5 | 54.0 |
| 12.40 | 2,185 | 107.0 | 406 | 378 | 93.0 | 1,185 | 1.25 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 23.8 | 22.0 | 47.0 | 45.0 | 54.0 |
| 12.45 | 2,178 | 108.7 | 410 | 377 | 91.8 | 1,181 | 1.58 | 2.0 | 2.3 | 183.5 | 423 | 429 | 128.4 | 3,400 | 369 | 505 | 73.0 | 94.7 | 77.1 | 24.0 | 22.0 | 48.0 | 45.0 | 54.0 |
| 1.00 | 2,185 | 107.9 | 409 | 377 | 92.2 | 1,186 | 1.17 | 2.0 | 2.3 | 183.5 | 423 | 429 | 128.4 | 3,400 | 369 | 505 | 73.0 | 94.7 | 77.1 | 23.5 | 24.0 | 49.0 | 45.5 | 55.0 |
| 1.10 | 2,178 | 104.0 | 392 | 363 | 92.6 | 1,182 | 1.50 | 2.0 | 2.3 | 191.5 | 441 | 447 | 114.0 | 3,205 | 362 | 485 | 74.6 | 94.7 | 78.8 | 25.0 | 25.0 | 49.5 | 47.0 | 55.5 |
| 1.25 | 2,184 | 103.6 | 392 | 363 | 92.6 | ..... | ..... | 2.0 | 2.3 | 190.5 | 439 | 445 | 114.0 | 3,205 | 361 | 485 | 74.4 | 94.7 | 78.5 | 25.5 | 24.5 | 50.0 | 47.0 | 56.0 |
| 1.40 | 2,198 | 102.8 | 391 | 354 | 90.6 | 1,182 | 1.50 | 2.0 | 2.3 | 195.5 | 450 | 456 | 101.8 | 3,025 | 349 | 474 | 73.7 | 94.65 | 77.9 | 25.5 | 24.5 | 49.5 | 46.5 | 56.5 |
| 1.50 | 2,188 | 102.4 | 388 | 353 | 91.0 | ..... | ..... | 2.1 | 2.4 | 196.0 | 452 | 458 | 100.8 | 3,015 | 350 | 473 | 74.1 | 94.65 | 78.3 | ..... | ..... | ..... | ..... | ..... |
| 2.00 | 2,198 | 95.0 | 361 | 331 | 91.6 | 1,189 | 0.92 | 2.5 | 2.8 | 199.0 | 459 | 464 | 82.5 | 2,725 | 321 | 443 | 72.5 | 94.5 | 76.7 | 25.5 | 23.5 | 48.5 | 45.5 | 57.0 |
| 2.20 | 2,188 | 95.4 | 362 | 331 | 91.4 | 1,192 | 0.67 | 2.5 | 2.8 | 199.0 | 459 | 464 | 82.5 | 2,725 | 321 | 443 | 72.5 | 94.5 | 76.7 | 25.5 | 25.0 | 47.5 | 44.5 | 56.5 |
| 2.25 | 2,198 | 88.3 | 336 | 309 | 92.0 | ..... | ..... | 3.0 | 3.4 | 203.5 | 469 | 474 | 57.5 | 2,275 | 273 | 414 | 66.0 | 94.2 | 70.1 | ..... | ..... | ..... | ..... | ..... |
| 2.35 | 2,198 | 81.2 | 309 | 280 | 90.7 | 1,195 | 0.42 | 3.5 | 4.0 | 212.0 | 488 | 492 | 36.6 | 1,815 | 226 | 375 | 60.2 | 94.0 | 64.0 | 25.5 | 25.0 | 46.0 | 43.5 | 57.0 |
| 2.40 | 2,198 | 72.6 | 276 | 250 | 90.6 | ..... | ..... | 3.5 | 4.0 | 218.0 | 502 | 506 | 17.5 | 1,255 | 161 | 336 | 47.9 | 93.8 | 51.0 | ..... | ..... | ..... | ..... | ..... |
| 2.45 | 2,201 | 65.6 | 250 | 223 | 89.2 | ..... | ..... | 4.0 | 4.5 | 225.0 | 518 | 521 | 0.0 | 0 | 0 | 299 | 0.0 | 93.5 | 0.0 | ..... | ..... | ..... | ..... | ..... |

*NOTE.—Temperatures as indicated below. Final rotor temperature after shut-down, 44° C.

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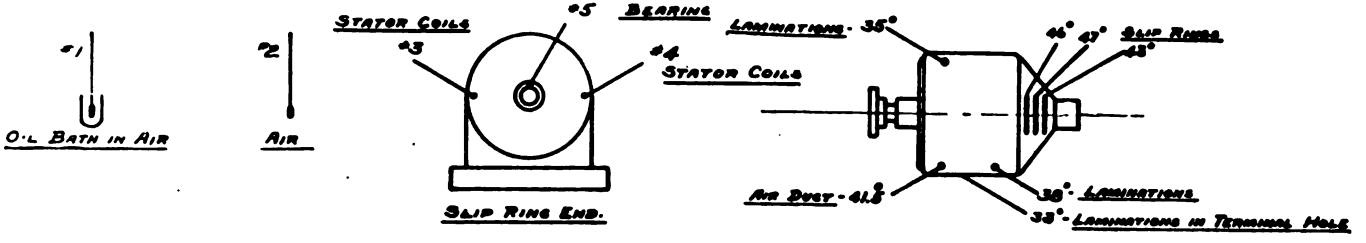
## TABLE VI.—*No. 2 Pump Unit for 16th Level Station.*

| Time | Motor | | | | | | | Pump | | | | | | | Efficiencies | | | | | *Temperatures | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | Head-feet | | | | | Capacity | | | | | | | | | | | |
| | Volts | Amperes | Kv.A. | Kw. | Power Factor | R.P.M. | Per Cent. Slip | Mercury Column Suction | Ft. S | Gauge Pressure Discharge | Ft. D | D−S+8 ft. Total | Gauge Pressure Nozzle | Gal. per Min. | Water Hp. | Motor Hp. Input | Wire to Water Efficiency | Motor Efficiency | Pump Efficiency | 1 | 2 | 3 | 4 | 5 |
| P.M. | | | | | | | | | | | | | | | | | | | | | | | | |
| 12.00 | 2,160 | 68.7 | 257 | .... | .... | ..... | | Hot P | acking | g on | Pump | p— | Shut | down | .... | ..... | ..... | ..... | ..... | 21.0 | 21.0 | 20.0 | 20.0 | 20.0 |
| 12.15 | 2,178 | 66.7 | 251 | 227 | 90.3 | 1,210 | | | Nos | zle | Chok | ed— | Shut | down | .... | ..... | ..... | ..... | ..... | 21.5 | 21.0 | 23.0 | 23.0 | 25.0 |
| 1.05 | 2,178 | 100.2 | 377 | 347 | 92.0 | 1,185 | | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 21.0 | 19.5 | 26.5 | 26.0 | 31.0 |
| 1.20 | 2,178 | 100.5 | 379 | 357 | 94.2 | 1,190 | | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 21.0 | 21.0 | 33.0 | 34.0 | 43.0 |
| 1.35 | 2,178 | 100.5 | 379 | 350 | 92.4 | 1,190 | | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 21.5 | 21.0 | 36.0 | 37.5 | 48.0 |
| 1.50 | 2,178 | 106.0 | 400 | 369 | 92.3 | 1,185 | | 2.0 | 2.3 | 174.5 | 402 | 408 | 133.8 | 3,475 | 358 | 495 | 72.4 | 94.7 | 76.4 | 21.5 | 21.0 | 38.0 | 39.5 | 51.0 |
| 2.20 | 2,178 | 105.1 | 396 | 365 | 92.2 | | | 2.0 | 2.3 | 174.0 | 401 | 407 | 132.3 | 3,455 | 356 | 490 | 72.6 | 94.65 | 76.8 | 21.5 | 20.5 | 39.0 | 40.5 | 51.5 |
| 2.45 | 2,178 | 100.8 | 380 | 361 | 95.0 | | | 2.4 | 2.7 | 190.0 | 438 | 443 | 114.0 | 3,205 | 359 | 484 | 74.2 | 94.65 | 78.4 | 21.0 | 20.0 | 39.0 | 41.5 | 54.0 |
| 3.35 | 2,178 | 100.8 | 380 | 352 | 92.7 | | | 2.4 | 2.7 | 186.7 | 430 | 435 | 111.7 | 3,175 | 349 | 471 | 74.0 | 94.65 | 78.3 | 19.5 | 18.5 | 38.5 | 40.0 | 54.0 |
| 3.45 | 2,178 | 95.4 | 359 | 334 | 93.1 | | | 2.5 | 2.8 | 193.0 | 445 | 450 | 91.6 | 2,870 | 327 | 448 | 72.1 | 94.6 | 77.1 | 19.5 | 18.5 | 38.0 | 39.5 | 54.0 |
| 3.50 | 2,178 | 91.4 | 345 | 321 | 93.1 | | | 3.0 | 3.4 | 198.0 | 456 | 461 | 73.0 | 2,565 | 299 | 431 | 69.4 | 94.55 | 73.4 | ..... | ..... | ..... | ..... | ..... |
| 3.55 | 2,178 | 83.6 | 315 | 290 | 92.1 | | | 3.8 | 4.3 | 204.3 | 471 | 475 | 46.5 | 2,050 | 246 | 388 | 63.3 | 94.3 | 67.2 | ..... | ..... | ..... | ..... | ..... |
| 4.00 | 2,178 | 75.7 | 285 | 262 | 92.0 | | | 4.5 | 5.1 | 214.5 | 494 | 497 | 27.1 | 1,560 | 196 | 351 | 55.8 | 93.9 | 59.5 | 20.0 | 19.0 | 36.5 | 38.0 | 54.0 |
| 4.05 | 2,178 | 71.8 | 271 | 245 | 90.4 | | | 5.0 | 5.7 | 218.0 | 502 | 504 | 16.5 | 1,225 | 156 | 328 | 47.7 | 93.75 | 50.9 | ..... | ..... | ..... | ..... | ..... |
| 4.15 | 2,178 | 64.8 | 245 | 225 | 91.8 | | | 5.0 | 5.7 | 228.0 | 525 | 527 | 0.0 | 0 | 0 | 302 | 0.0 | 93.4 | 0.0 | ..... | ..... | ..... | ..... | ..... |
| 4.40 | 2,178 | 108.0 | 407 | 377 | 92.7 | | | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 21.0 | 21.0 | 39.0 | 40.5 | 53.5 |

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| 60 Cycles. | Generator Speed not Constant. | | | | | | |

*NOTE.—Temperatures as indicated below. Final temperatures after shut-down—#3 42° C., #4 43° C., rotor 42° C. Other temperature readings shown in diagram.

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## TABLE VII.—*No. 3 Pump Unit for 12th Level Station*

| Time | Motor: Volts | Amperes | Kv. A. | Kw. | Power Factor | R.P.M. | Pump: Per Cent. Slip | Mer. Col. Suct. | Head-feet: Feet S | Gauge Pressure Discharge: L.P. | H.P. | Feet D | D−S+44 ft. Total | Capacity: Gauge Pressure Nozzle | Gallons per Min. | Water H.P. | Efficiencies: Motor H.P. Input | Wire to Water Efficiency | Motor Efficiency | Pump Efficiency | Temperatures* 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1.35 | 2,218 | 146.0 | 560 | 522 | 93.2 | 1,192 | 0.67 | 5.0 | 5.7 | 260.5 | 525 | 1,210 | 1,211 | 7.5 | 770 | 236 | 700 | 33.8 | 93.85 | 36.1 | 15.0 | 15.0 | 15.0 | 14.0 | 14.0 | 20.0 | 20.0 | 15.0 | 15.0 | 15.0 | 15.0 |
| 1.55 | 2,178 | 235.5 | 888 | 852 | 96.0 | 1,184 | 1.33 | 2.8 | 3.2 | 220. | 440 | 1,014 | 1,016 | 103.7 | 3,055 | 784 | 1,141 | 68.6 | 95.05 | 72.2 | 15.0 | 15.0 | 31.0 | 31.0 | 43.0 | 25.0 | 25.0 | 34.0 | 29.0 | .... |
| 2.15 | 2,320 | 244.8 | 984 | 894 | 90.8 | 1,183 | 1.42 | 2.3 | 2.6 | 187.5 | 384 | 884 | 886 | 149.5 | 3,672 | 823 | 1,197 | 68.8 | 95.1 | 72.3 | 15.0 | 15.0 | 43.0 | 42.0 | 49.0 | 43.0 | 32.0 | 41.5 | 38.0 | 41.0 |
| 2.20 | 2,316 | 243.2 | 975 | 894 | 91.7 | 1,183 | 1.42 | 2.2 | 2.5 | 188. | 385 | 886 | 888 | 149.5 | 3,672 | 825 | 1,197 | 69.0 | 95.1 | 72.5 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 3.00 | 2,218 | 249.5 | 958 | 894 | 93.2 | 1,183 | 1.42 | 2.2 | 2.5 | 188. | 385 | 886 | 888 | 149.5 | 3,672 | 825 | 1,197 | 69.0 | 95.1 | 72.5 | 17.0 | 17.0 | 52.0 | 48.0 | 55.0 | 46.0 | 36.5 | 49.0 | 45.0 | 48.0 |
| 3.05 | 2,218 | 244.8 | 940 | 874 | 92.9 | 1,182 | 1.50 | 2.5 | 2.8 | 201. | 409 | 941 | 943 | 134.5 | 3,485 | 831 | 1,170 | 71.0 | 95.1 | 74.6 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 3.10 | 2,168 | 248.0 | 932 | 864 | 92.8 | ..... | ..... | 2.5 | 2.8 | 200. | 407 | 937 | 939 | 134.5 | 3,485 | 830 | 1,157 | 71.7 | 95.05 | 75.5 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 3.15 | 2,205 | 235.5 | 900 | 841 | 93.4 | 1,183 | 1.42 | 2.75 | 3.1 | 216. | 440 | 1,014 | 1,016 | 115.0 | 3,220 | 827 | 1,126 | 73.5 | 95.05 | 77.4 | 13.0 | 8.0 | 53.5 | 50.0 | 56.0 | 47.0 | 38.0 | 49.5 | 39.0 | 48.0 |
| 3.20 | 2,178 | 235.5 | 888 | 833 | 93.8 | ..... | ..... | 2.75 | 3.1 | 215. | 440 | 1,014 | 1,016 | 115.0 | 3,220 | 827 | 1,115 | 74.2 | 95.0 | 78.2 | 14.0 | 12.0 | 51.0 | 47.0 | 55.0 | 46.0 | 36.0 | 47.5 | 40.0 | 47.0 |
| 3.30 | 2,146 | 230.8 | 858 | 800 | 93.2 | 1,184 | 1.33 | 3.0 | 3.4 | 227. | 462 | 1,064 | 1,065 | 98.8 | 2,980 | 803 | 1,071 | 74.9 | 94.95 | 78.8 | 15.0 | 16.0 | 51.0 | 48.0 | 56.0 | 46.0 | 39.0 | 48.0 | 40.0 | 47.0 |
| 3.35 | 2,198 | 225.4 | 858 | 807 | 94.0 | ..... | ..... | 3.0 | 3.4 | 229. | 463 | 1,067 | 1,068 | 98.8 | 2,980 | 804 | 1,080 | 74.5 | 94.95 | 78.5 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 3.40 | 2,158 | 216.0 | 807 | 760 | 94.1 | 1,184 | 1.33 | 3.5 | 4.0 | 243.5 | 493 | 1,136 | 1,137 | 78.4 | 2,660 | 764 | 1,018 | 75.0 | 94.85 | 79.1 | 11.0 | 11.0 | 51.0 | 48.0 | 56.0 | 46.0 | 36.5 | 47.5 | 39.5 | 47.0 |
| 3.45 | 2,198 | 196.2 | 746 | 702 | 94.0 | 1,186 | 1.17 | 4.0 | 4.5 | 258. | 516 | 1,189 | 1,189 | 56.5 | 2,258 | 679 | 941 | 72.1 | 94.7 | 76.2 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 3.50 | 2,168 | 199.4 | 749 | 702 | 93.6 | ..... | ..... | 4.0 | 4.5 | 258. | 517 | 1,191 | 1,191 | 56.5 | 2,258 | 680 | 941 | 72.1 | 94.7 | 76.2 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 3.55 | 2,178 | 186.8 | 704 | 659 | 93.5 | ..... | ..... | 4.25 | 4.8 | 262.5 | 527 | 1,213 | 1,213 | 42.0 | 1,950 | 599 | 883 | 67.8 | 94.55 | 71.7 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 4.00 | 2,178 | 172.7 | 651 | 612 | 94.0 | 1,187 | 1.08 | 4.5 | 5.1 | 270. | 543 | 1,251 | 1,251 | 31.4 | 1,680 | 532 | 820 | 65.0 | 94.35 | 69.0 | 16.5 | 17.0 | 48.0 | 45.0 | 55.5 | 47.0 | 35.0 | 46.0 | 38.0 | 45.0 |
| 4.05 | 2,188 | 164.8 | 624 | 574 | 92.0 | ..... | ..... | 4.8 | 5.4 | 269. | 539 | 1,241 | 1,240 | 26.0 | 1,525 | 480 | 770 | 62.3 | 94.15 | 66.2 | 12.0 | 12.0 | 46.0 | 45.0 | 58.0 | 50.0 | 33.0 | 45.0 | 39.5 | 45.0 |
| 4.10 | 2,198 | 117.8 | 448 | 406 | 90.6 | 1,196 | 0.33 | 5.5 | 6.2 | 268. | 530 | 1,221 | 1,220 | 0.0 | 0 | 0 | 544 | 0.0 | 92.85 | 0.0 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| | | | | | | | | | | | | | | 0.0 | | | | 0.0 | | | .... | .... | 45.0 | 44.0 | .... | .... | 42.5 | 49.0 | 43.0 | 49.0 |

* NOTE.—Temperatures as indicated below. Final temperatures after shut-down. Rotor—41° C. Slip rings—see diagram below.

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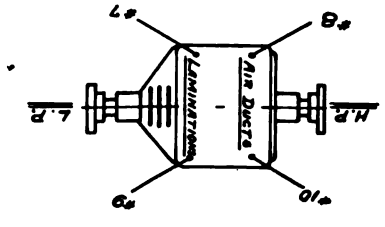
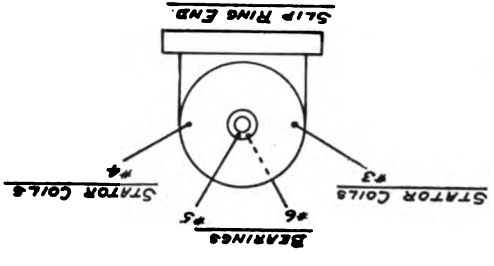
## TABLE VIII.—*No. 4 Pump Unit for 12th Level Station*

| Time | Motor | | | | | | | Pump | | | | | | | | Efficiencies | | | | | Temperatures* | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | Head-feet | | | | | | Capacity | | | | | | | | | | | | | | | |
| | | | | | | | | | | Gauge Pressure Discharge | | | | | | | | | | | | | | | | | | | |
| | Volts | Amperes | Kv.A. | Kw. | Power Factor. | R.P.M. | Per Cent. Slip | Mer. Col. Suct. | Feet S | L.P. | H.P. | Feet D | D−S+4½ ft. Total | Gauge Pressure Nozzle | Gallons per Min. | Water H.P. | Motor H.P. Input | Wire to Water Efficiency | Motor Efficiency | Pump Efficiency | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 |
| P.M. | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| 9.15 | 2,178 | 141.3 | 533 | 492 | 92.2 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 8.0 | 8.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 9.30 | 2,178 | 230.8 | 871 | 815 | 93.5 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 103.7 | 3,055 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 9.55 | 2,175 | 250.2 | 944 | 886 | 93.9 | 1,186 | 1.17 | 1.25 | 1.4 | 177. | 364 | 838 | 841 | 152.5 | 3,710 | 790 | 1,188 | 66.5 | 95.1 | 70.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.05 | 2,175 | 245.8 | 926 | 866 | 93.4 | ..... | ..... | 1.5 | 1.7 | 188. | 387 | 891 | 894 | 139.5 | 3,550 | 804 | 1,162 | 69.2 | 95.1 | 72.8 | 8.0 | 8.0 | 35.0 | 39.0 | 40.0 | 45.0 | 31.0 | 31.0 | 26.0 | 29.0 |
| 10.10 | 2,178 | 244.8 | 924 | 869 | 93.9 | 1,186 | 1.17 | 1.5 | 1.7 | 188. | 387 | 891 | 894 | 140.5 | 3,560 | 806 | 1,165 | 69.3 | 95.1 | 72.9 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.12 | 2,175 | 241.0 | 908 | 852 | 93.8 | ..... | ..... | 1.75 | 2.0 | 199. | 405 | 932 | 935 | 128.2 | 3,400 | 805 | 1,142 | 70.5 | 95.1 | 74.1 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.15 | 2,178 | 233.2 | 880 | 831 | 94.4 | 1,187 | 1.08 | 2.0 | 2.3 | 209. | 428 | 985 | 987 | 118.0 | 3,265 | 816 | 1,114 | 73.2 | 95.0 | 77.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.25 | 2,168 | 226.6 | 852 | 806 | 94.6 | 1,186 | 1.17 | 2.25 | 2.6 | 220. | 448 | 1,032 | 1,034 | 103.7 | 3,055 | 800 | 1,080 | 74.1 | 94.95 | 78.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.30 | 2,178 | 217.5 | 820 | 772 | 94.1 | 1,188 | 1.00 | 2.5 | 2.8 | 233. | 471 | 1,086 | 1,088 | 88.5 | 2,820 | 776 | 1,035 | 75.0 | 94.9 | 79.1 | 8.0 | 7.5 | 37.0 | 43.0 | 44.0 | 51.5 | 33.0 | 34.0 | 28.0 | 31.0 |
| 10.35 | 2,182 | 205.6 | 776 | 734 | 94.5 | 1,189 | 0.92 | 3.0 | 3.4 | 247.5 | 502 | 1,156 | 1,157 | 67.0 | 2,457 | 719 | 984 | 73.0 | 94.8 | 77.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.40 | 2,182 | 186.0 | 703 | 662 | 94.1 | ..... | ..... | 3.3 | 3.7 | 259.5 | 522 | 1,201 | 1,202 | 47.0 | 2,060 | 626 | 887 | 70.5 | 94.55 | 74.6 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.45 | 2,188 | 179.0 | 678 | 625 | 92.1 | 1,191 | 0.75 | 3.5 | 4.0 | 262.5 | 528 | 1,215 | 1,216 | 36.7 | 1,820 | 560 | 838 | 66.8 | 94.4 | 70.7 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.50 | 2,184 | 168.0 | 635 | 592 | 93.1 | 1,191 | 0.75 | 3.75 | 4.2 | 269. | 539 | 1,240 | 1,241 | 27.0 | 1,555 | 488 | 793 | 65.7 | 94.25 | 69.7 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 10.55 | 2,188 | 113.1 | 428 | 376 | 87.8 | ..... | ..... | 4.5 | 5.1 | 261.5 | 524 | 1,206 | 1,206 | 0.0 | 0 | 0 | 504 | 0.0 | 92.5 | 0.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 11.00 | 2,188 | 250.2 | 948 | 893 | 94.1 | 1,186 | 1.17 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 8.0 | 7.5 | 30.0 | 42.0 | 44.0 | 51.0 | 29.0 | 30.0 | 27.0 | 27.0 |
| 11.30 | 2,188 | 250.2 | 948 | 893 | 94.1 | 1,186 | 1.17 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 8.0 | 7.0 | 39.0 | 43.0 | 47.0 | 53.5 | 33.0 | 29.0 | 35.0 | 30.0 |

* NOTE.—Temperatures as indicated below.



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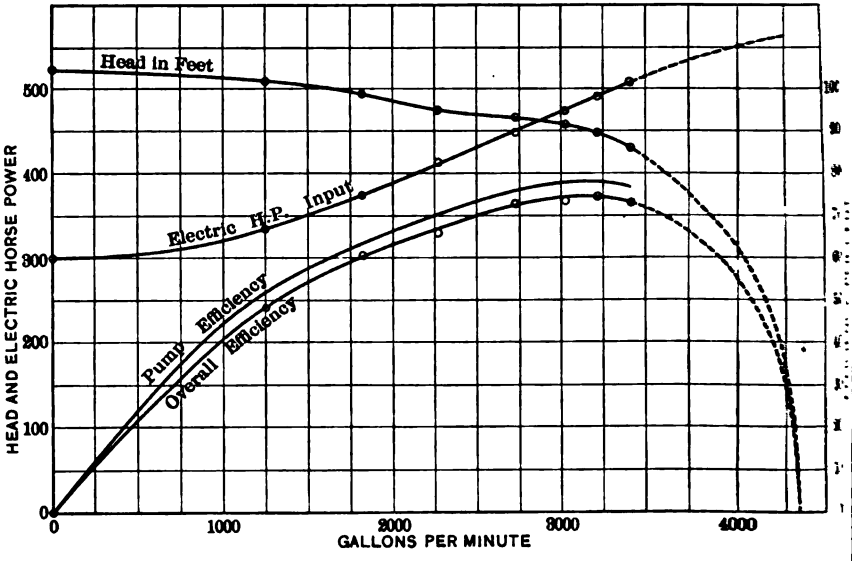


FIG. 7.—PUMP NO. 1, 16TH LEVEL.

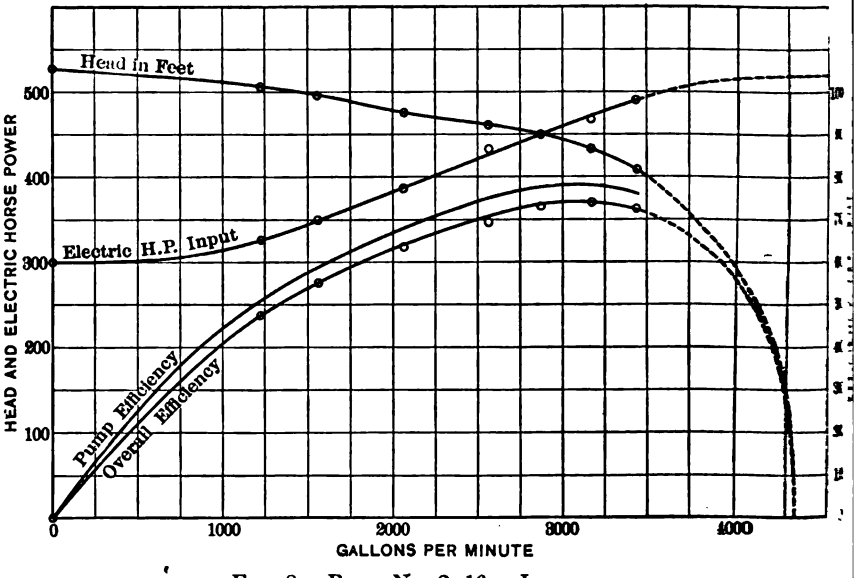


FIG. 8.—PUMP NO. 2, 16TH LEVEL.

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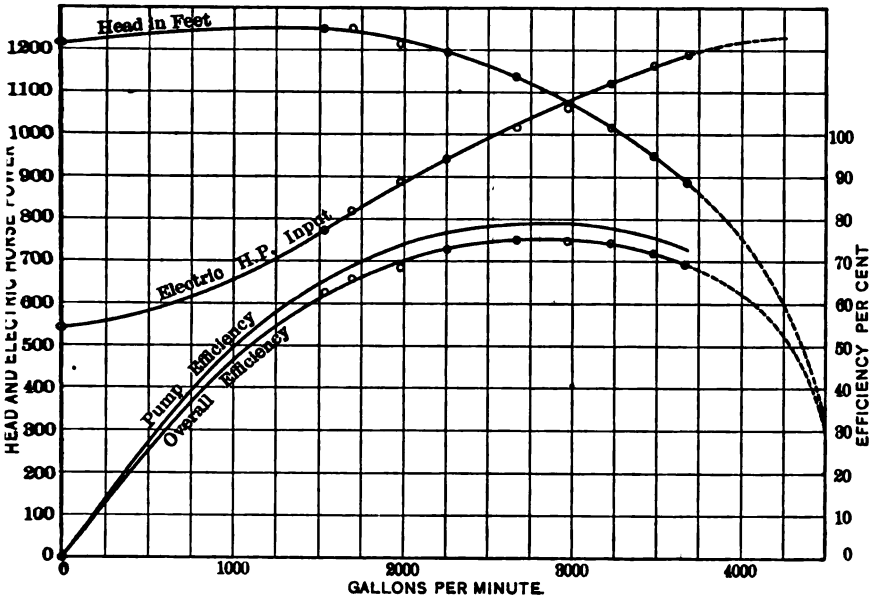


FIG. 9.—PUMP NO. 3, 12TH LEVEL.

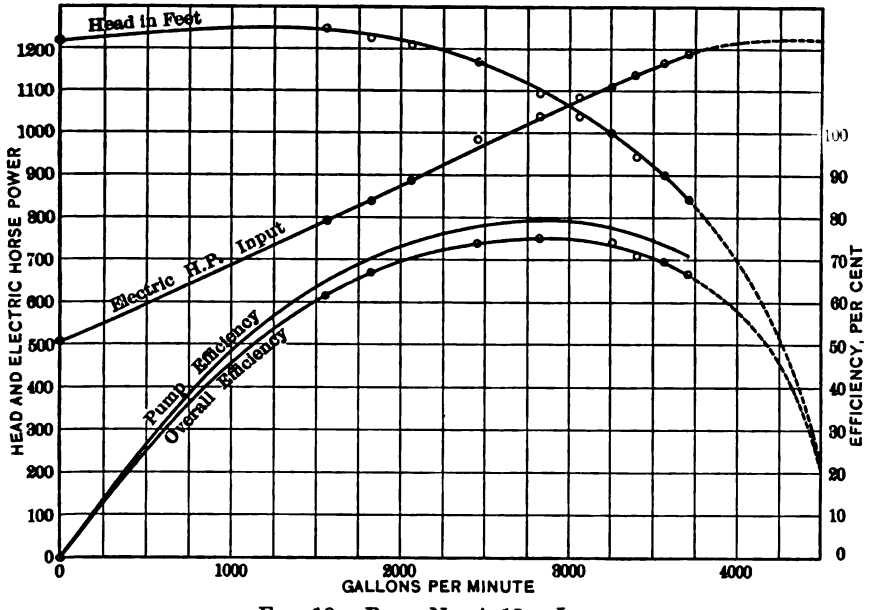


FIG. 10.—PUMP NO. 4, 12TH LEVEL.

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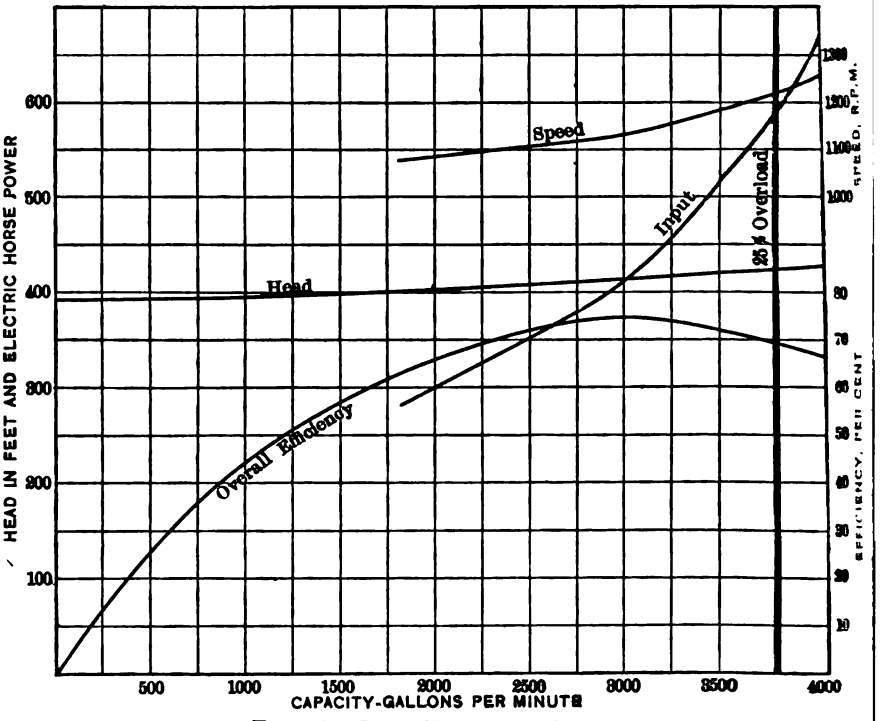


FIG. 11.—PUMP NO. 1, 16TH LEVEL.

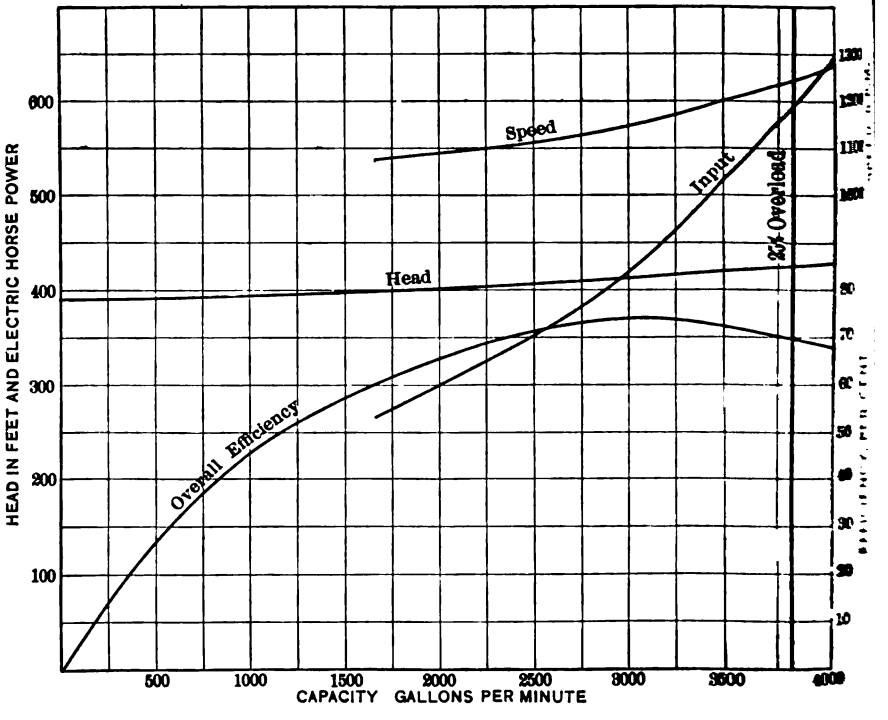


FIG. 12.—PUMP NO. 2, 16TH LEVEL.

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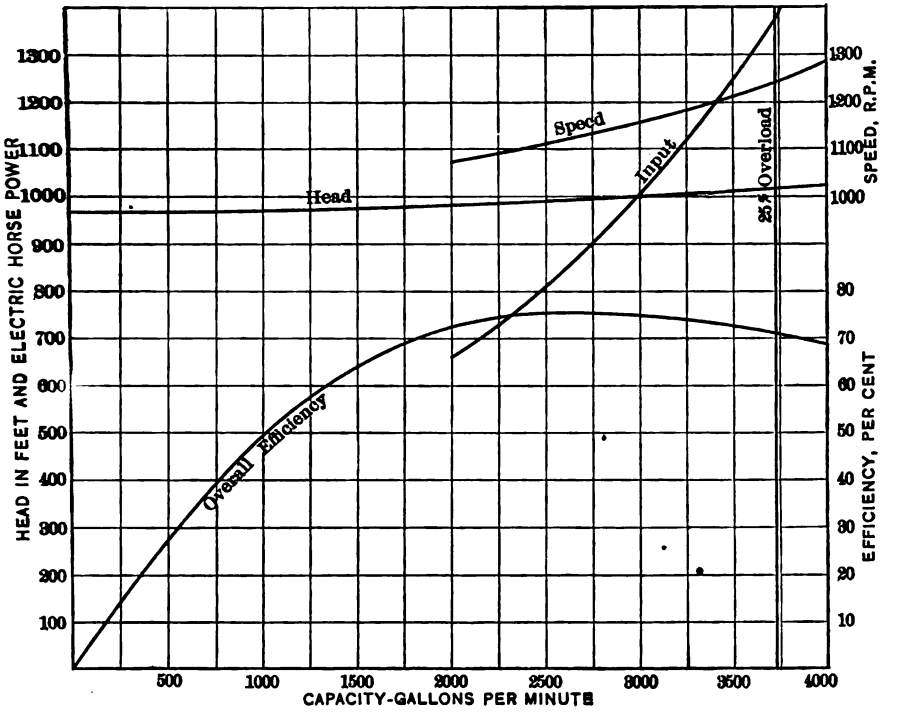


FIG. 13.—PUMP NO. 3, 12TH LEVEL.

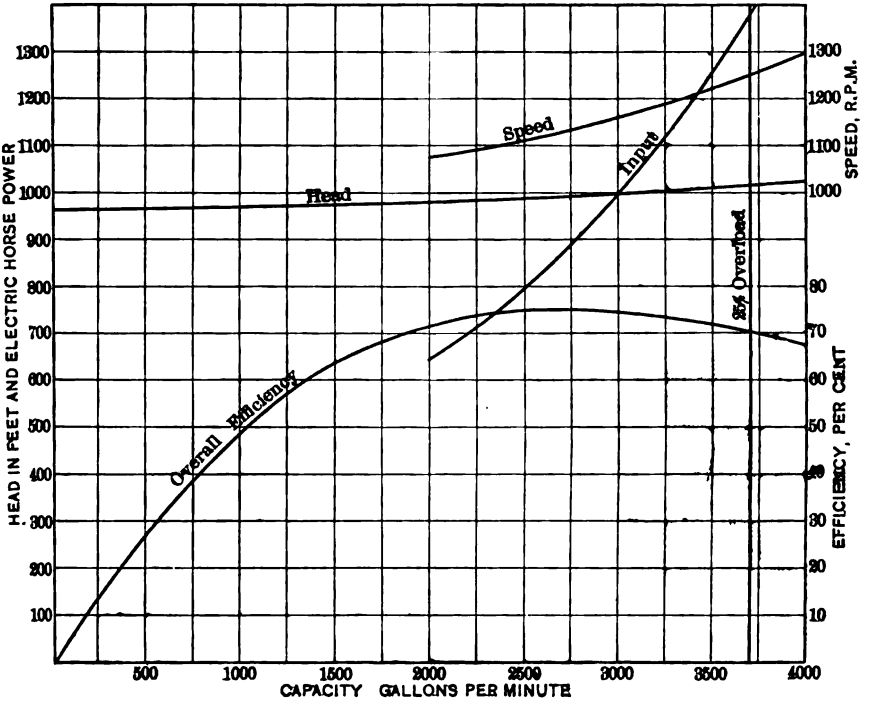


FIG. 14.—PUMP NO. 4, 12TH LEVEL.

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TABLE IX

| | Pump No. 1 | Pump No. 2 | Pump No. 3 | Pump No. 4 |
|---|---|---|---|---|
| *At 3,000 gal. per minute* | | | | |
| Obtained head in feet | 458 | 445 | 1,068 | 1,064 |
| Guaranteed head in feet | 412 | 412 | 1,000 | 1,000 |
| Guarantee exceeded by | 46 | 33 | 68 | 64 |
| *At 412 ft. head and 1,000 ft. head, respectively* | | | | |
| Obtained gallons per minute | 3,550 | 3,415 | 3,275 | 3,250 |
| Guaranteed gallons per minute | 3,000 | 3,000 | 3,000 | 3,000 |
| Guarantee exceeded by | 550 | 415 | 275 | 250 |
| *Wire to water efficiencies at 3,000 gal.* | | | | |
| Obtained efficiencies, per cent | 74.0 | 73.8 | 75.0 | 75.0 |
| Guaranteed efficiencies, per cent | 71.8 | 71.8 | 73.3 | 73.3 |
| Guarantee exceeded by, per cent | 2.2 | 2.0 | 1.7 | 1.7 |

TABLE X

| | Time in Minutes | Temperature rise above 25° C. | Average load in Horsepower |
|---|---|---|---|
| Pump No. 1 | 150 | 25.5 | 457 |
| Pump No. 2 | 155 | 20. | 450 |
| Pump No. 3 | 140 | 43. | 1,200 = 14.3 per cent. overload |
| Pump No. 4 | 75 | 37.5 | 1,170 = 10.5 per cent. overload |

All bearing temperatures were below normal throughout the tests. The pump bearings and oil for thrusts are water cooled and, although provision is made for the water cooling of motor bearings, no water was passed through the coils during the tests.

Some difficulty was experienced in pump thrust bearings due to foaming of oil, but this was finally overcome by a re-arrangement of oil piping and oil throwers.

*Mechanical Operation*

On test the pumps ran very quietly at all loads and the three-stage pumps with little vibration. Some vibration occurred in the six-stage pumps, because of the difficulty in holding alignment of the long cast-

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iron bases on the steel floor plates. Absolute alignment of these pumps is of prime importance and their successful operation will depend more upon this than any other one factor.

The curves in Figs. 11, 12, 13 and 14 have been prepared from the foregoing data in order to show pump characteristics at different speeds. With a variation in frequency of the prime movers from 55 to 65 cycles, a variation in pump output in gallons per minute can be obtained which will probably be within the limits required during normal pumping operations, which is at present between 3,000 and 3,250 gal. per minute. Because of the flat character of the speed-capacity curves shown, the importance of supplying a constant frequency for uniform pump output is clearly indicated.

The maximum continuous output of the pumps will be limited by the overload capacity of motors, which will probably be 25 per cent. This point is indicated on the curves.

Because of the slight difference in head-capacity curves of the three-stage and six-stage pumps, adjustment between pumps on 12th and 16th levels can be made with their discharge valves in order to pump equal quantities of water.



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[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Pittsburg meeting, October, 1914, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1914. Any discussion offered thereafter should preferably be in the form of a new paper.

# Coal-Mine Explosions Caused by Gas or Dust

BY HOWARD N. EAVENSON, GARY, W. VA.

(Pittsburg Meeting, October, 1914)

IN a discussion in the *Transactions* of the Institute (vol. xl, page 835 *et seq.*) the writer gave some data about explosions of gas and dust in the coal mines of the United States, Canada, and Mexico, among which was a table showing all explosions in the countries mentioned, in which five or more fatalities occurred, which had been reported from Oct. 2, 1871, to Mar. 28, 1908. Since the date of publication of this discussion, as opportunity has presented, this table has been extended by the addition of explosions occurring since that time, by including several explosions which occurred prior to 1871, and by adding to the list all explosions of gas or dust in which less than five fatalities occurred, this having been done by a search through the various mine reports, partly by the author and partly by the Librarian of the United Engineering Societies, and by correspondence with mine inspection officials of the various States, to all of whom acknowledgment of assistance is made. The accompanying list, Table I, is believed to be as nearly correct as such a statement can be made, and, while only extending to Dec. 20, 1912, contains all of the data now available in reports.

An analysis of Table I, showing the occurrence by months of all explosions of gas or dust, including those in which less as well as those in which more than five fatalities occurred, is given in Table II.

The statements are frequently made, in technical papers, that the number of explosions, particularly the larger ones, is increasing more rapidly than the number of mines, and that the number of fatalities caused by mine explosions, particularly the more disastrous ones, is increasing faster, proportionally, than the production of coal. To investigate the correctness of these statements Table III was prepared, in which the number of accidents and fatalities is taken from Table I, using those in the United States only, on account of lack of other data for Canada and Mexico; the production is from the records of the U.S. Geological Survey, and the number of mines, from 1909, from the same source; previous to 1909, the number of mines, for the decennial periods, was furnished by the U. S. Census records.

It had been the writer's intention to prepare similar data for European mines, but after considerable correspondence and labor this has been found

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impossible without convenient access to the records, which he has been unable to have. From the files of the *Colliery Guardian*, and from the editor of that paper, a great deal of data about British explosions was obtained; for the years 1889 to 1904, inclusive, the tables showing maximum and minimum temperatures, barometer readings, etc., published in the *Proceedings of North of England Institute of Mining and Mechanical Engineers*, vol. xxix, *et seq.*, gave complete data for all explosions; those between 1658 and 1841, inclusive, were obtained from Appendix E, Report of South Shields Committee to Investigate the Causes of Accidents in Coal Mines, by James Mather, Secretary.

The complete list is contained in Table IV. It is believed to be correct for those explosions in which more than five fatalities occurred, but is incomplete for those in which less than this number were killed.

An analysis of Table IV, showing the occurrence of the various kinds of explosions by months, is shown in Table V.

Through the courtesy of J. Taffanel, of Lievin, France, Mr. Lemaire prepared for the writer a list of the recorded explosions in French and Belgian mines, that for the French mines covering the period from 1814 to 1903, inclusive, no later data being available, and for the Belgian mines from 1887 to 1909, inclusive.

The list of French explosions is given in Table VI, and the analysis showing the occurrence by months of the various kinds of explosions in Table VII.

The list of explosions in Belgian mines is given in Table VIII, and the analysis showing occurrence by months in Table IX.

For more convenient study, the data shown in Table III, for the years since 1870, when our modern records may be said to begin, have been grouped into five-year periods, the results being shown in Table X. Too much weight must not be attached to the "Accidents per mine" columns, as prior to 1909 the number of mines is obtained from census reports and it is evident that a "mine" did not have the same meaning in each case, as the difference in number of mines reported in 1880 and in 1889 will show. The number of mines for each five-year period, of course, is interpolated from the data in Table III.

### *Conclusions*

From the data given, the following conclusions are evident:

1. In North America, minor explosions, or those in which less than five fatalities occurred, happen most frequently in October, November, December, January, and March, although nearly as many have happened in June as in March; those in December, January, and February are above the average in fatalities, as are also those in May and July, those occurring in May having a slightly greater average fatality than even those of December.

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Serious explosions, or those in which more than five lives have been lost, have happened most frequently in January, February, March, April, and November. Contrary to the usual belief, the number of explosions in December has been slightly below the average, although their intensity, and the number of lives lost, have been considerably greater than those of any other month. January, February, and May are also above the average in the number of fatalities per explosion, May being next to December in this respect.

For all explosions of gas or dust, January, March, November, October, and December, in the order named, are above the average in *number* of explosions; in number of fatalities per explosion, May, December, February, January, and March are above the average. For all explosions, therefore, May exceeds any of the winter months in number of fatalities per explosion.

2. In the coal mines of the United States, the total number of accidents and of fatalities due to explosions of gas or dust has been steadily increasing; there has been a slight increase in the number of accidents and a more decided increase in the number of fatalities per million tons produced; the serious explosions, causing five or more fatalities each, have been steadily increasing in actual number and number of fatalities, as well as in number of accidents and of fatalities per million tons produced; the number of accidents per mine, both serious and total, also shows a steady increase. It is therefore true that we have more explosions, and more serious ones, both actually and in relation to our number of mines and production, than we had years ago.

3. In the coal mines of Great Britain, minor explosions, so far as our records show, have occurred most frequently in the months of August, October, May, March, and September, while those in April, May, August, November, March, and July are above the average in number of fatalities per explosion. Serious explosions have occurred most frequently in December, October, November, and March, while those in June, December, July, February, and May have been of more than average intensity. For all explosions, August, October, December, March, May, and November are above the average in number, and December, June, July, February are above the average in intensity. By far the largest number of fatalities has occurred in December, June being second in this respect.

4. In the coal mines of France, from 1814 to 1904, minor explosions have occurred most frequently in July, August, February, April, May, and January and have been of more than average intensity in April, December, July, February, August, and October. Serious explosions have been above the average in frequency in August, April, July, and March, and in intensity in January, September, December, July, November, and October. For all explosions, July, August, April, and May are above the average in number, and December, January, November, July, March,

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October, and September in intensity. By far the largest number of both accidents and fatalities have occurred in July.

5. In the coal mines in Belgium, 1891 to 1909, minor explosions occurred more frequently than the average in June, May, and July, and in March, January, February, April, and June were of more than average intensity. Serious explosions occurred most frequently in May, March, and July and were of more than average intensity in March. For all explosions, May, July, and June are above the average in number and March in intensity. By far the largest number of fatalities occurred in March.

TABLE I.—*Mine Explosions Supposed to be Caused by Gas or Dust, Officially Reported in North America*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1839–March 18, | Black Heath, Va. | 40 |
| 1854–May 15, | Chesterfield, Va. | 19 |
| 1855–May 19, | Midlothian, Va. | 42 |
| 1869–Aug. 6, | Wadlingers, Pa. | 1 |
| Dec. 13, | Mine Hill Gap, Pa. | 1 |
| 1870–Feb. 11, | Norwegian Mine, Pa. | 1 |
| May 28, | Hein & Glasmire. | 1 |
| Oct. 18, | Phoenix No. 3, Pa. | 1 |
| Oct. 29, | St. Clair Mine, Pa. | 1 |
| Dec. 13, | Beechwood, Pa. | 1 |
| Dec. 18, | Beechwood, Pa. | 1 |
| 1871–Jan. 4, | Forestville, Pa. | 2 |
| Jan. 4, | Ravensdale, Pa. | 1 |
| Jan. 22, | Revenue, Pa. | 1 |
| April 14, | Big Lick, Pa. | 1 |
| April 15, | N. Philadelphia Mine, Pa. | 1 |
| June 2, | Live Oak Mine, Pa. | 2 |
| June 18, | Feeder Dam Mine, Pa. | 1 |
| July 22, | Swift Creek Mine, Pa. | 1 |
| Sept. 12, | Pine Forest, Pa. | 1 |
| Sept. 26, | Buckville, Pa. | 2 |
| Oct. 2, | Otto Red Ash, Pa. | 5 |
| Oct. 14, | Beechwood, Pa. | 2 |
| Oct. 3, | Otto Red Ash, Pa. | 1 |
| 1872–Jan. 1, | Montilius, Pa. | 1 |
| April 10, | Thomaston, Pa. | 1 |
| April 17, | Locust Run, Pa. | 2 |
| April 25, | Otto Red Ash, Pa. | 2 |
| Aug. 14, | Lower Ranch Creek, Pa. | 2 |
| Sept. 1, | East Pine Knot Mine, Pa. | 1 |
| Sept. 1, | St. Clair Shaft, Pa. | 1 |
| Sept. 27, | Diamond, Pa. | 7 |
| Oct. 14, | Otto White Ash, Pa. | 1 |
| Nov. 8, | Otto White Ash, Pa. | 2 |
| Dec. 12, | Daniel Webster Mine, Pa. | 1 |
| 1873–Jan. 28, | St. Clair Shaft, Pa. | 2 |
| Feb. 27, | Daniel Webster Mine, Pa. | 1 |
| April 29, | New Kirk, Pa. | 1 |
| May 13, | Intercolonial Coal Co., N. S. | 55 |
| June 8, | Glen Dower, Pa. | 1 |
| June 10, | Henry Clay, Shamokin, Pa. | 10 |
| Sept. 6, | Eagle Hill Mine, Pa. | 1 |
| 1873–Oct. 10, | Gilberton Mine, Pa. | 1 |
| Oct. 11, | Gilberton Mine, Pa. | 1 |
| Nov. 10, | Plank Ridge Mine, Pa. | 1 |
| Nov. 26, | Honey Brook Mine, Pa. | 1 |
| 1874–April 12, | Big Mine Run, Pa. | 1 |
| April 14, | Anchor Mine, Pa. | 1 |
| June 2, | Tunnel No. 8 Mine, Pa. | 1 |
| June 8, | Ravensdale Mine, Pa. | 1 |
| June 23, | Beechwood Mine, Pa. | 3 |
| July 27, | Anchor Mine, Pa. | 2 |
| Sept. 2, | Shenandoah City, Pa. | 1 |
| Sept. 26, | Treverton Mine, Pa. | 1 |
| Sept. 28, | Anchor Mine, Pa. | 1 |
| Oct. 23, | Big Mountain Mine, Pa. | 1 |
| Oct. 26, | Mariam Mine, Pa. | 1 |
| Nov. 2, | Buckville, Pa. | 1 |
| Nov. 10, | Mine Hill Gap Mine, Pa. | 2 |
| Dec. 1, | Girard, Pa. | 1 |
| 1875–Sept. 30, | Anchor Mine, Pa. | 3 |
| Sept. 30, | Hickory Shaft Mine, Pa. | 1 |
| Oct. 12, | Keystone Mine, Pa. | 1 |
| Dec. 5, | Ellangoan Mine, Pa. | 1 |
| Dec. 12, | Hickory Shaft Mine, Pa | 1 |
| Dec. 13, | Hickory Shaft Mine, Pa | 1 |
| 1876–Jan. 20, | Forestville, Pa. | 1 |
| Jan. 28, | Parsons Spring, Pa. | 1 |
| Feb. 12, | Exeter, West Pittston, Pa | 4 |
| 1876–Feb. 23, | Black Mine, Pa. | 1 |
| March, | Exeter, West Pittston, Pa. | 1 |
| May 20, | Midlothian, Va. | 8 |
| June 2, | Phoenix No. 2, Pa. | 1 |
| June 6, | Phoenix No. 2, Pa. | 1 |
| June 8, | Phoenix No. 2, Pa. | 1 |
| June 20, | Phoenix No. 2, Pa. | 1 |
| July 24, | Black Diamond, Cal. | 6 |
| Oct. 24, | Anchor, Pa. | 1 |
| Oct. 31, | Hickory Shaft, Pa. | 1 |
| Nov. 6, | Hickory Shaft, Pa. | 1 |
| Nov. 20, | Plymouth, Pa. | 1 |
| Dec. 2, | Henry Colliery, Pa. | 1 |
| 1877–March 16, | Mine Hill Gap, Pa. | 2 |
| June 9, | Wadesville, Pa. | 7 |
| 1878–Jan. 15, | Potts Colliery, Pa. | 5 |

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## *Mine Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1878–Feb. 28, | Preston Colliery, Pa | 1 |
| April 8, | Keystone, Pa | 2 |
| May 21, | Sydney, N. S | 5 |
| June 15, | Short Mt. Slope, Pa | 1 |
| Nov. 15, | Merriam Colliery, Pa | 1 |
| Nov. 21, | Sullivan, Ind | 8 |
| 1879–Jan. 1, | Beechwood, Pa | 1 |
| March 4, | Hausch Creek, Pa | 2 |
| May 28, | Wyoming, Pa | 1 |
| June 20, | Mill Creek, Pa | 5 |
| July 21, | Beechwood, Pa | 1 |
| Oct. 2, | Thomaston, Pa | 1 |
| Nov. 6, | Audenreid, Pa | 6 |
| 1880–April 8, | Preston No. 2, Pa | 1 |
| May 3, | Henry Clay Shaft, Pa. | 1 |
| May 3, | Lykens Valley Slope, Pa | 5 |
| Nov. 27, | Cameron, Pa | 1 |
| 1881–Jan. 8, | Diamond Colliery, Pa | 1 |
| Jan. 14, | Twin Shaft, Pittston, Pa | 1 |
| Feb. 10, | Robbins, Ohio | 6 |
| March 4, | Almy, Wyoming | 38 |
| March 5, | Shaft No. 2, Nanticoke, Pa. | 6 |
| April 11, | Van Storch Slope, Providence, Pa | 1 |
| May 7, | Central Shaft, Hyde Park, Pa | 1 |
| Sept. 5, | No. 4 Shaft, Pa | 2 |
| 1882–Feb. 3, | Midlothian, Va | 32 |
| March 25, | Laurel Run Slope, Pa | 1 |
| May 24, | Kohinoor Colliery, Pa | 5 |
| June 15, | Stanton Air Shaft, Pa | 2 |
| July 31, | West Penna., Pa | 1 |
| Sept. 5, | Wyoming Shaft, Pa | 1 |
| Sept. 6, | Conyngham, Pa | 2 |
| Sept. 25, | Dodson Shaft, Pa | 2 |
| Oct. 27, | Pine Ridge Shaft, Pa | 2 |
| Nov. 20, | Mahanoy City, Pa | 1 |
| 1883–Jan. 9, | Coulterville, Ill | 10 |
| Jan. 15, | Packer No. 4, Pa | 3 |
| Feb. 26, | Mt. Pleasant Slope, Scranton, Pa | 1 |
| March 12, | W. Penn, Pa | 1 |
| March 22, | Packer No. 2, Pa | 2 |
| April 19, | Keystone Colliery, Pa | 4 |
| May 4, | Dorrance, Pa | 2 |
| May 9, | No. 2 Shaft, Nanticoke, Pa. | 1 |
| May 21, | Mineral Spring, Pa | 2 |
| May 23, | Cameron Colliery, Pa | 1 |
| May 24, | Fair Lawn Slope, Scranton, Pa | 3 |
| June 5, | Luke Fidler Colliery, Pa | 1 |
| July 23, | No. 1 Shaft, Nanticoke, Pa | 1 |
| Oct. 29, | No. 7 Shaft, Pa | 2 |
| Nov. 5, | Enterprise, Pa | 1 |
| Dec. 13, | Eagle Shaft, Pittston, Pa | 1 |
| 1884–Jan. 24, | Crested Butte, Colo | 59 |
| Feb. 21, | Leisenring, No. 2, Pa | 19 |
| March 13, | Pocahontas, Va | 114 |
| April 14, | Lovedale, Pa | 2 |
| May 30, | Tunnel, Pa | 1 |
| Aug. 20, | Greenback, Pa | 7 |
| Oct. 24, | Youngstown, Pa | 14 |
| 1884–Dec. 8, | Henry Clay No. 1, P | 4 |
| 1885–Feb. 2, | Savanna, I. T | 6 |
| Feb. 17, | Hillman Vein, Pa | 2 |
| Feb. 18, | Vale Colliery, N. S | 13 |
| June 4, | No. 4 Slope, Nanticoke, Pa. | 3 |
| Oct. 21, | Plymouth, Pa | 6 |
| 1886–Jan. 13, | Almy, Wyoming | 13 |
| Jan. 21, | Newburg, W. Va | 39 |
| 1886–March 8, | Uniondale, Pa | 7 |
| Aug. 30, | Fair Lawn Colliery, Pa | 5 |
| Oct. 28, | Greenridge, Pa | 1 |
| Nov. 26, | Conyngham, Wilkes-Barre, Pa | 12 |
| 1887–April 2, | Slope No. 3, Luserne Co., Pa | 1 |
| April 4, | Savanna, I. T | 18 |
| June 23, | Slope No. 4, Luserne Co., Pa | 3 |
| Oct. 26, | Nottingham, Pa | 1 |
| 1888–March 29, | Rich Hill, Mo | 26 |
| Nov. 5, | Starkville No. 2, Colo | 2 |
| Nov. 7, | Kettle Creek, Pa | 17 |
| Nov. 9, | Frontenac, Kan | 40 |
| Dec. 3, | Newcastle, Colo | 3 |
| Dec. 10, | Coal Creek No. 2, Colo | 2 |
| 1889–Jan. 23, | Slope No. 4, Nanticoke, Pa. | 2 |
| Feb. 5, | Shaft No. 1, Edwardsdale, Pa. | 1 |
| April 14, | No. 3 Colliery, W. Nanticoke, Pa | 2 |
| April 23, | Shaft No. 2, Nanticoke, Pa. | 1 |
| May 10, | Powers Mine, Pa | 4 |
| June 10, | Greenridge Slope, Pa | 1 |
| July 24, | Central Shaft, Scranton, Pa. | 2 |
| Aug. 22, | Olyphant No. 2, Pa | 4 |
| Sept. 2, | Jersey No. 8, Ashley, Pa | 1 |
| 1890–Jan. 29, | Como No. 5, Colo | 1 |
| Feb. 1, | Nottingham Colliery, Pa | 8 |
| Mar. 3, | Shaft No. 3, S. Wilkes-Barre, Pa | 8 |
| April 2, | Slope No. 4, Nanticoke, Pa | 5 |
| May 15, | Jersey No. 8, Ashley, Pa | 28 |
| May 17, | Empire, Pa | 2 |
| Sept. 14, | Shaft No. 4, Edwardsdale, Pa | 1 |
| July 25, | Lincoln, Ill | 1 |
| 1891–Jan. 2, | Staunton, Ill | 1 |
| Jan. 3, | Centralia, Pa | 1 |
| Jan. 14, | Centralia No. 1, Nanticoke, Pa | 2 |
| Jan. 14, | Luke Fidler, Pa | 2 |
| Jan. 15, | Bast, Pa | 1 |
| Jan. 15, | Lincoln, Ill | 2 |
| Jan. 21, | Marissa, Ill | 3 |
| Jan. 27, | Mammoth, Pa | 109 |
| Feb. 21, | Springhill Mines, N. S | 125 |
| May 7, | Ocean, W. Va | 4 |
| May 8, | No. 10 Colliery Lehigh C. M. Co., Pa | 4 |
| Aug. 3, | West Fairmont, W. Va | 1 |
| Aug. 20, | Nottingham, Pa | 1 |
| Oct. 2, | York Farm, Pa | 1 |
| Oct. 8, | Richardson, Pa | 7 |

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## *Mine Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1891–Oct. 16, | Merriam, Pa. | 1 |
| Oct. 24, | Coal Ridge Mine, Colo. | 2 |
| Nov. 5, | New Boston, Pa. | 1 |
| Nov. 8, | Shaft No. 1, Nanticoke, Pa. | 12 |
| Dec. 26, | Cameron, Pa. | 1 |
| 1892–Jan. 4, | Primrose, Pa. | 1 |
| Jan. 7, | Krebs, Ind. Ter. | 67 |
| Jan. 18, | Glendon, Pa. | 3 |
| Jan. 22, | Packer No. 2, Pa. | 1 |
| Jan. 30, | Shaft No. 1, Lee Vein, Pa. | 3 |
| Feb. 20, | Packer No. 3, Pa. | 2 |
| March 9, | Alden, Pa. | 3 |
| May 10, | Roslyn, Wash. | 45 |
| May 31, | West End, Pa. | 4 |
| July 13, | Hillman Vein, Pa. | 1 |
| July 13, | Jersey No. 8, Pa. | 4 |
| July 14, | Shaft No. 1, Edwardsdale, Pa. | 2 |
| July 23, | York Farm, Pa. | 15 |
| Sept. 28, | Wolsen Mine, Colo. | 2 |
| Oct. 4, | Plymouth, Pa. | 1 |
| Oct. 8, | Shaft No. 2, Nanticoke, Pa. | 3 |
| Nov. 8, | Pekay Mine, Iowa. | 3 |
| Dec. 21, | Parrish, Pa. | 1 |
| Dec. 27, | Avondale, Pa. | 3 |
| 1893–Jan. 10, | Como, Colo. | 24 |
| Feb. 14, | Cedar Mine, Iowa. | 8 |
| March 2, | Glen Lyon, Pa. | 1 |
| March 27, | Shaft No. 1, Nanticoke, Pa. | 6 |
| April 5, | Shaft No. 4, Edwardsville, Pa. | 6 |
| April 7, | Big Mt. Mine, Tenn. | 5 |
| June 22, | Susquehanna No. 1, Nanticoke, Pa. | 5 |
| Sept. 19, | Dorrance, Pa. | 1 |
| Sept. 21, | Plymouth, Pa. | 6 |
| Oct. 20, | Shaft No. 3, Edwardsdale, Pa. | 1 |
| Dec. 22, | Krebs No. 11, Ind. Ter. | 3 |
| Dec. 26, | Nottingham, Pa. | 1 |
| 1894–Feb. 20, | Indian Ridge, Pa. | 1 |
| March 7, | Hillman Vein, Pa. | 1 |
| March 15, | Lawrence, Pa. | 1 |
| March 20, | Nottingham, Pa. | 2 |
| May 1, | Packer No. 5, Pa. | 2 |
| May 9, | Wimstown, Pa. | 1 |
| May 10, | Merriam, Pa. | 1 |
| May 14, | Bear Ridge, Pa. | 1 |
| May 14, | W. Penn, Pa. | 3 |
| May 28, | Nottingham, Pa. | 1 |
| June 23, | Girard, Pa. | 2 |
| July 16, | Wimstown, Pa. | 2 |
| July 16, | Henry Clay, Pa. | 1 |
| July 24, | Stanton, Pa. | 1 |
| July 25, | Phœnix No. 1, Ohio. | 1 |
| Aug. 16, | North Nanticoke, Pa. | 1 |
| Aug. 18, | Maple Hill, Pa. | 2 |
| Aug. 24, | Gilberton, Pa. | 1 |
| Aug. 24, | Shenandoah, Pa. | 1 |
| Oct. 9, | Coal Creek Mine, Wash. | 4 |
| Nov. 20, | Blanche, W. Va. | 8 |
| Nov. 27, | Jack Oak Mine, Colo. | 1 |
| 1894–Dec. 3, | Dorrance, Pa. | 1 |
| 1895–Jan. 22, | Sturgis, Ky. | 5 |
| Feb. 18, | Bear Ridge, Pa. | 5 |
| Feb. 21, | West Penn, Pa. | 1 |
| Feb. 27, | White Ash Mine, N. M. | 24 |
| March 4, | Packer No. 4, Pa. | 1 |
| March 20, | Red Canyon, Wyo. | 60 |
| April 8, | Blue Canyon, Wash. | 23 |
| April 17, | South No. 1 Shaft, Nanticoke, Pa. | 1 |
| May 10, | Sopris Mine, Colo. | 3 |
| July 6, | Glen Lyon, Pa. | 1 |
| July 9, | T. C. I. & R. R. No. 1, Ala. | 1 |
| Aug. 16, | Hammond, Pa. | 1 |
| Oct. 7, | Dorrance, Pa. | 7 |
| Oct. 19, | Knickerbocker, Pa. | 1 |
| Oct. 31, | Hillman Vein, Pa. | 1 |
| Dec. 12, | Indian Ridge, Pa. | 1 |
| Dec. 19, | Cumnock, N. C. | 39 |
| Dec. 20, | Dayton, Tenn. | 25–9 |
| 1896–Feb. 18, | New Castle, Colo. | 49 |
| March 23, | Berwinsdale, Pa. | 15 |
| March 25, | Shaft No. 1, Edwardsdale, Pa. | 1 |
| April 1, | Shaft No. 9, Sugar Notch, Pa. | 2 |
| April 27, | Alderson, Ind. Ter. | 1 |
| May 10, | Glenwood, Ohio. | 1 |
| May 26, | Alderson, Ind. Ter. | 1 |
| June 8, | Alderson, Ind. Ter. | 2 |
| July —, | Woodward, Pa. | 1 |
| Aug. 11, | Dodson, Pa. | 1 |
| Aug. 15, | Franklin, Pa. | 1 |
| Aug. 19, | Baltimore No. 2, Wilkes-Barre, Pa. | 1 |
| Oct. 29, | Shaft No. 3, S. Wilkes-Barre, Pa. | 6 |
| Dec. 26, | Princeton, Ind. | 7 |
| 1897–Jan. 4, | Alderson, Ind. Ter. | 5 |
| Feb. 12, | Stanton, Pa. | 1 |
| March 3, | Middle Creek, Pa. | 2 |
| March 3, | William A., Pa. | 2 |
| March 9, | Dickson, Pa. | 1 |
| March 22, | Lytle, Pa. | 1 |
| March 23, | W. Penn, Pa. | 2 |
| March 28, | Jermyn No. 1, Pa. | 5 |
| April 16, | Monarch Coal Co., Ky. | 2 |
| May 5, | Buttonwood, Pa. | 1 |
| May 7, | Auchincloss, Pa. | 2 |
| June 12, | Phœnix Shaft, Duryea, Pa. | 1 |
| June 24, | Marion, Pa. | 2 |
| July 20, | Gowen, Ind. Ter. | 1 |
| July 26, | St. Nicholas, Pa. | 1 |
| Aug. 25, | Shaft No. 1, Edwardsdale, Pa. | 1 |
| Sept. 3, | Sunshine, Colo. | 12 |
| Sept. 23, | Packer No. 4, Pa. | 3 |
| Sept. 23, | Shenandoah, Pa. | 1 |
| Sept. 27, | Alderson, Ind. Ter. | 1 |
| Oct. 2, | Pettibone Shaft, Pa | 2 |
| Oct. 5, | Parrish, Pa. | 3 |
| Oct. 18, | Harry E. Shaft, Pa. | 1 |

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## *Mine Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1897–Oct. 14, | Kaska William, Pa | 4 |
| Nov. 16, | Parrish, Pa | 1 |
| Dec. 6, | Clear Spring Shaft, Pa | 1 |
| Dec. 27, | Bellevue Shaft, Pa | 1 |
| 1898–Jan. 5, | Avondale, Pa | 3 |
| Jan. 17, | Shaft No. 4, Plymouth, Pa. | 2 |
| Jan. 28, | Alden, Pa | 1 |
| Feb. 9, | Dodson, Pa | 2 |
| Feb. 12, | Gowen, I. T | 1 |
| Feb. 14, | Woodward No. 1, Pa | 1 |
| Feb. 28, | Alderson, I. T | 2 |
| Feb. 28, | Rockwood Mine, Tenn | 1 |
| March 19, | T. C. I. & R. R. No. 2, Ala. | 6 |
| March 22, | Manown, Pa | 1 |
| May 26, | Hargrove, Ala | 2 |
| June 15, | Henry E. Shaft, Pa | 1 |
| June 16, | Parrish, Pa | 1 |
| July 5, | Glen Lyon, Pa | 1 |
| July 5, | Seneca Slope, Pa | 1 |
| Aug. 3, | Williamston, Pa | 2 |
| Aug. 15, | Newcastle, Ala | 2 |
| Aug. 19, | Hartshorne, I. T | 1 |
| Aug. 19, | Maxwell, Pa | 1 |
| Aug. 26, | Shaft No. 1, Nanticoke, Pa. | 1 |
| Sept. 15, | Shaft No. 2, Nanticoke, Pa. | 3 |
| Sept. 21, | Cameron, Pa | 1 |
| Sept. 23, | Wilburton, I. T | 1 |
| Sept. 23, | Brownsville, Pa | 8 |
| Oct. 17, | Forty Fort, Pa | 1 |
| Nov. 8, | Schovley Shaft, Pa | 1 |
| Nov. 26, | Preston No. 3, Pa | 1 |
| Dec. 6, | No. 4 Shaft, Pa | 1 |
| Dec. 7, | Luke Fidler, Pa | 3 |
| Dec. 15, | Rush Run, W. Va | 1 |
| Dec. 19, | Hollenback, Pa | 1 |
| Dec. 23, | Exeter, Pa | 1 |
| Dec. 24, | Luke Fidler, Pa | 2 |
| 1899–Jan. 17, | Shaft No. 5, Plymouth, Pa. | 1 |
| Jan. 25, | Gilberton, Pa | 2 |
| Jan. 28, | Shaft No. 1, Edwardsdale, Pa | 2 |
| Feb. 21, | Blocton No. 2, Ala | 5 |
| March 1, | Shaft No. 5, Plymouth, Pa. | 1 |
| March 9, | Mahanoy City, Pa | 8 |
| March 13, | Turkey Run, Pa | 1 |
| April 21, | Madrid, New Mexico | 5 |
| April 22, | Ola, I. T | 2 |
| April 27, | Woodward No. 2, Pa | 1 |
| May 3, | Maple Hill, Pa | 1 |
| May 7, | Morgan Slope, Wash | 1 |
| May 23, | Cumnock, N. C | 23 |
| June 16, | Dominion Colliery No. 4, N. S | 11 |
| June 22, | Maxwell, Pa | 1 |
| July 24, | Shaft No. 4, Edwardsdale, Pa | 1 |
| July 24, | Grindstone, Pa | 5 |
| Aug. 17, | Shenandoah City, Pa | 1 |
| Aug. 21, | Red Star, W. Va | 1 |
| Sept. 1, | Gaston, W. Va | 1 |
| Oct. 9, | Elm Grove, W. Va | 1 |
| Oct. 23, | Nottingham, Pa | 1 |
| 1899–Oct. 23, | Stanton, Pa | 1 |
| Oct. 26, | Alderson, I. T | 1 |
| Dec. 9, | Carbonado, Wash | 31 |
| Dec. 15, | Ola, I. T | 1 |
| Dec. 23, | Summer, Pa. (Brasnell) | 19 |
| 1900–Jan. 15, | No. 3, So. Wilkes-Barre, Pa. | 2 |
| Jan. 22, | Marvine, Pa | 1 |
| Feb. 19, | Bottom Creek, W. Va | 1 |
| Feb. 21, | Black Diamond, No. 14, Wash | 1 |
| March 6, | Red Ash, W. Va | 46 |
| March 9, | Pancoast, Pa | 1 |
| March 21, | Richmond No. 3, Pa | 1 |
| March 26, | Packer No. 3, Pa | 2 |
| March 26, | Hartshorne, I. T | 1 |
| May 1, | Scofield, Utah | 200 |
| June 4, | Warrior Penn, Pa | 1 |
| June 7, | Royal Oak, Pa | 1 |
| June 8, | Phœnix No. 2, Ohio | 3 |
| June 12, | Williamston, Pa | 2 |
| July 9, | Maxwell No. 20, Pa | 1 |
| July 31, | No. 5, So. Wilkes-Barre, Pa. | 2 |
| Aug. 11, | No. 1 Shaft, Lackawanna, Pa | 1 |
| Aug. 27, | McAlester, I. T | 2 |
| Sept. 6, | Stanton, Pa | 1 |
| Oct. 3, | Shaft No. 1, Kingston, Pa | 4 |
| Oct. 10, | Panama, I. T | 2 |
| Oct. 22, | Witterville, I. T | 1 |
| Nov. 1, | Marvine, Pa | 2 |
| Nov. 2, | Berryburg, W. Va | 15 |
| Nov. 5, | Hartshorne, I. T | 1 |
| Nov. 7, | Alderson, I. T | 1 |
| Nov. 9, | Buck Mountain, Pa | 7 |
| Nov. 30, | Maxwell, Pa | 1 |
| Dec. 5, | Luke Fidler, Pa | 2 |
| Dec. 6, | Krebs, I. T | 1 |
| Dec. 14, | McAlester, I. T | 1 |
| Dec. 18, | Henry Clay, Pa | 1 |
| 1901–Jan. 3, | Laurel Run Slope, Pa | 3 |
| Jan. 3, | Delaware Shaft, Pa | 1 |
| Jan. 3, | No. 2 Colliery, Plymouth, Pa | 1 |
| Jan. 5, | Cleveland No. 4 Mine, Iowa | 2 |
| Jan. 27, | Coal City, Ill | 1 |
| Feb. 15, | Union, B. C | 64 |
| Feb. 28, | Krebs, I. T | 3 |
| March 5, | Stanton, Pa | 2 |
| March 25, | Gates, Pa | 4 |
| March 26, | Wanamie No. 18, Pa | 1 |
| April 9, | Wanamie No. 18, Pa | 1 |
| April 29, | Alderson, I. T | 6 |
| May 15, | Chatham, W. Va | 10 |
| May 20, | Richland, Tenn | 20 |
| May 27, | No. 4 Shaft, Kingston, Pa. | 1 |
| June 10, | Port Royal, Pa | 19 |
| July 2, | Lookout Shaft, Pa | 2 |
| July 29, | Wilburton, I. T | 1 |
| Aug. 16, | Lance No. 11, Pa | 1 |
| Aug. 29, | Dow, I. T | 1 |
| Aug. 30, | Sugar Notch No. 9, Pa | 1 |
| Sept. 11, | Stevens Slope, Pa | 1 |

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## *Mine Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1901–Sept. 16, | Spring Gulch, Colo. | 6 |
| Oct. 2, | Harry E. Shaft, Pa. | 1 |
| Oct. 7, | Hartshorne, I. T. | 2 |
| Oct. 25, | Buttonwood, Pa. | 6 |
| Oct. 30, | Twin No. 1 Shaft, Luzerne Co., Pa. | 1 |
| Oct. 31, | Parrish, Pa. | 2 |
| Nov. 11, | Labelle, Ohio. | 3 |
| Dec. 3, | Grindstone, Pa. | 3 |
| Dec. 20, | Ola, I. T. | 4 |
| 1902–Jan. 4, | Cameron, Pa. | 2 |
| Jan. 23, | Twin Shaft, No. 2, Luzerne, Pa. | 3 |
| Jan. 24, | Lost Creek, Iowa. | 20 |
| Jan. 25, | No. 4 Shaft, Luzerne, Pa. | 1 |
| Jan. 31, | Henry Clay, Pa. | 1 |
| Feb. 18, | Hocking Mine, Iowa. | 2 |
| Feb. 26, | Haileyville, I. T. | 2 |
| March 6, | Catsburg, Pa. | 5 |
| March 31, | Dayton, Tenn. | 16 |
| April 8, | Wyoming Shaft, Pa. | 1 |
| May 19, | Coal Creek (Fraterville), Tenn. | 184 |
| May 20, | Colgate, I. T. | 2 |
| May 22, | Fernie, B. C. | 125 |
| July 10, | Rolling Mill Mine, Johnstown, Pa. | 112 |
| July 27, | McCurtain, I. T. | 2 |
| Aug. 7, | Bowen, Colo. | 13 |
| Sept. 2, | Mt. Lookout, Pa. | 1 |
| Sept. 16, | Algoma, W. Va. | 17 |
| Sept. 22, | Stafford, W. Va. | 6 |
| Oct. 1, | Lawson, Wash. | 11 |
| Nov. 2, | Locust Run, Ohio. | 3 |
| Nov. 29, | Luke Fidler, Pa. | 7 |
| Dec. 8, | Wilburton, I. T. | 1 |
| Dec. 27, | Little Redstone, Pa. | 4 |
| 1903–Jan. 6, | Alderson, I. T. | 1 |
| Jan. 13, | Bliss, Pa. | 1 |
| Jan. 14, | Packer No. 4, Pa. | 2 |
| Jan. 14, | Taylor Mine, Pa. | 1 |
| Jan. 10, | Hartshorne, I. T. | 1 |
| Feb. 5, | Old Forge No. 1, Pa. | 1 |
| Feb. 25, | Mahanoy City, Pa. | 1 |
| March 12, | Wilburton, I. T. | 2 |
| March 13, | Livingston, Ill. | 3 |
| March 14, | Warrior Run, Pa. | 1 |
| March 15, | Cardiff, Ill. | 5 |
| April 13, | Carbon, I. T. | 6 |
| April 16, | Carbon, I. T. | 3 |
| May 12, | Ellangowan, Pa. | 1 |
| May 18, | Wilburton, Pa. | 1 |
| May 21, | Poos Mine, W. Va. | 1 |
| May 21, | No. 14 Shaft, Luzerne Co., Pa. | 1 |
| May 29, | Bliss, Pa. | 1 |
| June 17, | Warrior Run, Pa. | 2 |
| June 19, | Blossburg, N. Mex. | 5 |
| June 22, | Warrior Run, Pa. | 1 |
| June 22, | Lehigh C. & N. Co. No. 8, Pa. | 1 |
| June 29, | Clear Spring, Pa. | 1 |
| 1903–June 30, | Hanna, Wyo. | 169 |
| June 30, | Colgate, I. T. | 1 |
| July 2, | Storrs No. 1, Pa. | 3 |
| July 15, | Union, B. C. | 16 |
| Sept. 30, | Avondale, Pa. | 1 |
| Oct. 6, | So. Wilkes-Barre, Pa. | 1 |
| Oct. 9, | Haileyville, I. T. | 1 |
| Nov. 18, | Lehigh C. & N. Co. No. 10, Pa. | 1 |
| Nov. 21, | Ferguson, Pa. | 17 |
| Nov. 24, | Century, W. Va. | 3 |
| Nov. 24, | Henry, W. Va. | 1 |
| Nov. 28, | Stanton, Pa. | 1 |
| Dec. 3, | No. 14 Shaft, Luzerne Co., Pa. | 4 |
| Dec. 12, | Wilburton, I. T. | 1 |
| Dec. 18, | Big Mountain, Pa. | 1 |
| Dec. —, | Flat Top, Ala. | 5 |
| Dec. 21, | Nottingham, Pa. | 1 |
| Dec. 23, | Exeter, Pa. | 1 |
| Dec. 30, | Sutter, Pa. | 2 |
| 1904–Jan. 5, | McCurtain, I. T. | 2 |
| Jan. 5, | Auchincloss, Pa. | 1 |
| Jan. 8, | Crows Nest, B. C. | 7 |
| Jan. 15, | Corbin, Pa. | 3 |
| Jan. 25, | Harwick, Pa. | 178 |
| Jan. 25, | Foster Mine, Iowa. | 2 |
| Feb. 6, | Storrs, Pa. | 2 |
| Feb. 8, | Buck, I. T. | 2 |
| Feb. 12, | McAlester, I. T. | 1 |
| Feb. 24, | Hillman Slope, Pa. | 1 |
| March 9, | Tunnel Ridge, Pa. | 2 |
| March 14, | Tunnel Ridge, Pa. | 1 |
| March 15, | Henry No. 33, W. Va. | 2 |
| April 6, | No. 14 Shaft, Luzerne Co., Pa. | 1 |
| April 17, | Nixon, Pa. | 1 |
| April 20, | Stearns, Ky. | 5 |
| June 20, | Maxwell, Pa. | 1 |
| June 25, | Kangley, Ill. | 1 |
| Aug. 5, | Pittsburg & Eastern No. 2, Pa. | 1 |
| Aug. 19, | Short Mt., Pa. | 1 |
| Aug. 20, | Burnside, Pa. | 1 |
| Aug. 23, | Buck Ridge, Pa. | 1 |
| Sept. 1, | Henry Shaft, Pa. | 1 |
| Sept. 17, | Primrose, Pa. | 3 |
| Sept. 22, | No. 14 Shaft, Luzerne Co., Pa. | 1 |
| Sept. 28, | Mt. Jessop, Pa. | 1 |
| Sept. 30, | Sutter, I. T. | 1 |
| Oct. 26, | Oxford, Lackawanna Co., Pa. | 4 |
| Oct. 28, | Tercio, Colo. | 19 |
| Nov. 10, | Krebs, I. T. | 2 |
| Nov. 15, | Adkins, I. T. | 1 |
| Nov. 18, | Crows Nest, B. C. | 14 |
| Nov. 25, | Wilburton, I. T. | 2 |
| Dec. 7, | Burnett, Wash. | 17 |
| Dec. 8, | Penna. Mine, Northumberland Co., Pa. | 1 |
| Dec. 16, | McAlester, I. T. | 1 |

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## *Mine Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1904–Dec. 19, | Oneida No. 1, Pa. | 2 |
| Dec. 29, | Alderson, I. T. | 1 |
| Dec. 29, | McCurtain, I. T. | 1 |
| 1905–Jan. 20, | Colgate, I. T. | 1 |
| Feb. 14, | Williamstown, Pa. | 1 |
| Feb. 17, | Shenandoah City, Pa. | 1 |
| Feb. 20, | Virginia City, Ala. | 111 |
| Feb. 25, | McAlester, I. T. | 2 |
| Feb. 26, | Wilcoe, W. Va. | 7 |
| March 5, | Burton No. 13 Mine, Iowa. | 1 |
| March 11, | Park Place, Pa. | 1 |
| March 18, | Red Ash, W. Va. | 24 |
| March 20, | Kingston, Pa | 1 |
| March 22, | Princeton, Ind. | 9 |
| April 3, | Ziegler, Ill. | 53 |
| April 3, | West End, Pa. | 1 |
| April 20, | Mt. Pleasant, Pa. | 1 |
| April 20, | Cabin Creek, W. Va. | 6 |
| April 27, | Maple Hill, Pa. | 2 |
| April 27, | Dubois, Pa. | 13 |
| April 30, | Wilburton, I. T. | 13 |
| May 8, | Seneca, Pa. | 1 |
| May 15, | Duquoin, Ill. | 1 |
| June 2, | Phœnix Mine, W. Va. | 1 |
| June 19, | Hartshorne, I. T. | 1 |
| June 26, | Harry E. Mine, Pa. | 1 |
| July 5, | Vivian, W. Va. | 5 |
| July 6, | No. 11 Shaft, Pa. | 1 |
| July 12, | Buck, I. T. | 2 |
| July 16, | Lutie, I. T. | 1 |
| July 16, | Carlisle Mine, Fayette Co., W. Va. | 2 |
| July 18, | Lance, Luzerne Co., Pa. | 1 |
| July 20, | Victoria No. 2 Mine, W. Va. | 1 |
| July 24, | Mahanoy City, Pa. | 1 |
| Aug. 3, | Midvale Slope, Luzerne Co., Pa. | 1 |
| Aug. 19, | No. 6 Shaft, Luzerne Co., Pa. | 1 |
| Aug. 25, | Phœnix Park, Pa. | 1 |
| Sept. 12, | Boston Run, Pa. | 1 |
| Oct. 2, | Brookside, Pa. | 1 |
| Oct. 5, | No. 14 Shaft, Pa. | 1 |
| Oct. 15, | Johnsville, I. T. | 2 |
| Oct. 19, | Silver Creek, Pa. | 1 |
| Oct. 29, | Haselkirk, Pa. | 5 |
| Nov. 4, | Vivian, W. Va. | 7 |
| Nov. 10, | Buck Run, Pa. | 1 |
| Nov. 15, | Penn, Pa. | 1 |
| Nov. 15, | Bentleyville, Pa. | 6 |
| Nov. 16, | Hughes, I. T. | 1 |
| Nov. 22, | Wilburton, I. T. | 2 |
| Nov. 23, | Stevens, Pa. | 1 |
| Dec. 1, | Diamondville, Wyo. | 18 |
| Dec. 7, | Maxwell, Pa. | 1 |
| Dec. 12, | Buck Mt., W. Va. | 1 |
| Dec. 13, | Luke Fidler, Pa. | 4 |
| Dec. 15, | Harry E. Mine, Pa. | 1 |
| Dec. 21, | Mt. Lookout, Pa. | 1 |
| 1906–Jan. 2, | Wanamie No. 18, Pa. | 1 |
| Jan. 4, | Coaldale, W. Va. | 22 |
| Jan. 15, | No. 14 Tunnel, Luzerne Co., Pa. | 2 |
| 1906–Jan. 18, | Detroit, W. Va. | 18 |
| Jan. 24, | Wittenville, I. T. | 14 |
| Jan. 29, | Sherman, Ill. | 2 |
| Feb. 1, | Bald Knob, W. Va. | 3 |
| Feb. 8, | Parral, W. Va. | 27 |
| Feb. 19, | Maitland, Colo. | 14 |
| Feb. 27, | Piper, Ala. | 12 |
| March 3, | Midvale Slope, Luzerne Co., Pa. | 1 |
| March 4, | Packer No. 3, Pa. | 1 |
| March 21, | L. C. & N. Co. No. 10, Pa. | 1 |
| March 22, | Century, W. Va. | 23 |
| April 10, | Penn No. 10, Cambria Co., Pa. | 1 |
| April 22, | Cautro, Colo. | 19 |
| May 7, | Storrs, Pa. | 1 |
| May 14, | Parrish, Pa. | 2 |
| June 9, | Alaska, Pa. | 1 |
| June 11, | Nesquhoning No. 1, Pa. | 1 |
| June 12, | Richards, Pa. | 1 |
| June 18, | Piney No. 1, W. Va. | 1 |
| June 19, | Avondale, Pa. | 1 |
| June 21, | Jamison No. 3, Pa. | 1 |
| July 2, | Keystone, W. Va. | 4 |
| July 16, | Cameron, Pa. | 1 |
| July 19, | Huger, W. Va. | 4 |
| July 20, | Nottingham, Pa. | 3 |
| Aug. 1, | Warrior Run, Pa. | 1 |
| Aug. 6, | North Shaft No. 7, Pa. | 6 |
| Aug. 21, | Vulcan, Pa. | 2 |
| Aug. 23, | Sugar Notch, Pa. | 2 |
| Sept. 6, | Johnson City, Ill. | 1 |
| Oct. 2, | Muhlenburg Co., Ky. | 1 |
| Oct. 3, | Pocahontas, Va. | 36 |
| Oct. 5, | Blossburg, N. Mex. | 10 |
| Oct. 12, | Exeter, Pa. | 1 |
| Oct. 18, | Nottingham, Pa. | 1 |
| Oct. 19, | Prospect Shaft, Luzerne Co., Pa. | 1 |
| Oct. 24, | Johnstown, Pa. | 7 |
| Oct. 25, | Vulcan, Pa. | 1 |
| Oct. 27, | Herrin, Ill. | 2 |
| Nov. 7, | Midvalley, Pa. | 1 |
| Nov. 13, | Central, Pa. | 1 |
| Nov. 21, | Central, Pa. | 2 |
| Nov. 26, | Palmer, Wash. | 1 |
| Dec. 6, | Woodward, Pa. | 1 |
| Dec. 6, | Bettonwood, Pa. | 4 |
| 1907–Jan. 3, | Shenandoah City, Pa. | 1 |
| Jan. 17, | Pine Ridge, Pa. | 1 |
| Jan. 18, | Harrisburg, Ill. | 1 |
| Jan. 23, | Primero, Colo. | 24 |
| Jan. 26, | Penco, W. Va. | 12 |
| Jan. 29, | Fayetteville, W. Va. | 85 |
| Jan. 29, | Nottingham, Pa. | 1 |
| Feb. 4, | Thomas No. 25, V. Wa. | 25 |
| Feb. 12, | Lansford No. 4, Pa. | 1 |
| March 2, | Holden, Pa. | 7 |
| March 2, | Woodward, Pa. | 2 |
| March 5, | Tunnel Ridge, Pa. | 1 |
| March 16, | Tacoma, Va. | 11 |
| March 16, | Howard, Pa. | 1 |

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## *Mine Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1907–March 19, | Nottingham, Pa. | 1 |
| March 23, | Barnum No 2, Pa. | 1 |
| March 23, | Sugar Creek, Ohio. | 3 |
| April 6, | Pettibone, Pa. | 1 |
| April 8, | Locust Spring, Pa. | 2 |
| April 11, | Consolidation Coal Co., Iowa. | 1 |
| April 24, | Midway, Colo. | 1 |
| April 26, | Black Diamond, Wash. | 7 |
| May 1, | Whipple, W. Va. | 18 |
| May 15, | No. 4 Shaft, Luserne Co., Pa. | 1 |
| June 17, | Reliance, Pa. | 1 |
| June 18, | Johnson No. 1, Pa. | 7 |
| July 17, | North Mahanoy, Pa. | 1 |
| July 20, | No. 14 Shaft, Pa. | 2 |
| Aug. 1, | Penn, Pa. | 1 |
| Aug. 20, | Storrs No. 3, Pa. | 1 |
| Aug. 23, | Franklin, Washington. | 3 |
| Aug. 29, | Stanton No. 7, Pa. | 1 |
| Sept. 21, | So. Wilkes-Barre, Pa. | 4 |
| Sept. 28, | Mineral Spring Shaft, Pa. | 1 |
| Sept. 30, | Storrs No. 1, Pa. | 1 |
| Sept. 7, | W. Frankfort, Ill. | 4 |
| Nov. 12, | Harry E. Mine, Pa. | 1 |
| Nov. 14, | So. Wilkes-Barre, Pa. | 1 |
| Nov. 27, | Stanton No. 7, Pa. | 1 |
| Dec. 1, | Naomi, Pa. | 34 |
| Dec. 6, | Monongah, W. Va. | 358 |
| Dec. 16, | Yolande, Ala. | 16 |
| Dec. 18, | Darr, Pa. | 239 |
| Dec. 20, | Pictou Mine, Colo. | 1 |
| Dec. 23, | Buck, Okla. | 1 |
| Dec. 31, | Carthage, New Mexico. | 11 |
| 1908–Jan. 9, | Scott, Pa. | 3 |
| Jan. 11, | Harrisburg, Ill. | 1 |
| Jan. 13, | Conyngham, Pa. | 1 |
| Jan. 13, | Woodward, Pa. | 2 |
| Jan. 25, | Stonington, Ill. | 1 |
| Jan. 30, | Bachman, W. Va. | 9 |
| Jan. 31, | No. 14 Shaft, Pa. | 1 |
| Feb. 7, | Port Hood, N. S. | 10 |
| Feb. 10, | Baltimore No. 5, Pa. | 2 |
| Feb. 10, | South Carrollton, Ky. | 9 |
| Feb. 12, | Eldorado, Ill. | 1 |
| Feb. 15, | Chant, Okla. | 2 |
| Feb. 17, | Eagle Hill, Pa. | 1 |
| Feb. 24, | No. 14 Shaft, Pa. | 1 |
| Feb. 27, | Roseta, Mexico. | 83 |
| March 15, | Standard Mine, Colo. | 1 |
| March 24, | No. 6 Shaft, Luserne Co., Pa. | 1 |
| March 28, | Hanna, Wyo. | 59 |
| March 31, | Oakdale Mine, Colo. | 1 |
| March 31, | Sugar Notch No. 9, Pa. | 1 |
| April 8, | Wm. Penn, Pa. | 2 |
| April 14, | Champion Mine, Colo. | 1 |
| April 18, | Woodward, Pa. | 2 |
| April 23, | Ellsworth No. 1, Pa. | 1 |
| May 11, | No. 7 Colliery, Luserne Co., Pa. | 4 |
| May 12, | Mt. Lookout, Pa. | 12 |
| 1908–May 23, | Otto, Pa. | 2 |
| May 27, | Salineville, Ohio. | 3 |
| June 19, | Ellsworth No. 1, Pa. | 3 |
| July 8, | Lance No. 11, Pa. | 1 |
| Aug. 12, | Agnes, Washington Co., Pa. | 1 |
| Aug. 28, | Packer No. 2, Pa. | 2 |
| Aug. 31, | Brewer, Okla. | 1 |
| Sept. 1, | Nonpareil Mine, Colo. | 1 |
| Sept. 3, | Shenandoah City, Pa. | 2 |
| Sept. 8, | Coaldale No. 1, Pa. | 1 |
| Sept. 17, | Sugar Notch No. 9, Pa. | 1 |
| Sept. 26, | Hartshorne, Okla. | 1 |
| Oct. 7, | Phœnix Park, Pa. | 2 |
| Oct. 8, | No. 14 Shaft, Luserne Co., Pa. | 1 |
| Oct. 14, | Carbonado, Wash. | 1 |
| Oct. 22, | Packer No. 3, Pa. | 1 |
| Oct. 29, | Short Creek, Ala. | 6 |
| Oct. 31, | Wilburton, Okla. | 1 |
| Nov. 5, | Rend City, Ill. | 4 |
| Nov. 6, | Buck Mt., Pa. | 1 |
| Nov. 9, | DeRush Mine, Colo. | 1 |
| Nov. 9, | Oxford, Pa. | 1 |
| Nov. 10, | Maxwell No. 20, Pa. | 1 |
| Nov. 13, | Woodward, Pa. | 1 |
| Nov. 18, | Shenandoah City, Pa. | 1 |
| Nov. 19, | Rend City, Ill. | 3 |
| Nov. 23, | Black Diamond, Pa. | 1 |
| Nov. 28, | Marianna, Pa. | 154 |
| Dec. 29, | Lick Branch, W. Va. | 54 |
| 1909–Jan. 10, | Ziegler, Ill. | 26 |
| Jan. 12, | Lick Branch, W. Va. | 69 |
| Jan. 19, | Chancellor, Colo. | 6 |
| Jan. 25, | Boswell, Pa. | 5 |
| Jan. 29, | Sewickley, Pa. | 2 |
| Jan. 29, | N. W. I. & S. Co., Wash. | 2 |
| Feb. 2, | Short Creek, Ala. | 18 |
| Feb. 9, | Ziegler, Ill. | 3 |
| Feb. 16, | West Frankfort, Ill. | 4 |
| March 2, | No. 14 Shaft, Luserne Co., Pa. | 8 |
| March 20, | Evansville, Ind. | 6 |
| March 23, | Sliver Creek, Pa. | 1 |
| March 29, | Coahuila, Mexico. | 35 |
| March 31, | Beury, W. Va. | 6 |
| April 7, | Coast Coal Co., Wash. | 1 |
| April 9, | Windber, Pa. | 7 |
| April 13, | Superior Mine, Ind. | 20 |
| April 15, | Air Shaft No. 3, Marion Co., W. Va. | 3 |
| May 1, | No. 5 Mine, Luserne Co., Pa. | 1 |
| May 7, | No. 6 Mine, Luserne Co., Pa. | 1 |
| May 22, | Gilberton, Pa. | 1 |
| June 2, | Richards, Pa. | 1 |
| June 3, | Richards, Pa. | 1 |
| June 3, | Lytle, Pa. | 1 |
| June 16, | No. 8 Colliery, Schuylkill Co., Pa. | 1 |
| June 18, | Spring Brook, Pa. | 1 |
| June 23, | Wehrum, Pa. | 21 |

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## *Mine Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1909–June 23, | Nottingham, Pa. | 2 |
| July 6, | Tollerville, Colo. | 9 |
| July 2, | Grayson, Ill. | 2 |
| July 15, | Newcastle, Pa. | 1 |
| July 19, | Revere, Pa. | 1 |
| Aug. 18, | Richards, Pa. | 1 |
| Aug. 30, | No. 8 Colliery, Schuylkill Co., Pa. | 1 |
| Sept. 10, | Nottingham, Pa. | 1 |
| Sept. 16, | Parral, W. Va. | 1 |
| Sept. 29, | W. Eureka No. 7, Pa. | 1 |
| Oct. 3, | Roslyn, Wash. | 10 |
| Oct. 5, | Nanaimo, B. C. | 30 |
| Oct. 11, | Sesser, Ill. | 1 |
| Oct. 21, | Hartshorne, Okla. | 10 |
| Oct. 31, | Franklin No. 2, Cambria Co., Pa. | 13 |
| Nov. 2, | Maple Hill, Pa | 1 |
| Nov. 5, | Prospect, Pa | 1 |
| Nov. 17, | Lance No. 11, Luserne Co., Pa. | 1 |
| Nov. 17, | Reliance, Pa. | 1 |
| Nov. 21, | West Frankfort, Ill. | 1 |
| Nov. 29, | Johnson City, Ill. | 1 |
| Dec. 11, | Clay, Ky. | 7 |
| Dec. 23, | Herrin, Ill. | 8 |
| Dec. 27, | Centralia, Ill. | 4 |
| 1910–Jan. 11, | Nottingham, Pa. | 7 |
| Jan. 17, | Ernest No. 2, Pa. | 5 |
| Jan. 20, | Woodward, Pa. | 1 |
| Jan. 31, | Primero, Colo. | 75 |
| Feb. 1, | Browder, Ky. | 34 |
| Feb. 2, | Palau Mine, Mexico | 57 |
| Feb. 3, | Oxford, Pa. | 1 |
| Feb. 5, | Ernest No. 2, Pa. | 6 |
| Feb. 8, | Stearns, Ky. | 6 |
| Feb. 21, | Lytle, Pa. | 1 |
| Feb. 22, | Pettibone, Pa. | 2 |
| Feb. 26, | Christopher, Ill. | 1 |
| March 2, | Parrish, Pa. | 1 |
| March 12, | So. Wilkes-Barre, Pa. | 7 |
| March 21, | Lansford, Pa. | 1 |
| March 24, | Allison, Pa. | 2 |
| March 26, | Cambria, Okla. | 4 |
| March 31, | Wilburton, Okla. | 6 |
| March 31, | Maxwell No. 20, Pa. | 1 |
| April 20, | Mulga, Ala. | 40 |
| April 21, | Amsterdam, Ohio. | 15 |
| May 5, | Palos, Ala. | 83 |
| May 28, | St. Nicholas, Pa. | 1 |
| June 15, | Eddy Creek, Pa. | 2 |
| June 15, | So. Wilkes-Barre No. 5, Pa. | 1 |
| June 23, | Woodward, Pa. | 1 |
| July 1, | Kaska William, Pa. | 1 |
| July 2, | Mt. Lookout, Pa. | 1 |
| July 6, | Wanamie, Pa. | 1 |
| July 11, | Ewen, Pa. | 1 |
| Sept. 9, | Ewen, Pa. | 1 |
| Sept. 30, | Palau No. 2 Mine, Menor, Mex. | 78 |
| Oct. 3, | Roslyn, Wash. | 10 |
| Oct. 8, | Starkville, Colo. | 56 |
| 1910–Oct. 13, | Bliss, Luserne Co., Pa. | 2 |
| Nov. 4, | Yolande, Ala. | 5 |
| Nov. 6, | Black Diamond, Wash. | 16 |
| Nov. 8, | Delagua, Colo. | 79 |
| Nov. 11, | Panama, Ill. | 6 |
| Nov. 25, | Providence, Ky. | 10 |
| Nov. 14, | Greeno, Va. | 11 |
| Nov. 14, | Middleton Mine, W. Va. | 2 |
| Nov. 26, | Grayson, Ill. | 2 |
| Nov. 28, | Antlers, Okla. | 13 |
| Dec. 9, | Bellevue Mine, B. C. | 31 |
| Dec. 13, | Ravensdale, Wash. | 2 |
| Dec. 18, | Harrisburg, Ill. | 1 |
| Dec. 19, | Consolidation No. 47, W. Va. | 2 |
| Dec. 23, | Herrin, Ill. | 8 |
| Dec. 27, | Centralia, Ill. | 4 |
| Dec. 29, | Wick Haven, Pa. | 1 |
| 1911–Jan. 3, | Sydney No. 3 Mine, Nova Scotia. | 8 |
| Jan. 20, | Carbon Hill, Va. | 5 |
| Jan. 25, | Pittston, Pa. | 6 |
| Feb. 9, | Cokedale Mine, Colo. | 17 |
| March 18, | West Mineral, Kans. | 5 |
| April 8, | Banner Mine, Ala. | 128 |
| April 14, | Herrin, Ill. | 1 |
| April 24, | Elk Garden, W. Va. | 23 |
| May 27, | Cameron Mine, Pa. | 5 |
| July 15, | Sykesville, Pa. | 21 |
| Aug. 1, | Standard Mine, W. Va. | 4 |
| Oct. 23, | Harrisburg, Ill. | 9 |
| Nov. 9, | Adrian Mine, Pa. | 8 |
| Nov. 18, | Bottom Creek Mine, W. Va. | 18 |
| Dec. 9, | Briceville, Tenn. | 84 |
| 1912–Jan. 9, | Plymouth, Pa. | 6 |
| Jan. 13, | Windsor, Mo. | 2 |
| Jan. 15, | Regal Mine, Iowa | 2 |
| Jan. 17, | Central City, Ky. | 5 |
| Jan. 20, | Susie, Wyo. | 5 |
| March 7, | Merritt, B. C. | 7 |
| March 10, | Johnson City, Ill. | 1 |
| March 20, | McCurtain, Okla. | 74 |
| March 26, | Jed, W. Va. | 83 |
| April 15, | Tacoma, Wash. | 1 |
| April 21, | Madisonville, Ky. | 5 |
| April 23, | Panama, Ill. | 2 |
| June 18, | Hastings, Colo. | 12 |
| July 11, | Moundsville, W. Va. | 8 |
| July 16, | Carbon Hill, Va. | 8 |
| July 17, | So. Wilkes-Barre, Pa. | 4 |
| July 24, | D. & H. No. 2, Pa. | 2 |
| Aug. 13, | Abernant, Ala. | 18 |
| Aug. 30, | Piedmont, Colo. | 2 |
| Sept. 16, | Coral, Pa. | 1 |
| Oct. 1, | South Canon, Colo. | 2 |
| Oct. —, | Oak Hill Mine, Ind. | 2 |
| Oct. 25, | Mayer Coal Co., Iowa | 2 |
| Nov. 8, | Simpson, Colo. | 1 |
| Nov. 16, | Clifton Mine, Ill. | 1 |
| Nov. 22, | Peoria, W. Va. | 2 |
| Dec. 20, | Taylor Mine, Pa. | 2 |

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TABLE II.—*Explosions of Gas and Dust in Coal Mines of North America, by Months, 1839 to 1912, Inc.*

| Month | Explosions Having Less than Five Fatalities | | | | Explosions Having More than Five Fatalities | | | | Total Explosions | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions |
| January | 69 | 109 | 1.58 | 10.53 | 33 | 953 | 28.88 | 14.23 | 102 | 1,062 | 10.41 | 11.50 |
| February | 44 | 68 | 1.50 | 6.72 | 28 | 800 | 28.57 | 12.07 | 72 | 868 | 12.06 | 8.12 |
| March | 57 | 83 | 1.46 | 8.70 | 33 | 781 | 23.67 | 14.23 | 90 | 864 | 9.60 | 10.14 |
| April | 44 | 64 | 1.45 | 6.72 | 22 | 428 | 19.45 | 9.48 | 66 | 492 | 7.45 | 7.44 |
| May | 47 | 78 | 1.67 | 7.18 | 18 | 833 | 46.28 | 7.76 | 65 | 911 | 14.01 | 7.33 |
| June | 54 | 74 | 1.37 | 8.24 | 11 | 271 | 24.64 | 4.74 | 65 | 345 | 5.31 | 7.33 |
| July | 51 | 79 | 1.55 | 7.79 | 10 | 205 | 20.50 | 4.31 | 61 | 284 | 4.66 | 6.88 |
| August | 47 | 64 | 1.36 | 7.18 | 5 | 49 | 9.80 | 2.15 | 52 | 113 | 2.17 | 5.86 |
| September | 50 | 70 | 1.40 | 7.63 | 8 | 140 | 17.50 | 3.45 | 58 | 210 | 3.62 | 6.54 |
| October | 65 | 96 | 1.48 | 9.92 | 21 | 283 | 13.48 | 9.06 | 86 | 379 | 4.41 | 9.69 |
| November | 63 | 87 | 1.38 | 9.62 | 24 | 496 | 20.67 | 10.34 | 87 | 583 | 6.70 | 9.81 |
| December | 64 | 105 | 1.64 | 9.77 | 19 | 1,011 | 53.21 | 8.18 | 83 | 1,116 | 13.45 | 9.36 |
| Totals | 655 | 977 | | 100.00 | 232 | 6,250 | | 100.00 | 887 | 7,227 | | 100.00 |
| Averages | | | 1.49 | | | | 26.94 | | | | 8.15 | |

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TABLE III.—*Production in Short Tons and Fatal Accidents Caused by Explosions of Gas and Dust in Coal Mines of United States*

| Year | Production | Accidents | Fatalities | Accidents per Million Tons | Fatalities per Million Tons | Accidents in which 5 or more Fatalities Occurred | | | | Mines | Average Production Per Mine |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | Number | Fatalities | Accidents per Million Tons | Fatalities Per Million Tons | | |
| 1820 | 3,450 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1821 | 1,322 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1822 | 58 583 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1823 | 68,563 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1824 | 80,725 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1825 | 117,988 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1826 | 147,914 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1827 | 172,151 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1828 | 195,908 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1829 | 240,086 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1830 | 320,072 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1831 | 337,942 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1832 | 594,050 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1833 | 734,657 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1834 | 600,515 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1835 | 824,854 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1836 | 984,832 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1837 | 1,253,651 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1838 | 1,355,527 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1839 | 1,560,360 | 1 | 40 | 0.64 | 25.6 | 1 | 40 | 0.64 | 25.6 | ..... | ..... |
| 1840 | 2,070,039 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1841 | 2,291,141 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1842 | 2,610,057 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1843 | 3,060,874 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1844 | 3,681,252 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1845 | 4,309,904 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1846 | 4,865,522 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1847 | 5,286,067 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1848 | 5,773,974 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1849 | 6,448,831 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1850 | 7,018,181 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | 510 | 12,539 |
| 1851 | 8,734,525 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1852 | 9,816,664 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1853 | 10,570,288 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1854 | 11,977,102 | 1 | 19 | 0.08 | 1.51 | 1 | 19 | 0.08 | 1.51 | ..... | ..... |
| 1855 | 12,926,673 | 1 | 55 | 0.08 | 4.27 | 1 | 55 | 0.08 | 4.27 | ..... | ..... |
| 1856 | 13,546,925 | .. | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1857 | 13,340,189 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1858 | 13,974,478 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1859 | 15,633,175 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1860 | 14,610,042 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | 622 | 23,045 |
| 1861 | 16,488,012 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1862 | 17,485,835 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1863 | 21,319,062 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1864 | 23,605,123 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1865 | 23,792,173 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1866 | 29,003,583 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1867 | 30,724,422 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1868 | 32,861,960 | .... | ..... | ..... | ..... | .... | .... | ..... | ..... | ..... | ..... |
| 1869 | 32,904,360 | 2 | 2 | 0.06 | 0.06 | .... | .... | ..... | ..... | ..... | ..... |
| 1870 | 33,035,580 | 6 | 6 | 0.18 | 0.18 | .... | .... | ..... | ..... | 1,566 | 20,986 |
| 1871 | 46,885,080 | 13 | 21 | 0.28 | 0.45 | 1 | 5 | 0.02 | 0.10 | ..... | ..... |
| 1872 | 51,453,399 | 11 | 21 | 0.21 | 0.41 | 1 | 7 | 0.02 | 0.14 | ..... | ..... |
| 1873 | 57,602,480 | 10 | 20 | 0.17 | 0.35 | 1 | 10 | 0.02 | 0.18 | ..... | ..... |
| 1874 | 52,605,920 | 14 | 18 | 0.26 | 0.34 | .... | .... | ..... | ..... | ..... | ..... |
| 1875 | 52,348,320 | 6 | 8 | 0.12 | 0.15 | .... | .... | ..... | ..... | ..... | ..... |
| 1876 | 55,280,000 | 16 | 31 | 0.30 | 0.58 | 2 | 14 | 0.04 | 0.26 | ..... | ..... |

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TABLE III.— *Continued.*

| Year | Production | Accidents | Fatalities | Accidents per Million Tons | Fatalities per Million Tons | Accidents in which 5 or more Fatalities Occurred | | | | Mines | Average Production per Mine |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | Number | Fatalities | Accidents per Million Tons | Fatalities per Million Tons | | |
| 1877 | 60,501,760 | 2 | 9 | 0.03 | 0.15 | 1 | 7 | 0.02 | 0.12 | ..... | ..... |
| 1878 | 57,955,600 | 6 | 18 | 0.10 | 0.31 | 2 | 13 | 0.04 | 0.22 | ..... | ..... |
| 1879 | 68,105,799 | 7 | 17 | 0.10 | 0.25 | 2 | 11 | 0.03 | 0.16 | ..... | ..... |
| 1880 | 71,481,570 | 4 | 8 | 0.06 | 0.11 | 1 | 5 | 0.01 | 0.07 | 3,294 | 21,701 |
| 1881 | 85,881,030 | 8 | 56 | 0.09 | 0.65 | 3 | 50 | 0.04 | 0.58 | ..... | ..... |
| 1882 | 103,551,189 | 10 | 49 | 0.10 | 0.47 | 2 | 37 | 0.02 | 0.36 | ..... | ..... |
| 1883 | 115,707,525 | 16 | 36 | 0.15 | 0.31 | 1 | 10 | 0.01 | 0.09 | ..... | ..... |
| 1884 | 120,155,551 | 8 | 220 | 0.07 | 1.83 | 5 | 213 | 0.04 | 1.77 | ..... | ..... |
| 1885 | 111,160,295 | 4 | 17 | 0.04 | 0.16 | 2 | 12 | 0.02 | 0.10 | ..... | ..... |
| 1886 | 113,680,427 | 6 | 77 | 0.05 | 0.68 | 5 | 76 | 0.04 | 0.67 | ..... | ..... |
| 1887 | 130,650,511 | 1 | 23 | 0.03 | 0.18 | 1 | 18 | 0.01 | 0.14 | ..... | ..... |
| 1888 | 148,659,657 | 6 | 90 | 0.04 | 0.61 | 3 | 83 | 0.02 | 0.56 | ..... | ..... |
| 1889 | 141,229,513 | 9 | 18 | 0.06 | 0.13 | ..... | ..... | ..... | ..... | 2,583 | 53,578 |
| 1890 | 157,770,963 | 8 | 54 | 0.05 | 0.34 | 4 | 49 | 0.03 | 0.31 | ..... | ..... |
| 1891 | 168,566,669 | 19 | 156 | 0.12 | 0.93 | 3 | 128 | 0.02 | 0.76 | ..... | ..... |
| 1892 | 179,329,071 | 19 | 164 | 0.11 | 0.91 | 3 | 127 | 0.02 | 0.71 | ..... | ..... |
| 1893 | 182,352,774 | 12 | 67 | 0.07 | 0.37 | 7 | 60 | 0.04 | 0.33 | ..... | ..... |
| 1894 | 170,741,526 | 23 | 40 | 0.14 | 0.23 | 1 | 8 | 0.01 | 0.05 | ..... | ..... |
| 1895 | 193,117,530 | 18 | 200 | 0.09 | 1.04 | 8 | 188 | 0.04 | 0.97 | ..... | ..... |
| 1896 | 191,986,357 | 15 | 95 | 0.08 | 0.50 | 4 | 77 | 0.02 | 0.40 | ..... | ..... |
| 1897 | 200,229,199 | 27 | 60 | 0.14 | 0.30 | 3 | 22 | 0.02 | 0.11 | ..... | ..... |
| 1898 | 219,976,267 | 33 | 58 | 0.16 | 0.26 | 2 | 14 | 0.01 | 0.07 | ..... | ..... |
| 1899 | 253,741,192 | 26 | 118 | 0.10 | 0.47 | 7 | 96 | 0.03 | 0.38 | ..... | ..... |
| 1900 | 269,684,027 | 32 | 309 | 0.12 | 1.11 | 4 | 268 | 0.02 | 1.00 | ..... | ..... |
| 1901 | 293,299,816 | 30 | 110 | 0.10 | 0.38 | 6 | 67 | 0.02 | 0.23 | ..... | ..... |
| 1902 | 301,590,439 | 23 | 416 | 0.08 | 1.38 | 10 | 391 | 0.03 | 1.30 | 5,986 | 50,383 |
| 1903 | 357,356,416 | 41 | 257 | 0.12 | 0.72 | 6 | 207 | 0.02 | 0.57 | ..... | ..... |
| 1904 | 351,816,398 | 37 | 268 | 0.11 | 0.76 | 4 | 219 | 0.01 | 0.60 | ..... | ..... |
| 1905 | 392,722,635 | 53 | 326 | 0.13 | 0.83 | 13 | 277 | 0.03 | 0.70 | ..... | ..... |
| 1906 | 414,157,278 | 48 | 265 | 0.12 | 0.64 | 13 | 213 | 0.03 | 0.51 | ..... | ..... |
| 1907 | 480,363,424 | 46 | 899 | 0.10 | 1.87 | 14 | 854 | 0.03 | 1.78 | ..... | ..... |
| 1908 | 415,842,698 | 52 | 365 | 0.13 | 0.88 | 6 | 297 | 0.02 | 0.72 | ..... | ..... |
| 1909 | 460,814,616 | 49 | 295 | 0.11 | 0.64 | 17 | 249 | 0.04 | 0.54 | 6,079 | 75,804 |
| 1910 | 501,596,378 | 48 | 539 | 0.10 | 1.07 | 21 | 498 | 0.04 | 0.99 | 6,126 | 81,553 |
| 1911 | 490,601,317 | 14 | 334 | 0.03 | 0.63 | 12 | 329 | 0.02 | 0.67 | 6,179 | 79,398 |
| 1912 | 534,446,580 | 26 | 253 | 0.05 | 0.47 | 10 | 224 | 0.02 | 0.42 | 6,022 | 88,752 |

TABLE IV.—*Mine Explosions in Great Britain*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1803–Sept. 25, | Wall's End | 13 |
| 1805–Oct. 21, | Hebburn | 35 |
| Nov. 29, | Oxclose | 38 |
| 1806–Nov. 28, | Killingworth | 10 |
| 1808–Nov. 29, | Harraton | 4 |
| 1809–Sept. 14, | Killingworth | 12 |
| 1812–May 25, | Felling | 92 |
| Oct. 10, | Barrington Mill Pit | 24 |
| 1813–July 17, | Collingwood Main | 8 |
| Sept. 28, | Hall Pit, Fatfield | 32 |
| Dec. 24, | Felling | 22 |
| 1814–April 5, | Hawden Pit, Percy, Main | 4 |
| Aug. 12, | Hebburn | 11 |
| Sept. 9, | Leefield | 4 |
| 1815–June 2, | Sheriff Hill | 11 |
| Dec. 18, | Townley | 1 |
| 1817–June 30, | Row Pit, Harraton | 38 |
| 1817–Sept. 25, | Jarrow | 6 |
| Aug. 5, | Wall's End | 4 |
| Nov. 3, | Ouston | 1 |
| Dec. 18, | Plain Pit, Rainton | 27 |
| 1819–July 19, | Sheriff Hill | 35 |
| Oct. 9, | George Pit, Lumley | 13 |
| 1821–Oct. 19, | Nesham's Newbottle | 6 |
| Oct. 23, | Wall's End | 52 |
| Oct. 23, | Felling | 6 |
| 1823–Feb. 21, | Ouston | 4 |
| Nov. 3, | Plain Pit, Rainton | 59 |
| 1824–Oct. 5, | George Pit, Lumley | 14 |
| Nov. 19, | Dolly Pit, Newbottle | 11 |
| 1825–July 3, | Julit Pit, Fatfield | 11 |
| Oct. 5, | Hebburn | 4 |
| 1826–Jan. 17, | Jarrow | 34 |
| May 30, | Townley | 38 |

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## *Mine Explosions in Great Britain—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1826–Sept. 5, | Heworth | 5 |
| Oct. 27, | Benwell | 2 |
| 1828–March 15, | Jarrow | 8 |
| Sept. 1, | New Pit, Houghton | 7 |
| Nov. 20, | J. Pit, Washington | 14 |
| 1829–Dec. 3, | Willington | 4 |
| 1830–Aug. 3, | Jarrow | 42 |
| 1831–July 9, | King Pit, Wreckington | 3 |
| Sept. 20, | Willington | 7 |
| 1833–May 9, | Springwell | 47 |
| May 24, | Great Lumley | 2 |
| Nov. 1, | Black Fell | 3 |
| 1835–June 18, | Wall's End | 102 |
| Nov. 19, | Burdon Main | 11 |
| 1836–Jan. 28, | Down's Pit, Hetton | 22 |
| 1837–Dec. 6, | Springwell | 30 |
| 1838–Dec. 19, | Wall's End | 11 |
| 1839–June 28, | St. Hilda | 51 |
| 1840–June 6, | Haswell | 1 |
| 1841–April 19, | Bigge Pit, Willington | 32 |
| Aug. 5, | Thornley | 9 |
| Aug. 17, | Haswell | 1 |
| 1842–March 2, | West Cramlington | 1 |
| 1843–April 5, | King Pit, Wreckington | 28 |
| 1844–Jan. 18, | West Moor | 5 |
| Sept. 28, | Haswell | 95 |
| Oct. 15, | Coxlodge | 1 |
| Dec. 1, | Seghill | 2 |
| 1849–June 10, | Ferndale, Eng. | — |
| 1851–March 15, | Victoria Pit | 61 |
| May 26, | Washington Pit, Newcastle. | 2 |
| Aug. 19, | Washington Pit, Newcastle. | 35 |
| Oct. 23, | Killingworth Pit, Newcastle | 1 |
| Oct. 31, | Kilingworth Pit, Newcastle. | 9 |
| Dec. 20, | Warren Vale, Yorkshire | 52 |
| Dec. 22, | Ince Hall, Lancashire | 13 |
| 1852–April 24, | Norley Hall, Lancashire | 12 |
| May 6, | Hebburn, Durham | 22 |
| May 10, | Middle Duffryn | 65 |
| May 20, | Coppull, Lancashire | 36 |
| Dec. 22, | Elsecar, Yorkshire | 10 |
| 1853–March 12, | Risca, Monmouth | 10 |
| March 24, | Arley Mine | 58 |
| April 26, | Old Park, Worcestershire | 11 |
| July 1, | Bent Grange, Lancashire | 20 |
| 1854–Feb. 18, | Ince Hall | 89 |
| 1856–July 3, | Old Coal Pit, Monmouth | 11 |
| July 15, | Cymmer, Wales | 114 |
| Aug. 13, | Ramrod Hall, Staffordshire. | 11 |
| May 24, | Cyme Avon, Glamorgan | 12 |
| 1857–Feb. 19, | Lund Hill | 189 |
| May 27, | Tyr Nicholas, Monmouth. | 13 |
| July 31, | Keys, Lancashire | 40 |
| 1858–Feb. 2, | Bardsley | 55 |
| Feb. 25, | Lower Duffryn | 19 |
| May 28, | Bryndu, Glamorgan | 12 |
| Dec. 11, | Tyldesley, Lancashire | 25 |
| 1860–Feb. 15, | Higham, Yorkshire | 13 |
| March 2, | Burradon | 76 |
| Aug. 3, | Winstanley, Lancashire | 13 |
| Nov. 6, | Lower Duffryn, Glamorgan. | 12 |
| Dec. 1, | Risca, Wales | 142 |
| Dec. 20, | Hotton Sutton, Durham | 22 |
| 1861–March 8, | Blaengwawr, Glamorgan | 13 |
| Sept. 26, | South Mostyn, Flintshire | 10 |
| Nov. 1, | Shevington, Lancashire | 13 |
| 1862–Feb. 19, | Gethin, Glamorgan | 47 |
| Nov. 22, | Walker, Northumberland | 16 |
| Dec. 8, | Edmunds Main, Yorkshire. | 59 |
| 1863–March 6, | Coxlodge, Northumberland. | 26 |
| Oct. 17, | Morfa | 39 |
| Dec. 9, | Wynnstay, Denbigh | 13 |
| Dec. 26, | Lleynoi, Glamorgan | 15 |
| 1865–June 16, | Bedwelty | 26 |
| Dec. 20, | Cethin Cyforthfa, Glamorgan | 34 |
| 1866–Jan. 23, | High Brooks, Lancashire | 30 |
| May 4, | Garswood Park, Lancashire. | 12 |
| Dec. 12, | Oaks Colliery | 361 |
| Dec. 13, | Talk O'The Hill | 91 |
| Oct. 31, | Palton | 24 |
| June 14, | Victoria Pit | 38 |
| 1867–Aug. 20, | Garswood | 14 |
| Nov. 6, | Ferndale, Wales | 178 |
| Nov. 11, | Homer Hill, Staffordshire. | 12 |
| 1868–Sept. 30, | Wynnstay, Denbigh | 10 |
| Nov. 25, | Hindley Green | 62 |
| Dec. 26, | Haydock, Lancashire | 26 |
| 1869–April 1, | Highbrooks | 37 |
| June 10, | Ferndale, Glamorgan | 53 |
| July 21, | Haydock | 59 |
| Oct. 22, | Newbury, Somerset | 11 |
| Nov. 15, | Low Hall, Lancashire | 27 |
| 1870–Feb. 14, | Morfa, Glamorgan | 30 |
| July 7, | Silverdale, Staffordshire | 19 |
| July 23, | Charles Colliery | 19 |
| Aug. 19, | Brynn Hall | 20 |
| 1871–Jan. 10, | Renishaw | 20 |
| Feb. 24, | Pentre | 38 |
| March 2, | Victoria, Monmouth | 19 |
| Sept. 6, | Moss Pit | 70 |
| Oct. 25, | Seaham | 26 |
| 1872–Feb. 14, | Maesteg Marthyr, Glamorgan | 11 |
| March 28, | Lover's Lane, Lancashire | 27 |
| Oct. 7, | Morley | 34 |
| 1873–Feb. 18, | Talke, Staffordshire | 18 |
| 1874–April 14, | Dunkinfield | 54 |
| April 18, | Ince Hall, Lancashire | 15 |
| Nov. 20, | Rawmarsh, Yorkshire | 23 |
| Dec. 24, | Bignall Hill, Staffordshire. | 17 |
| 1875–April 30, | Bunker's Hill | 43 |
| Dec. 4, | Tredegar, Monmouth | 23 |
| Dec. 6, | Llan, Glamorgan | 16 |
| Dec. 6, | Swaithe Main | 143 |
| Dec. 15, | Mons Colliery | |
| 1876–Dec. 18, | South Wales, Monmouth | 23 |
| 1877–Jan. 23, | Stonehill, Lancashire | 18 |
| Feb. 7, | Foggs, Lancashire | 10 |
| March 10, | Wiegfach, Glamorgan | 18 |
| Oct. 11, | Pemberton, Lancashire | 36 |
| Oct. 22, | Blantyre | 207 |
| 1878–March 8, | Barwood Colliery | 17 |
| March 12, | Unity Brook | 43 |
| March 27, | Apedale, Staffordshire | 23 |
| June 7, | Haywood Wood | 189 |
| Sept. 11, | Aberdarne, Monmouth | 268 |
| 1879–March 4, | Victoria, Yorkshire | 21 |



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## *Mine Explosions in Great Britain—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1879–Jan. 13, | Dinas | 63 |
| July 2, | Blantyre | 28 |
| 1880–Jan. 21, | Leycett Colliery | 62 |
| July 15, | Risca, Wales | 120 |
| Sept. 8, | Seaham | 164 |
| Dec. 10, | Naval, Wales | 101 |
| 1881–Feb. 7, | Whitfield | 25 |
| Dec. 19, | Abram | 48 |
| 1882–Feb. 16, | Trimdon Grange | 74 |
| April 18, | Tudhoe | 37 |
| April 19, | West Stanley, Durham | 13 |
| April 25, | Whitehaven, Cumberland | 4 |
| May 2, | Baddesley | 23 |
| Nov. 7, | Claycross No. 7, Parkhouse, Derbyshire | 45 |
| 1883–Oct. 18, | Wharncliffe | 20 |
| Nov. 7, | Altham | 68 |
| 1884–Jan. 27, | Penygraig, Glamorgan | 14 |
| Sept. 6, | Hall End, Staffordshire | 7 |
| Nov. 8, | Pochin, Monmouth | 14 |
| 1885–March 2, | Usworth | 42 |
| April 8, | Great Fenton, Staffordshire | 8 |
| June 18, | Clifton Hall | 178 |
| Aug. 20, | Apedale, Staffordshire | 9 |
| Dec. 23, | Mardy | 81 |
| 1886–Aug. 13, | Woodends Pit | 38 |
| Sept. 10, | Dean Lane, Somerset | 10 |
| Oct. 2, | Altofts, Yorkshire | 22 |
| Dec. 2, | Elemore, Durham | 28 |
| 1887–Feb. 18, | Ynshir Colliery | 39 |
| May 28, | Udston Colliery | 73 |
| 1888–April 19, | St. Helens, Cumberland | 30 |
| 1889–Jan. 18, | Hyde | 23 |
| Jan. 28, | Dean Lane | 4 |
| Jan. 31, | Kirkland, Scotland | 1 |
| Feb. 25, | Lower Llanmorlais, Wales | 2 |
| March 13, | Brynmally | 20 |
| March 21, | Dipton | 1 |
| April 19, | St. Helens | 30 |
| June 27, | Dairy Farm | 1 |
| July 9, | Cwmbran, Wales | 1 |
| July 23, | Bothwell Park No. 2, Scotland | 1 |
| Aug. 17, | Chell | 3 |
| Aug. 21, | Whitwood | 2 |
| Oct. 12, | Lammark No. 2, Scotland | 1 |
| Oct. 16, | Mossfields | 64 |
| Oct. 25, | Aslmore Park | 1 |
| Oct. 31, | Saltwells | 2 |
| Nov. 4, | Hebburn A | 6 |
| Nov. 4, | Meiklehill No. 5, Scotland | 2 |
| Dec. 26, | Barrwood No. 1, Scotland | 1 |
| 1890–Jan. 22, | Glyn | 5 |
| Feb. 6, | Llanerc, Wales | 176 |
| March 3, | Devon, Scotland | 1 |
| March 10, | Morfa, Wales | 87 |
| March 24, | Gilvertheld No. 2, Scotland | 1 |
| April 4, | Hallside, Scotland | 1 |
| April 6, | Broomhouse, Scotland | 1 |
| May 2, | Thorncliffe | 2 |
| May 16, | Aberdure No. 9, Wales | 2 |
| May 16, | Vochriw, Wales | 1 |
| May 19, | Shetton Deep Pit | 1 |
| June 10, | Holmes, Scotland | 1 |
| Aug. 2, | Braich Y Gimmer, Wales | 1 |
| 1890–Aug. 28, | Malago Vale Argus Pit, Wales | 1 |
| Aug. 30, | Kinneil, Scotland | 2 |
| Sept. 8, | Butterworth Hall | 1 |
| Sept. 14, | Hanby Northwood Pit | 1 |
| Sept. 19, | Nether Croy No. 1, Scotland | 1 |
| Oct. 13, | Eastfield, Scotland | 1 |
| Nov. 1, | Peel Hall White | 1 |
| Nov. 3, | Raverhead | 1 |
| Dec. 19, | Longrigg, Scotland | 1 |
| 1891–Jan. 7, | Dowlais South Tunnel Pit, Wales | 1 |
| Feb. 10, | Beamish Second Pit | 2 |
| March 6, | Oak Farm | 1 |
| March 24, | Dearham | 1 |
| April 2, | Apedale Sladderhill Pit | 10 |
| May 9, | Shelton No. 1 | 1 |
| May 13, | Franch, Wales | 1 |
| May 13, | Brithdir, Wales | 1 |
| May 14, | Ashton Moss | 2 |
| June 19, | Llanbradach, Wales | 1 |
| July 16, | Llantwit Red Ash, Wales | 1 |
| July 23, | Addiewell, Scotland | 1 |
| Aug. 1, | Ross, Scotland | 2 |
| Aug. 2, | Haughhead, Scotland | 1 |
| Aug. 7, | Glass Houghton, Eng | 2 |
| Aug. 23, | Abarcanaid, Wales | 2 |
| Aug. 25, | Rhos Llantwit, Wales | 2 |
| Aug. 31, | Malago Vale | 10 |
| Sept. 10, | Low Hall | 2 |
| Oct. 1, | Aberdare Merthyr, Wales | 1 |
| Oct. 13, | Backworth C Pit | 1 |
| Nov. 11, | Meiros, Wales | 3 |
| Dec. 9, | Montagu Main | 2 |
| 1892–Feb. 8, | Dinnington | 1 |
| Feb. 10, | Hall End | 1 |
| Feb. 19, | Gadis Graig Pit, Wales | 1 |
| May 13, | Coxlodge | 1 |
| May 19, | Car House | 1 |
| June 1, | Browneyside, Scotland | 1 |
| June 17, | Holmes, Scotland | 1 |
| July 28, | Gartshore No. 3, Scotland | 1 |
| Aug. 8, | Ebbw Vale Marine, Wales | 1 |
| Aug. 26, | Park Slip, Wales | 112 |
| Sept. 24, | Netherton Old | 1 |
| Oct. 11, | Gartshore No. 9, Scotland | 1 |
| 1893–Feb. 11, | Whitwick No. 6 | 2 |
| Feb. 11, | Worfa, Wales | 1 |
| April 21, | Himley | 2 |
| May 1, | Sharlston | 3 |
| May 5, | Fishley | 1 |
| May 24, | Dunston | 2 |
| June 30, | Herbertshire No. 3, Scotland | 2 |
| June 30, | Butterknowle | 1 |
| July 4, | Combo, Thornhill | 139 |
| Aug. 19, | Hattonrig No. 3, Scotland | 1 |
| Oct. 24, | Elms | 1 |
| Nov. 13, | Cowdenbeath, Scotland | 1 |
| Oct. 13, | Camerton | 2 |
| Dec. 2, | Hem Heath | 1 |
| Dec. 8, | Wooley | 1 |
| 1894–Jan. 1, | Gorseinon, Wales | 1 |
| March 23, | Ayr-Sundrum No. 3, Scotland | 1 |
| April 18, | Portland No. 5, Scotland | 1 |

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## *Mine Explosions in Great Britain—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1894–May 19, | Allerton Main | 3 |
| June 23, | Albion, Wales | 290 |
| Aug. 1, | Alwynpia No. 3, Wales | 1 |
| Aug. 8, | Bank House | 1 |
| Aug. 9, | Rowley Hall | 1 |
| Aug. 17, | Black Boy | 1 |
| Sept. 6, | West Auckland | 1 |
| Sept. 10, | Radcliffe | 1 |
| Sept. 21, | East Connock | 1 |
| Sept. 26, | Wednesbury | 1 |
| Sept. 27, | International Anthracite, Wales | 1 |
| Oct. 15, | Common No. 10, Scotland. | 1 |
| Oct. 16, | Harecastle & Woodshutts Col | 1 |
| Oct. 17 | Ellismuer's No. 2, Scotland | 1 |
| Oct. 25, | Blair Hall, Scotland | 1 |
| Oct. 27, | Common No. 14, Scotland. | 1 |
| Oct. 27, | Sandwell Park | 4 |
| Nov. 10, | Gadlys-Dyllas, Wales | 2 |
| Nov. 22, | Ashmore Park | 1 |
| 1895–Feb. 6, | Timsbury | 2 |
| Feb. 9, | Morlais Vale, Wales | 2 |
| Feb. 13, | Glynea, Wales | 1 |
| Feb. 25, | Bwllfa Dare, Wales | 1 |
| March 15, | Malago Vale | 2 |
| April 6, | Seafield, Scotland | 1 |
| April 22, | Heddon | 1 |
| June 23, | Quarter, Scotland | 13 |
| May 6, | Llantwit Red Ash, Walcs | 1 |
| June 18, | Bullfield | 2 |
| July 10, | Crawshays Castle, Wales | 3 |
| July 16, | Abercrave, Wales | 1 |
| Aug. 1, | Aberdare Merthyr, Wales | 1 |
| Aug. 1, | Aldridge | 2 |
| Aug. 14, | Pentland, Scotland | 1 |
| Sept. 14, | Lea Green | 1 |
| Oct. 1, | Shakerley | 5 |
| Nov. 5, | Tannockside No. 2, Scotland | 1 |
| Oct. 11, | Blackwell A | 7 |
| Nov. 21, | Newmarket | 2 |
| 1896–Jan. 11, | West Cannock | 1 |
| Jan 11, | Lanmorlais, Wales | 2 |
| Jan. 27, | Ferndale Nos. 7 & 8, Wales | 57 |
| Jan. 28, | Bleennant, Wales | 1 |
| Feb. 1, | Foxhole, Wales | 1 |
| Feb. 4, | Saltwells, | 1 |
| Feb. 10, | Axwell Park | 2 |
| Feb 29, | Lammark No. 2, Scotland. | 1 |
| March 28, | Meiros, Wales | 3 |
| March 30, | Straiton, Scotland | 1 |
| April 13, | Brancepeth | 20 |
| April 27, | Swalwell | 1 |
| April 30, | Micklefield | 63 |
| May 6, | Birkrigg, Scotland | 1 |
| June 8, | Carfin No. 6, Scotland | 1 |
| June 15, | Bellfield, Scotland | 1 |
| June 19, | Abercave, Wales | 1 |
| Aug. 4, | Main Bryncoch, Wales | 7 |
| Aug. 17, | Sough Duffryn, Wales | 1 |
| Aug. 17, | Gawn | 1 |
| Sept. 16, | Cornsilloch, Scotland | 2 |
| Sept. 26, | Primrose Main | 1 |
| Dec. 3, | Drumsmudden, Scotland | 1 |
| Dec. 5, | Seaton Delaval | 1 |
| 1896–Dec. 29, | Garscube, Scotland | 1 |
| 1897–Jan. 5, | Broadoak, Wales | 5 |
| March 12, | Bothwell Park, Scotland | 1 |
| March 18, | Knowle | 1 |
| April 1, | Gadlys, Wales | 1 |
| May 13, | Darmgavd, Scotland | 2 |
| July 3, | Dalmeny, Scotland | 1 |
| Aug. 6, | Raveresthrope | 3 |
| Sept. 29, | Drumpeller No. 3 and No. 4, Scotland | 1 |
| Oct. 11, | Newliston, Scotland | 1 |
| Nov. 29, | Olvecote Tamworth | 1 |
| Dec. 10, | East Cannock | 1 |
| Dec. 20, | Blaendare | 1 |
| 1898–Jan. 28, | Drumpeller Nos. 3 and 4, Scotland | 4 |
| March 5, | Whinney Moor | 1 |
| March 31, | Kinneal, Scotland | 8 |
| Aug. 3, | Millfield | 1 |
| Aug. 17, | Rosehall, Scotland | 1 |
| Aug. 21, | St. Helens | 2 |
| Sept. 6, | Greasbro | 2 |
| Sept. 9, | Ayr, Scotland | 7 |
| Sept. 9, | Thankerton, Scotland | 1 |
| Sept. 29, | Annagher No. 5, Ireland | 3 |
| Oct. 23, | Nethercroy, Scotland | 1 |
| Nov. 3, | South Elswick | 1 |
| 1899–Jan. 9, | Gilbertfield No. 2, Scotland. | 1 |
| Jan. 10, | Pumpherston, Scotland | 1 |
| Feb. 17, | Walbottle | 1 |
| March 11, | Cadeby Main | 2 |
| March 18, | Gertshore No. 9, Scotland. | 1 |
| April 4, | Drumpeller Nos. 3 and 4, Scotland | 1 |
| May 15, | Etherty Grange | 1 |
| May 22, | Benarty, Scotland | 2 |
| May 24, | Woodhall No. 1, Scotland. | 1 |
| May 25, | Millfield | 4 |
| June 9, | Ynisgeinon, Wales | 1 |
| June 19, | Brownhill | 1 |
| June 20, | Holytown No. 5, Scotland. | 2 |
| Aug. 15, | Brandon | 6 |
| Aug. 15, | Wester Gartshore, Scotland. | 1 |
| Aug. 18, | Llest, Wales | 19 |
| Oct. 10, | Hamilton Palace No. 1, Scotland | 1 |
| Oct. 26, | Colderbank No. 1, Scotland | 1 |
| Nov. 3, | Bedlington A Pit | 1 |
| Nov. 23, | Point of Air | 2 |
| Dec. 6, | Kinneil, Scotland | 2 |
| Dec. 25, | Longriggend, Scotland | 1 |
| Dec. 27, | Ingliscard, Scotland | 1 |
| 1900–Jan. 8, | Middrie, Scotland | 1 |
| Feb. 12, | Carbarns, Scotland | 3 |
| Feb. 22, | Glespin, Scotland | 1 |
| March 8, | Hinley | 1 |
| March 12, | Pumpherston, Scotland | 1 |
| March 25, | Drumbrook No. 2, Scotland | 1 |
| March 26, | Cowdenbeath, Scotland | 1 |
| April 7, | Cadley Hill | 2 |
| April 9, | Walsall Wood | 1 |
| April 12, | Holytown No. 1, Scotland. | 2 |
| May 28, | Clynhebog, Wales | 2 |
| June 10, | Clynmil Drift, Wales | 1 |
| June 22, | Garw Feehan, Wales | 1 |

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## *Mine Explosions in Great Britain—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1900–Aug. 12, | Tannochside No. 3, Scotland | 1 |
| Aug. 17, | Portland No. 5, Scotland | 6 |
| Sept. 3, | Kemmirhill No. 2, Scotland | 1 |
| Oct. 3, | Hattonrigg Nos. 3 and 4, Scotland | 1 |
| Oct. 9, | Fieldhouse | 1 |
| Nov. 16, | Preston | 4 |
| Nov. 24, | Gilmithscroft No. 4, Scotland | 1 |
| Nov. 30, | Deelmont No. 1, Scotland | 1 |
| Dec. 3, | Oatlands | 1 |
| Dec. 9, | Hamstead | 1 |
| Dec. 28, | Ross, Scotland | 1 |
| 1901–Jan. 16, | Ystradgynlas | 1 |
| Jan. 23, | Kirkwood, Scotland | 1 |
| Jan. 25, | Haughhead, Scotland | 2 |
| Jan. 27, | Holytown No. 8, Scotland | 1 |
| Feb. 28, | Blaendare Slope | 2 |
| March 25, | Dinas, Wales | 4 |
| April 1, | Sandback | 2 |
| April 10, | Orrell | 4 |
| May 6, | Tursdale | 1 |
| May 16, | Gilmithscroft No. 3, Scotland | 1 |
| May 24, | Universal, Wales | 81 |
| May 27, | Talk O'The Hill | 4 |
| June 26, | Meiros, Wales | 1 |
| July 8, | Stenedough | 4 |
| July 20, | Fochriw, Wales | 1 |
| Aug. 19, | Hamstead | 2 |
| Aug. 26, | New Dale | 1 |
| Sept. 10, | Llanbradach, Wales | 8 |
| Sept. 18, | Pencoed, Wales | 1 |
| Nov. 12, | Edmondsley | 1 |
| Dec. 8, | Cornsilloch No. 3, Scotland | 2 |
| 1902–March 3, | Pemberton | 2 |
| April 2, | Garwood Hall No. 9 | 9 |
| April 3, | Glencraig, Scotland | 4 |
| April 19, | Bernarty, Scotland | 1 |
| May 14, | Todmorten Moor | 1 |
| May 21, | Dunnington | 1 |
| June 4, | Folchriw, Wales | 8 |
| July 4, | Woodhorn | 3 |
| July 8 | Elba, Wales | 1 |
| July 9, | Wrexham & Action | 1 |
| July 19, | Collena, Wales | 1 |
| July 26, | Shut End No. 13 | 1 |
| July 26, | Cleland, Scotland | 1 |
| July 29, | Darran, Wales | 1 |
| Aug. 1, | Pumpherston, Scotland | 3 |
| Aug. 15, | Skellyton No. 3, Scotland | 2 |
| Aug. 23, | Park Hill | 1 |
| Sept. 3, | McLaren, Wales | 16 |
| Oct. 3, | Blair No. 2, Scotland | 1 |
| Oct. 15, | Holmside | 1 |
| Nov. 25, | Prior Lee No. 2 | 1 |
| Dec. 1, | East Plean No. 3, Scotland | 1 |
| Dec. 9, | Orrell | 2 |
| 1903–Feb. 9, | Stanrigg, Scotland | 1 |
| Feb. 24, | Aldridge No. 1 | 1 |
| Feb. 25, | Darran, Wales | 2 |
| Mar. 23, | Ferndale No. 5, Wales | 1 |
| 1903–June 10, | Biggarford, Scotland | 1 |
| July 17, | Bogleshole, No. 4, Scotland | 2 |
| July 24, | Garriongill, Scotland | 1 |
| Oct. 5, | Westburg No. 2, Scotland | 1 |
| Nov. 9, | Llwynypia No. 3, Wales | 1 |
| Nov. 12, | Stargate Pit, | 1 |
| Dec. 11, | Braidhurst, Scotland | 2 |
| 1904–Jan. 3, | Saltwells No. 27 | 1 |
| Jan. 7, | St. Helens | 2 |
| Jan. 19, | Calder | 1 |
| March 19, | Saltwells No. 27 | 1 |
| May 16, | Shirebrook | 2 |
| June 13, | Watergate | 1 |
| June 14, | Allanshaw No. 1, Scotland | 2 |
| June 25, | St. Johns | 1 |
| July 25, | Drumbow, Scotland | 1 |
| Aug. 23, | Backworth | 1 |
| Aug. 23, | Sneyd | 3 |
| Sept. 1, | Blantyre Ferme, Scotland | 1 |
| Sept. 6, | Broomhouse, Scotland | 1 |
| Sept. 26, | Rosehall No. 14, Scotland | 1 |
| Oct. 2, | Swinhill No. 2, Scotland | 1 |
| Dec. 31, | Bowhill, Scotland | 1 |
| 1905–Jan. 21, | Elba, Wales | 11 |
| March 10, | Cambrian Coll., Wales | 33 |
| July 11, | Nat. Coll., Wales | 119 |
| 1906–June 1, | Count Herbert, Glamorgan | 5 |
| Nov. 10, | Albion, Glamorgan | 6 |
| Dec. 17, | Urpeth Coll | 4 |
| Dec. 14, | Wingate Grange | 25 |
| 1907–Nov. 26, | Whitehaven | 5 |
| March 19, | Benwell | 4 |
| Dec. 14, | Dinas Main, Wales | 7 |
| March 5, | Genwen, Wales | 6 |
| 1908–Feb. 20, | Washington Globe | 14 |
| April 9, | Norton Hill | 10 |
| Aug. 18, | Maypole | 75 |
| 1909–Feb. 17, | West Stanley | 167 |
| Oct. 29, | Darran Coll., Wales | 27 |
| 1910–May 11, | Wellington Mine, Whitehaven | 136 |
| Dec. 21, | Pretoria Coll., Boeton | 344 |
| 1911–April 12, | Ynyscedwyn Coll., Wales | 1 |
| April 30, | North Motherwell, Scotland | 1 |
| June 8, | Seven Sisters, Glamorgan, W | 1 |
| Aug. 19, | Meiklehill Coll., Scotland | 2 |
| Aug. 21, | Maltby Main Coll | 3 |
| Sept. 15, | Netherton Coll | 3 |
| Sept. 16, | Cwmaman Coll., Wales | 2 |
| Oct. 15, | Dechmont No. 3 Coll., Wale | 1 |
| Oct. 23, | Kellan Coll., Wales | 1 |
| Oct. 26, | Annandale Coll., Scotland | 1 |
| Nov. 3, | Holytown Coll., Scotland | 1 |
| Nov. 25, | Biggall Hill Coll | 6 |
| 1912–March 11, | Bentley Coll | 3 |
| March, — | Great Western Coll., Wales | 2 |
| March 27, | Navigation Coll., Wales | 2 |
| May 18, | Newport Coll., Monmouthshire | 4 |
| July 6, | Barnsley, Yorkshire | 3 |
| July 9, | Cadesby Mines, Conisborough | 87 |
| Sept. 15, | Dudley | 1 |

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TABLE V.—*Explosions Caused by Gas or Dust, in Great Britain, to Sept.* 15, 1912

| Month | Explosions Having Less than Five Fatalities | | | | Explosions Having More than Five Fatalities | | | | Total Explosions | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions |
| January | 18 | 27 | 1.50 | 5.82 | 14 | 369 | 26.36 | 6.45 | 32 | 396 | 12.38 | 6.08 |
| February | 23 | 34 | 1.48 | 7.44 | 18 | 919 | 51.06 | 8.30 | 41 | 953 | 23.24 | 7.79 |
| March | 27 | 44 | 1.63 | 8.74 | 19 | 608 | 32.00 | 8.75 | 46 | 652 | 14.17 | 8.75 |
| April | 20 | 36 | 1.80 | 6.48 | 18 | 462 | 25.67 | 8.30 | 38 | 498 | 13.11 | 7.22 |
| May | 31 | 53 | 1.71 | 10.03 | 14 | 662 | 47.29 | 6.45 | 45 | 715 | 15.89 | 8.56 |
| June | 23 | 27 | 1.17 | 7.44 | 13 | 1,002 | 77.08 | 5.99 | 36 | 1,029 | 28.58 | 6.84 |
| July | 22 | 34 | 1.55 | 7.12 | 15 | 829 | 55.27 | 6.91 | 37 | 863 | 23.32 | 7.03 |
| August | 38 | 62 | 1.63 | 12.30 | 17 | 537 | 31.59 | 7.83 | 55 | 599 | 10.89 | 10.46 |
| September | 26 | 37 | 1.42 | 8.41 | 18 | 747 | 41.50 | 8.30 | 44 | 784 | 17.59 | 8.36 |
| October | 32 | 41 | 1.28 | 10.36 | 22 | 706 | 32.09 | 10.14 | 54 | 747 | 13.83 | 10.27 |
| November | 24 | 39 | 1.63 | 7.77 | 21 | 636 | 30.29 | 9.68 | 45 | 675 | 15.00 | 8.56 |
| December | 25 | 37 | 1.48 | 8.09 | 28 | 1,784 | 63.71 | 12.90 | 53 | 1,821 | 34.36 | 10.08 |
| Totals | 309 | 471 | ..... | 100.00 | 217 | 9,261 | ..... | 100.00 | 526 | 9,732 | ..... | 100.00 |
| Averages | ..... | ..... | 1.52 | ..... | ..... | ..... | 42.68 | ..... | ..... | ..... | 18.50 | ..... |

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## TABLE VI.—*French Explosions Caused by Gas or Dust*

| Date | Place | Number of Fatalities |
|---|---|---|
| 1814–Oct. 28, | du Garston | 1 |
| 1817–June 8, | Latour | 1 |
| Aug. 7, | Lans Nom | 1 |
| 1818–Feb. 14 | des Souchettes | 1 |
| March 13, | Latour | 1 |
| March 29, | Latour | 1 |
| 1823–Aug. 9, | du Chanfour | 22 |
| 1826–June 29, | St. Jean de Valeriscle | 1 |
| Aug. 2, | de la Larade | 1 |
| 1832–July 2, | de l'Estang | 3 |
| Aug. 2, | St. Joseph | 10 |
| 1833–April 16, | Mamby | 1 |
| Oct. 30, | Mamby | 17 |
| 1834–May 29, | du Port de Lane | 1 |
| 1835–Aug. 30, | St. Claude | 5 |
| 1836–July 2, | Osmond | 2 |
| July 4, | Osmond | 4 |
| 1837–Sept. 20, | Osmond, | 1 |
| 1839–March 6, | du Creusot | 1 |
| 1842–July 3, | Beaunier | 4 |
| Aug. 18, | Charles | 15 |
| Sept. 4, | Luce | 1 |
| 1844–Oct. 15, | du Chene | 1 |
| 1845–April 16, | Abylon | 1 |
| 1846–Oct. 12, | Villefosse | 1 |
| Nov. 12, | de Ste. Barbe | 1 |
| 1847–Sept. 5, | Fournier | 3 |
| 1848–April 3, | Fournier | 5 |
| Sept. 20, | Bertrand | 1 |
| 1849–May 7, | Abylon | 1 |
| June 30, | de Cherond | 1 |
| Oct. 30, | Roche-la-Moliere | 1 |
| 1850–July 3, | de Villars | 5 |
| 1851–March 26, | d'Avaire | 11 |
| April 25, | Cinq Sous | 6 |
| May 9 , | St. Jean de Valeriscle | 1 |
| July 25, | du Raone | 3 |
| Aug. 26, | Marie | 1 |
| 1852–Oct. 21, | Luce | 1 |
| 1853–July 9, | Raves | 9 |
| July 29, | Cinq Sous | 13 |
| Oct. 30, | Abylon | 4 |
| 1854–Jan. 8, | Meyron | 1 |
| Feb. 6, | du Re Sokil | 1 |
| Feb. 6, | Canol | 2 |
| Aug. 5, | du Cret de Mars | 1 |
| July 2, | Abylon | 12 |
| Aug. 13, | La Respie | 4 |
| Aug. 25, | d'Agin Court | 6 |
| Sept. 2, | Marie Louise | 1 |
| 1855–Jan. 30, | Charles | 2 |
| Aug. 29, | Charles | 4 |
| Oct. 22, | Raves | 29 |
| Oct. 26, | Hippolyte | 2 |
| Dec. 17, | de Ste. Barbe | 1 |
| 1856–June 13, | Lans Nom | 2 |
| July 17, | de la Garenne | 1 |
| July 5, | du Cret de Mars | 1 |
| 1857–Jan. 26 | de la Pompe | 1 |
| April 2, | Marie Louise | 2 |
| June 4, | St. Mathieu | 3 |
| 1858–Jan. 20, | Fendue | |
| May 29, | Werbrauck | |
| Aug. 18, | Delaynaud | |
| Aug. 16, | Cinq Sous | |
| Oct. 2, | de la Garenne | |
| 1859–May 14, | du Lud | |
| 1860–April 6, | Ste. Marie | |
| July 30, | Bleuse Bome | |
| Aug. 30, | Villars | |
| Oct. 15, | Ernestine | |
| 1861–Jan. 27, | de la Chaux | |
| May 26, | du Breuil | |
| July 2, | Gallois | |
| July 9, | Ste. Amelie | |
| July 22, | Grangier | |
| 1862–March 24, | Beauvais | |
| April 10, | St. Urbain | |
| May 22, | St. Charles | |
| June 1, | Mircs | |
| Aug. 23, | St. Claude | |
| 1863–Jan. 26, | Bleuse-Bome | |
| May 22, | Fendue de Villars | |
| Aug. 24, | Davy | |
| Aug. 27, | du Gabet | |
| Oct. 28, | Luey | |
| 1864–Feb. 11, | Monterral | |
| 1865–Jan. 30, | Beaunier | |
| Feb. 1, | de la Chaux | |
| Feb. 9, | Ourenne | |
| July 17, | Lacroix | |
| 1866–April 9, | Jabin | |
| April 16, | Beauvais | |
| 1867–Aug. 11, | Fendue de Villars | |
| Oct. 12, | Cinq Sous | |
| 1868–Jan. 3, | St. Augustin | |
| Feb. 2, | No. 1 Ostricourt | |
| Feb. 19, | St. Martin | |
| Aug. 23, | Monterrax No. 1 | |
| Sept. 25, | No. 1 Lievin | |
| 1869–March 19, | d'Herin | |
| May 21, | Monterrax No. 2 | |
| Aug. 12, | St. Louis | |
| Aug. 21, | Monterrax No. 2 | |
| Nov. 22, | de la Chaux | |
| 1870–May 14, | Lucy | |
| June 9, | St. Paul | |
| July 7, | Abraham | |
| July 14, | Lucy | |
| July 19, | de la Compe | |
| Aug. 21, | Fournier | |
| 1871–Feb. 1, | Ste. Marie | |
| Feb. 21, | Ste. Helene | |
| April 17, | de la Garenne | |
| June 19, | Jabin | |
| July 3, | No. 2 Lievin | |
| Sept. 8, | Jabin | |
| 1872–April 22, | No. 1 Auchy-la-Tour | |
| Nov. 8, | Ste. Eugenie | |
| 1873–Feb. 21, | Le Chanfour | |
| Feb. 23, | Charles | |
| April 20, | Devillaine | |

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## *French Explosions—Continued*

| Date | Place | Number of Fatalities |
|---|---|---|
| '3–May 15, | Devillaine | 1 |
| June 7, | No. 2 Auchy-au-Bois | 7 |
| Sept. 17, | No. 3 Lens | 1 |
| '4–Jan. 30, | du Raone | 1 |
| March 25, | No. 4 Lens | 5 |
| July 23, | Davy | 2 |
| July 31, | No. 2 Noaix | 2 |
| Oct. 12, | Reussite | 3 |
| '5–Jan. 3, | du Sosprat | 1 |
| Jan. 15, | de Orelys | 1 |
| April 5, | Dolonieux | 7 |
| May 9, | No. 2 Auchy-au-Bois | 2 |
| July 28, | No. 1 Auchy-au-Bois | 3 |
| Oct. 22, | Marie | 4 |
| '6–Jan. 4, | Jabin | 186 |
| Dec. 24, | Malastre | 3 |
| '7–Feb. 22, | No. 1, Lievin | 3 |
| June 9, | No. 3 Lens | 1 |
| '8–Nov. 7, | de l'Ouert | 1 |
| '80–Feb. 7, | No. 2 Ferfay-Cauchy | 1 |
| April 5, | du Feljas | 4 |
| April 5, | Bonnepart | 1 |
| April 29, | de l'Echo | 2 |
| July 20, | No. 1 Flechinelle | 1 |
| '81–April 16, | No. 1 de la Vernade | 2 |
| May 25, | No. 1 de Lampret | 1 |
| May 17, | du Lavpiat | 8 |
| May 28, | No. 1 de Lampret | 5 |
| July 29 | St. Mathieu | 4 |
| Oct. 31, | des Moronniers | 1 |
| Aug. 22, | de Leforest | 1 |
| '82–April 13, | No. 3 Lievin | 8 |
| Jan. 28, | des Rosiers | 1 |
| July 12 | St. Felix | 1 |
| '83–Jan. 24, | No. 1 Courcelles-les-Lens | 1 |
| Feb. 12, | No. 5 Lievin | 2 |
| Feb. 21, | Ourenne | 8 |
| April 9, | Beaunier | 1 |
| April 13, | l'Echirreur | 4 |
| April 16, | No. 1 Lievin | 7 |
| '84–Jan. 5, | Devillaine | 1 |
| 1884–Jan. 11, | No. 2 Ferfay-Cauchy | 17 |
| July 24, | Beaunier | 2 |
| Aug. 23, | No. 1 Bully Grenay | 1 |
| 1885–Jan. 14, | No. 1 Lievin | 28 |
| May 21, | de l'Arbonnet | 1 |
| June 7, | No. 1 Noaix | 3 |
| Aug. 21, | No. 1 Courcelles-les-Lens | 10 |
| 1886–Feb. 27, | No. 1 l'Escaipelle | 1 |
| June 24, | St. Charles | 23 |
| 1887–March 1, | Chatehes No. 1 | 79 |
| July 10, | Sud | 1 |
| July 10, | Notre Dame | 1 |
| Nov. 16, | Montmarte No. 2 | 1 |
| 1888–July 15, | de Molieres | 1 |
| Aug. 28, | No. 2 Ferfay-Cauchy | 2 |
| Nov. 2, | de Compagnac | 49 |
| 1889–March 15, | Central | 17 |
| May 1, | No. 1 Noaix | 1 |
| July 3, | Verpellieux | 207 |
| 1890–July 8, | Chapelon | 1 |
| July 29, | Pelisnier | 113 |
| Aug. 4, | Pelisnier | 3 |
| 1891–March 3, | de Creal | 2 |
| Aug. 7, | de Compagnac | 1 |
| Dec. 6, | de la Manufacture | 62 |
| 1895–July 9, | Pontil | 1 |
| Sept. 25, | No. 5 Noaix | 1 |
| 1896–May 6, | No. 1 Vicoigne | 1 |
| Aug. 14, | Beneges | 1 |
| Aug. 17, | Herin | 1 |
| 1897–March 5, | No. 1 Droeviert | 1 |
| April 6, | Ste. Engenie | 4 |
| July 26, | de l'Arbonnet | 1 |
| 1900–May 2, | Renard | 1 |
| June 29, | de Creal | 1 |
| Nov. 12, | No. 3 Bully-Grenay | 1 |
| 1901–March 21, | No. 2 Droeviert | 4 |
| July 19, | de Molieres | 9 |
| Aug. 1, | No. 1 Bully-Grenay | 1 |
| 1902–Oct. 15, | No. 8 de Lampret | 8 |
| 1903–Aug. 26, | No. 2 Bully-Grenay | 2 |

## TABLE VIII.—*Belgian Explosions Caused by Gas or Dust*

| Date | Place | Number of Fatalities |
|---|---|---|
| 887–March 4, | LaBoule, Quaregnon | 113 |
| 891–July 19, | Forchies | 27 |
| Oct. 28, | No. 8 de la Lourire | 2 |
| 892–Feb. 27, | Belle Vue & Bienvinne | 2 |
| Mar. 11, | No. 3 Bois de la Haye | 160 |
| May 31, | No. 6 D'Hornee-Wasmes | 1 |
| 893–May 23, | Chaivirs | 5 |
| Aug. 7, | Chaivirs | 1 |
| Oct. 21, | Soxhluse | 1 |
| 894–April 1, | No. 7 du Gouppe | 3 |
| May 12, | No. 10 de Feni-Raisin | 1 |
| May 27, | Viernoy | 9 |
| June 2, | Pays-Bas | 3 |
| June 29, | No. 2 Charbonnages Rewnis | 2 |
| June 12, | Alliance | 2 |
| 1894–Sept. 9, | Ste. Ive | 1 |
| 1895–July 29, | St. Alphonse | 1 |
| 1898–Mar. 25, | Cinq-Gustave | 4 |
| May 25, | Crachet No. 12 | 16 |
| July 3, | No. 7 du Grand Homu | 1 |
| Aug. 13, | St. Arthur | 8 |
| 1900–May 14, | St. Bernard | 1 |
| July 12, | Albart | 1 |
| 1901–April 26, | de Buisson | 19 |
| 1903–June 5, | St. Jacques | 1 |
| 1904–Jan. 30, | No. 9 du Gouppe | 2 |
| 1905–July 17, | Viernoy | 16 |
| 1908–Jan. 19, | No. 5 Couchant du Fluin | 10 |
| Feb. 29, | No. 1 de Ohlin | 2 |
| 1909–Apr. 17, | Leval | 1 |

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TABLE VII.—*Explosions of Gas and Dust in Coal Mines of France, by Months,* 1814 *to* 1904

| Month | Explosions Having Less than Five Fatalities | | | | Explosions Having More than Five Fatalities | | | | Total Explosions | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions |
| January | 14 | 18 | 1.29 | 8.97 | 3 | 231 | 77.00 | 6.25 | 17 | 249 | 14.65 | 8.33 |
| February | 15 | 26 | 1.73 | 9.62 | 2 | 47 | 23.50 | 4.17 | 17 | 73 | 4.29 | 8.33 |
| March | 7 | 11 | 1.57 | 4.49 | 5 | 117 | 23.40 | 10.42 | 12 | 128 | 10.67 | 5.88 |
| April | 15 | 31 | 2.07 | 9.62 | 7 | 66 | 9.43 | 14.58 | 22 | 97 | 4.41 | 10.79 |
| May | 15 | 18 | 1.20 | 9.62 | 4 | 49 | 12.25 | 8.33 | 19 | 67 | 3.53 | 9.31 |
| June | 11 | 18 | 1.64 | 7.05 | 2 | 30 | 15.00 | 4.17 | 13 | 48 | 3.69 | 6.37 |
| July | 29 | 56 | 1.93 | 18.59 | 7 | 368 | 52.57 | 14.58 | 36 | 424 | 11.78 | 17.65 |
| August | 22 | 38 | 1.73 | 14.10 | 10 | 136 | 13.60 | 20.84 | 32 | 174 | 5.44 | 15.70 |
| September | 8 | 11 | 1.37 | 5.13 | 1 | 70 | 70.00 | 2.08 | 9 | 81 | 9.00 | 4.41 |
| October | 13 | 22 | 1.69 | 8.33 | 4 | 143 | 35.75 | 8.33 | 17 | 165 | 9.71 | 8.33 |
| November | 5 | 7 | 1.40 | 3.20 | 2 | 90 | 45.00 | 4.17 | 7 | 97 | 13.86 | 3.43 |
| December | 2 | 4 | 2.00 | 1.28 | 1 | 62 | 62.00 | 2.08 | 3 | 66 | 22.00 | 1.47 |
| Totals | 156 | 260 | ..... | 100.00 | 48 | 1,409 | ..... | 100.00 | 204 | 1,669 | ..... | 100.00 |
| Averages | ..... | ..... | 1.67 | ..... | ..... | ..... | 29.35 | ..... | ..... | ..... | 8.18 | ..... |

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TABLE IX.—*Explosions Caused by Gas or Dust in Belgium*, 1891 *to* 1909

| Month | Explosions Having Less than Five Fatalities | | | | Explosions Having More than Five Fatalities | | | | Total Explosions | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions | Number | Fatalities | Average Number of Fatalities per Explosion | Percentage of Total Explosions |
| January | 1 | 2 | 2.00 | 5.00 | 1 | 10 | 10.00 | 10.00 | 2 | 12 | 6.00 | 6.67 |
| February | 2 | 4 | 2.00 | 10.00 | ..... | ..... | ..... | ..... | 2 | 4 | 2.00 | 6.67 |
| March | 1 | 4 | 4.00 | 5.00 | 2 | 273 | 136.50 | 20.00 | 3 | 277 | 92.33 | 10.00 |
| April | 2 | 4 | 2.00 | 10.00 | 1 | 19 | 19.00 | 10.00 | 3 | 23 | 7.67 | 10.00 |
| May | 3 | 3 | 1.00 | 15.00 | 3 | 30 | 10.00 | 30.00 | 6 | 33 | 5.50 | 20.00 |
| June | 4 | 8 | 2.00 | 20.00 | ..... | ..... | ..... | ..... | 4 | 8 | 2.00 | 13.33 |
| July | 3 | 3 | 1.00 | 15.00 | 2 | 43 | 21.50 | 20.00 | 5 | 46 | 9.20 | 16.66 |
| August | 1 | 1 | 1.00 | 5.00 | 1 | 8 | 8.00 | 10.00 | 2 | 9 | 4.50 | 6.67 |
| September | 1 | 1 | 1.00 | 5.00 | ..... | ..... | ..... | ..... | 1 | 1 | 1.00 | 3.33 |
| October | 2 | 3 | 1.50 | 10.00 | ..... | ..... | ..... | ..... | 2 | 3 | 1.50 | 6.67 |
| November | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| December | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| Totals | 20 | 33 | ..... | 100.00 | 10 | 383 | ..... | 100.00 | 30 | 416 | ..... | 100.00 |
| Averages | ..... | ..... | 1.65 | ..... | ..... | ..... | 38.30 | ..... | ..... | ..... | 13.87 | ..... |

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TABLE X.—*Production, and Fatal Accidents Caused by Explosions of Gas or Dust in Coal Mines of United States, for Five-Year Periods from* 1870 *to* 1912, *Inclusive*

| Period | Production, Short Tons | Total Number of Accidents | | | | Accidents Having More Than Five Fatalities | | | | Average Number of Mines | Average Production per Year per Mine | Accidents per Mine | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Accidents | Fatalities | Accidents per Million Tons | Fatalities per Million Tons | Accidents | Fatalities | Accidents per Million Tons | Fatalities per Million Tons | | | Total Number | More than Five Fatalities |
| 1870–74 | 241,582,459 | 54 | 86 | 0.22 | 0.36 | 3 | 22 | 0.01 | 0.09 | 1,998 | 24,100 | 0.027 | 0.0015 |
| 1875–79 | 292,171,479 | 37 | 83 | 0.13 | 0.28 | 7 | 45 | 0.02 | 0.15 | 2,862 | 20,400 | 0.013 | 0.0024 |
| 1880–84 | 496,776,865 | 46 | 369 | 0.09 | 0.74 | 12 | 315 | 0.02 | 0.63 | 3,118 | 31,800 | 0.015 | 0.0039 |
| 1885–89 | 645,380,403 | 29 | 225 | 0.04 | 0.35 | 11 | 189 | 0.02 | 0.29 | 2,761 | 46,700 | 0.011 | 0.0040 |
| 1890–94 | 858,761,003 | 81 | 481 | 0.09 | 0.56 | 18 | 372 | 0.02 | 0.43 | 3,369 | 48,000 | 0.024 | 0.0053 |
| 1895–99 | 1,059,050,545 | 119 | 531 | 0.11 | 0.50 | 24 | 397 | 0.02 | 0.37 | 4,679 | 45,300 | 0.025 | 0.0051 |
| 1900–04 | 1,573,747,096 | 163 | 1,360 | 0.10 | 0.86 | 30 | 1,152 | 0.02 | 0.73 | 5,985 | 52,600 | 0.027 | 0.0050 |
| 1905–09 | 2,163,900,651 | 248 | 2,150 | 0.11 | 0.99 | 63 | 1,890 | 0.03 | 0.87 | 6,051 | 71,500 | 0.041 | 0.0104 |
| 1910–12[a] | 1,526,664,275 | 88 | 1,126 | 0.06 | 0.74 | 43 | 1,051 | 0.03 | 0.68 | 6,109 | 83,300 | 0.015 | 0.0070 |
| Totals.. | 8,858,034,776 | 865 | 6,411 | ..... | ..... | 211 | 5,433 | ..... | ..... | ..... | ..... | ..... | ..... |
| Average. | ..... | ..... | ..... | 0.10 | 0.72 | ..... | ..... | 0.02 | 0.61 | 4,018 | 51,300 | 0.023 | 0.0057 |

[a]Three years only.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

# Copper Smelting in Japan*

BY MANUEL EISSLER, PARIS, FRANCE

EDITED BY BURR A. ROBINSON, ASSISTANT SECRETARY

(New York Meeting, February, 1915)

## INTRODUCTION

THE history of copper metallurgy in Japan goes back into remote ages, of this there is abundant proof, and that the working of this metal is closely connected with the artistic development of this nation is evidenced by some of the glorious monuments which have been produced by these ancient metal workers. The traveler sees to-day in a perfect state of preservation bronze statues 150 ft. high, and over, 300 ft. in width, and 170 ft. in depth, delicately chased and beautifully marked, and when one reflects that some of them date back to the eighth century A. D., the question naturally suggests itself, how did these ancient metallurgists with their crude and primitive appliances accomplish this stupendous task?

The bodies of these images are formed of bronze castings 10 to 12 in. thick, which were gradually built up and soldered together, but other pieces of magnitude, like the heads and other portions of the body are single castings. In many cases another method of construction must have been adopted. The walls of the molds were gradually built up and as the lower portion of the castings cooled, the walls were raised higher and higher and fresh metal must have been added, instead of constructing the whole mold first and then making the casting in one piece.

There are still traces of substantial gilding visible on these statues, and how the pious Buddhist priests who were the constructors of them got the precious metal, is preserved in legendary tales.

As a matter of interest I will mention that mines in Japan are not generally worked as joint stock enterprises, but are mostly family properties which have descended to their present owners from their ancestors. Only in recent years have some new discoveries, in rare

* Based on notes taken during a journey to the principal mining districts of the Japanese Empire in 1906 and 1907.



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cases, sought the assistance of foreign capital. Only one mining company has its shares dealt in on the stock exchange and mines are worked like other industrial undertakings. Owing to the abundance of cheap labor they are yielding profits on exceedingly low-grade ores. The present production of copper in Japan is about 45,000 tons a year and from what I have seen of the country I believe it will gradually increase.

What actuated me in presenting this modest work to our Institute was my desire to place before the members a description of how old methods are still adhered to, even at some of the larger mines, where they are installed alongside the most modern and improved American and European methods. It was particularly this commingling of the old with the new which interested me and may also prove of interest to others.

I take occasion to thank all those gentlemen in Japan who have given me the opportunity of visiting their mines and who extended to me their kind hospitality, and I am specially indebted to Dr. Abe and to Dr. Watanabe for their introduction to the mining community of Japan.

This paper shows that Japan understood hundreds of years ago how to extract copper and other metals, and produce with them works of art which will stand as everlasting monuments to prove the high culture of this nation in past ages, when some of our European countries were still in darkness, and when Grecian and Roman art was forgotten.

## I. Smelting at the Besshi Works of the Sumitomo Company

The mineral at the Besshi mine is an intimate mixture of iron and copper pyrites and contains considerable magnetic iron and silica. The average contents of the ores are about 4 per cent. of copper. The quantity of ore smelted in 1905 amounted to 32,000,000 kwans (1 kwan = 8.26 lb.) or 132,160 tons and the production of copper amounted to 5,200 tons, showing an extraction of 3.93 per cent.; consequently 25.4 tons of ore were required to produce one ton of copper. On the basis of 340 working days, 388 tons are smelted daily. There are no precious metals in the Besshi ore.

The smelting works are situated on Shisaka Island in the Inland sea and distant 10 miles from the mines which are on Shikoku mainland. The ores are brought in boats of 30 tons capacity, and generally six of these are propelled by means of a tug boat from the landing stage on the island. The ore is hoisted in 1-ton trucks on a double-track line up an incline by means of a winding engine to a station 158 ft. above sea level and is distributed from here to the roasting stalls which are disposed along the slope of the hill. There are 635 ore-roasting stalls and 126 matte-roasting stalls.

The roasting stalls, which are made of slag bricks, are 13 ft. long, 5.5 ft.

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wide and 6 ft. deep. The front walls are 3 ft. thick and the division walls 2.5 ft. In the front there are two charge openings 4 ft. high and 3 ft. wide. There are nine small openings in the back walls which communicate with the flue. The four benches or terraces on which the stalls are built have each a length of 448 ft.; the width of the first bench is 88 ft. The flues into which the stalls debouch are 5 ft. 8 in. wide and 6 ft. in height. The transverse flue, into which the longitudinal flues debouch, is 6 ft. wide and 9 ft. 9 in. high and connects with a dust chamber 10 ft. wide and 21 ft. high leading to the smoke-stack.

For roasting, wood and 5 per cent. of the weight of ore in coke are employed. The ore-roasting stalls hold 35 tons each and the roast is finished in 45 days. The matte-roasting stalls hold 30 tons each; roasting is finished in two weeks. For taking the roasted matte and ores down hill to the smelters, incline tracks working automatically are employed. The stalls for matte roasting are under sheds, those for ores in the open.

### *The Blast Furnaces*

There are three blast furnaces in the ore-smelting house, which have each a smelting capacity of 200 tons of ore, and as only two furnaces are at work and one is held in reserve, the quantity of ore smelted daily is about 380 to 400 tons. Owing to the basic nature of ore, 60 tons of siliceous rock are added; the daily consumption of coke is about 70 tons or 15 per cent. of the furnace charge. The siliceous rock is mined from an adjacent island and is barren. The ore after stall roasting contains from 10 to 12 per cent. of sulphur.

The three water-jacket furnaces are each 16 ft. long and 3 ft. 3 in. wide, equipped with ten 4-in. tuyères on one side, 12 tuyères on the other or back portion and one tuyère on one of the ends. The front consists of six water jackets, 5 ft. 8 in. high, 3 ft. wide, with a water space of 4 in. These water jackets are built at the workshops of the company in the Niihama mining village. The tuyères are all provided with cutoff valves. The water jackets are held together by a ring made of steel rails. From the center of the tuyères to the feed floor is only 7.5 ft. The cast-iron drop bottom of each furnace is held in position by jack screws. The top portion of the furnace is supported by 10 iron columns. The air pipes around the furnaces are 12 in. in diameter. The cold-water pipes to jackets are 2.5 in. in diameter and so are the outlet pipes on top of jackets. The slag spout is water jacketed with coils of pipes. Each water jacket is provided with two hand holes and two valves for drawing off the sediment. As sea water is used in the jackets, great attention is paid to the cleaning of jackets. When cleaning out the jackets, which is done once a month, a hose is attached to one of the valves and water under pressure is pumped into them to stir up the sediment. The water

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used in the water jackets is pumped from the sea into a large brick reservoir holding 253,600 gal.

On top of each furnace are three short iron chimneys, 3 ft. in diameter and lined with brick, leading into a circular dust chamber 11 ft. in diameter lined with fire clay which runs the total length of the three furnaces and on top of them. This dust chamber is cleaned out with compressed air.

The slag and matte run continuously into a circular forehearth, 10 ft. in diameter and 4 ft. high, which is water jacketed. In the bottom are several layers of fire bricks. The matte can be tapped from both sides of the hearth. There are also several hand holes near the bottom of this forehearth to clean out the sediment. Matte is drawn every 4 hr. Any slag collected from the matte pots is resmelted.

The daily production is about 60 tons of matte, or 15 per cent. of the weight of ore smelted and the same contains from 27 to 30 per cent. of copper. Each furnace requires 4,000 gal. of jacket water per hour and experiments are now being made with water jackets made of copper.

The matte composition is as follows: Iron, 40.22 per cent.; sulphur, 18.88 per cent.; copper, 30.78 per cent. The slag composition is: $SiO_2$, 35.48 per cent.; FeO, 51.73 per cent.; $Al_2O_3$, 7.35 per cent.; copper, 0.429 per cent.

*Smelting for Second Matte*

The first matte produced is broken up by hand labor and goes to the matte-roasting stalls and after roasting still contains 12 to 13 per cent. of sulphur. It is then smelted in blast furnaces, situated in a separate building. This concentrated or second matte contains 70 per cent. of copper. The molten matte from these furnaces is run into reverberatory furnaces, which convert it into blister copper 98 to 99 per cent. fine. The blister copper is taken to another building and smelted in reverberatory refining furnaces and made into fine rose copper of 99.7 per cent. fineness.

There are three circular matte-smelting furnaces, which have a capacity of 30 tons each; two are constantly in operation, while one is held in reserve or is undergoing repairs. With the matte is also smelted all refuse from the blister and refining furnaces as well as all skimmings and dross. To the charges is added 15 to 16 per cent. of the siliceous rock as above mentioned and 17 per cent. of coke is used. Why the owner of the mine does not buy siliceous copper ores to utilize in place of waste rock, I do not know, unless there is a lack of such material in the neighboring mines. These matting furnaces are 3 ft. in diameter and have eight 2.5-in. tuyères each. The water jackets are 4 ft. high and the distance from center of the tuyères to bottom of the jackets is 6 in. The crucible shell is 4 ft. deep. The distance from the tuyère level to feed floor is 7.5 ft.

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The slag is run into pots and after cooling the bottom cone is broken off and returned to the ore-smelting furnaces. The same is also done with the slag which is drawn off from the forehearth in the first matte-smelting department just before tapping the matte. The slags from the No. 2 matte smelting, which are thus returned to the ore-smelting furnaces, contain 1.5 per cent. copper.

The crucibles of the matting furnaces are lined with fire bricks and with a layer of brasque composed of charcoal powder and clay. These furnaces are erected on an elevated platform, so that there is sufficient grade for the molten material to run direct into the blister furnaces. In the dust chamber 10 per cent. of the weight of material is collected and the dust contains 30 per cent. of copper; it is melted down and returned to the furnace.

### *The Blister Furnaces*

There are six reverberatories, two for each cupola. Their dimensions are 16 ft. 8 in. by 7 ft. 8 in. with fireplaces 5 ft. 5 in. by 3 ft. 5 in. The matte runs alternately into these furnaces by means of sheet-iron gutters lined with clay and charcoal, which are suspended on chains. Compressed air at 6-lb. pressure is blown in on both sides of the reverberatory furnaces. Each furnace holds 3 tons of matte and every 12 hr. 1.8 tons of blister copper is drawn. The furnaces are filled with the molten material from an opening in the top. The quantity of fuel required is equal to 10 per cent. of the weight of the matte. The slags from these furnaces go back to the matting furnaces. The chimneys from the six furnaces debouch into an overhead flue, or dust chamber, supported on independent iron girders. The top portions of the chimneys, where they connect with the dust chamber, are made of boiler iron. The blister copper is ladled out, cast into ingots and sent to the refining furnace. The ingots weigh 40 lb., and the ratio of blister copper to refined copper is 95 per cent.

### *Refining Furnaces*

There are two refining furnaces having each a capacity of 15 tons in 24 hr. The consumption of fuel as given to me was: Coal 32 per cent.; charcoal, 1.6 per cent.; poling wood, 6 per cent.; total, 39.6 per cent. of the weight of copper, which seems to me extremely high.

The sand or ganister for making the bottom of these furnaces comes from the Province of Iwagi in the northern portion of Japan.

For home consumption the copper is cast into square plates 12 by 12 in., weighing 42 *kin* or 54.6 lb. (1 kin = 1.3 lb.). The ingots for export weigh 23 lb. The molds are of copper and rest on a circular table which rotates by mechanical means. A thick muddy water is poured into the hot molds and as the water evaporates a clay lining is left behind which

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prevents the ingots sticking to the molds. When ladling the metal into them, a woman drops into each mold a small strip of refined copper, which superstition says imparts great virtues to the metal. This practice dates from mediæval times and is religiously kept up. The Sumitomo brands enjoy a high reputation not only in Japan, but also in other countries.

*Dust Chambers, Flues and Chimneys*

In the ore-smelting department, the iron flue on top of the blast furnaces is 13.7 ft. in diameter, lined with fire clay, giving an inner diameter of 11 ft.; the length of this iron flue is 153 ft. This flue debouches into a dust chamber 121 ft. in length, 14 ft. wide and 21 ft. high, which connects with another dust chamber 220 ft. long, 11 ft. high and 14 ft. wide. The latter joins the chimney which has an internal diameter of 15 ft. at the base and a height of 210 ft. Its outer diameter is 28 ft. at the base, consequently the brick work is 6.5 ft. thick. At the top of the chimney, the outer diameter is 17 ft. The foundations of the chimney are 35 ft. in diameter. The walls of the dust chambers have a thickness of 1.87 ft. and the arches 1.49 ft.

The flue from the matte-smelting furnaces is also of sheet iron. It connects with a vertical brick flue leading to a dust chamber which is 45.5 ft. long, 9 ft. wide, 15 ft. high and connects with another chamber 76.5 ft. long, two stories high, each story 12 ft. wide and 24 ft. high. A flue 51 ft. long, 7 ft. wide, 10.5 ft. high connects the last chamber with the chimney. The flues from the roasting stalls are also connected with extensive dust chambers leading to an independent chimney 204 ft. high of similar dimensions as described above. The works present an imposing appearance and were erected regardless of expense.

Sea water is used in the boilers and they are cleaned once a month. The mine is provided with extensive machine and repair shops and an iron foundry.

*Cement Copper from the Mine Water*

The mine discharges about 25 cu. ft. of water per minute, which is passed through a series of brick tanks, lined with cement. There are 152 of these tanks arranged in four rows of 38 each. The dimensions of these tanks are 12 ft. in length, 8 ft. in width and 4.5 ft. in depth.

At the head of the tanks is a large distributing reservoir for settling the sediment. Between the second and third row is a main gutter into which the water is discharged when cleaning up. There is an outer gutter for the first row and one for the fourth. The tanks contain coke and scrap iron in twelve layers arranged alternately. The coke when saturated is sent direct to the smelter, as the cement copper settles in its pores. The yearly production is 225 tons cement copper, which contains 60 per cent. copper.

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*Remarks and Observations*

I have obtained no data as to cost of production. Judging from the character of the ore I think that eventually pyritic smelting will be adopted at these works, but it will be necessary to mix the ores with a siliceous copper ore so as to slag excess of iron and this will materially reduce the cost of making copper. Considering the successful operation of all the other metallurgical operations, I hardly think that, on a small output of say 15 tons copper per day, the discard of the rest of the plant and the introduction of converters would be justified.

## II. Smelting at the Ikuno Mines

The metallurgical plant at this mine consists of: Concentration plant; Kiss process plant; smelting works. It is only the last that is of interest to us for the present; the following materials are sent to the blast furnaces: Gold and silver ores from the Tasei mine; copper and silver ores from the Kanagase mine; concentrates and the fines from the copper ore, made into briquettes.

The product is black copper assaying: copper 96 per cent.; silver, 0.8 per cent. or 266 oz. per ton; gold, 0.015 per cent. or 5 oz. per ton.

The various ores which are smelted here come from several mines which are within a radius of 10 miles.

The Tasei mine furnishes a brownish colored quartz, which contains sulphide of silver in association with native gold, a small quantity of pyrite, copper sulphides and blende.

The Kanagase mine furnishes quartzose ores with copper sulphides, some galena, blende and pyrite.

The Otsubushi ores contain metallic silver, galena, blende, argentic sulphide, bornite, some antimony, nickel and cobalt. The galena ores do not seem to carry much silver and good samples of tetrahedrite and argentite are frequent in the mine.

The Mikobata mine carries silver ore with gold.

The Nakase mine furnishes silver ores and gold in association with cupriferous pyrites. Native silver is often found with pyrargyrite.

The smelting as carried out at present is pyritic and the fuel is only added in the outer ring of the furnace, when the same does not run regularly.

The ores contain about 17 per cent. of sulphur, of which 9 per cent. burns and acts as a fuel and 8 per cent. is absorbed in the matte. This system was introduced in 1902 or 1903; prior to that period the ore only contained 9 to 10 per cent. of sulphur. As an average of 17 per cent. of sulphur is not sufficient, pyritic ores are brought from outside mines whenever obtainable and cheap enough to leave a profit.

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There are 35 roasting stalls, dimensions 5 by 9 ft., each holding 7 tons. The fuel is mostly pine wood and is used at the rate of 1 per cent. of the weight of ore. The roasting takes one week, but why there is such a large quantity of ore roasted when there is a lack of sulphur, I did not learn.

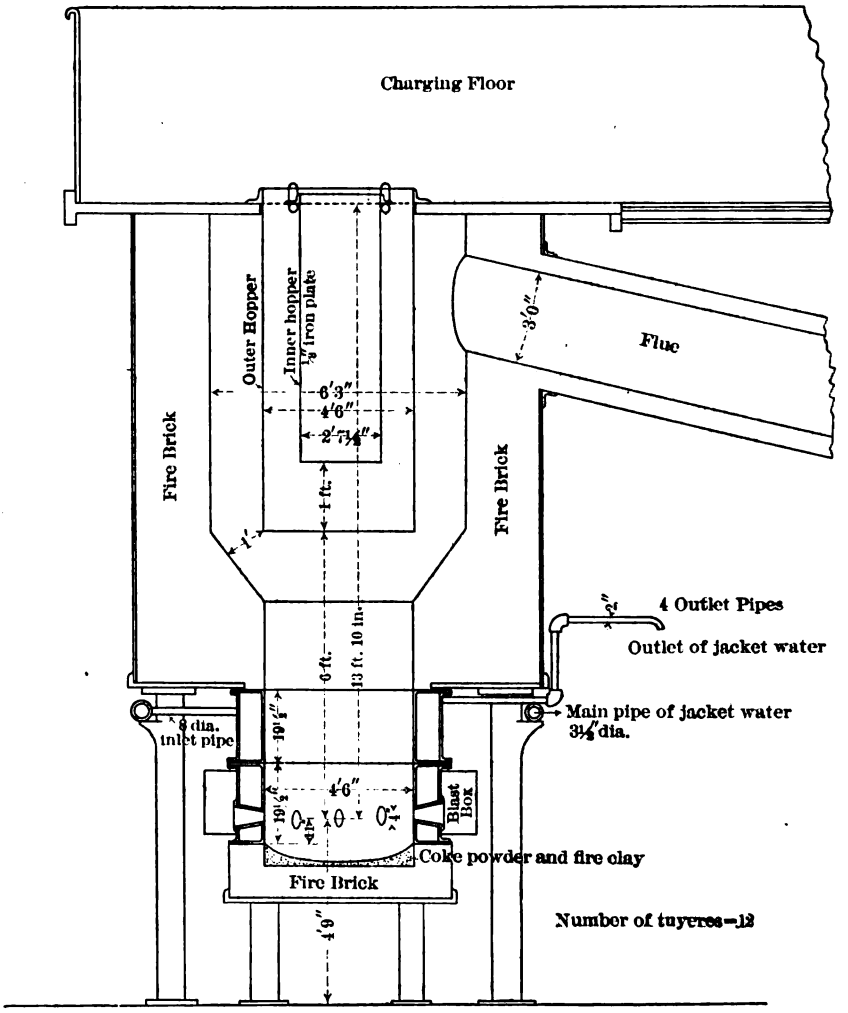


FIG. 1.—SEMI-PYRITIC BLAST FURNACE AT IKUNO WORKS.

The concentrates are made into briquettes by being mixed with 5 per cent. of milk of lime. About 30 women are employed at this work, each woman makes 800 briquettes in 9 hr. or 24,000 are made daily. These briquettes are dried in the main flue leading to the chimney which is on top of the hill. They are cylindrical in shape, 4 in. in diameter

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and 5 in. high. The main flue near the briquetting shed is enlarged and divided by a longitudinal wall into two parts, and when one section is being employed the draft is turned into the other section. These drying chambers are, 30 ft. in length, 9 ft. in height and 8 ft. in width, each holding 30,000 briquettes and the drying is effected in 34 hr.

The ore smelting is done in three cupolas. The No. 1 furnace is 14 ft. high from tuyère to feed floor. The diameter at the tuyères is 52 in. There are eight tuyères. There is an inner circle to this furnace formed of a cast-iron pipe, 30 in. in diameter, which reaches within 6 ft. of the tuyères, consequently this pipe is 8 ft. in length. Into this inner circle are fed coke and ore rich in sulphur, or the fuel portion of the charge.

The furnace is built of two series of water jackets one on top of the other, and discharges into a square forehearth. The spout connecting furnace with forehearth is also water jacketed. The bottom of the furnace is 10 in. below the tuyères. From the first forehearth the slag discharges into another forehearth and then into the slag pots.

The capacity of the No. 1 furnace is 50 tons of smelting mixture. The furnaces are connected with dust chambers built at the back of the works. The percentage of fuel employed is 9 per cent. The matte produced carries about 40 per cent. copper. The cost of smelting is given to me at 6½ yen per ton (equal to 13 shillings) which includes all the roasting and refining charges.

| The furnace charges consist of: | Kilograms |
|---|---|
| Copper ores No. 3 carrying 15 per cent. of copper | 100 |
| Dressed copper ores | 150 |
| Copper ores No. 4 carrying 7 per cent. of copper | 100 |
| Raw briquettes, only dried | 50 |
| Siliceous gold and silver ores | 20 |
| Mabuki-Doko slags | 150 |
| Limestone | 100 |
| Residues from sulphuric acid works (roasted pyrites with some copper) | 290 |
| Matte | 100 |
| Coke | 95 |
| Total | 1,155 |

The matte composition is as follows:

| | Per Cent. |
|---|---|
| Copper | 41.20 |
| Iron | 22.60 |
| Sulphur | 26.40 |

The slag composition is as follows:

| | Per Cent. |
|---|---|
| Silica | 43 |
| Iron | 30 |
| Lime | 27 |

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The ores carry on an average 6 per cent. of zinc. The iron which is lacking is supplied from the Osaka Chemical works which sells to various mines the residues of its roasted sulphides, which contain about 2.5 per cent. of copper, 45 per cent. of iron and 5 per cent. of sulphur; their cost is about 12s. per ton delivered at the mine. The limestone costs 5s., coke 20s., and charcoal 28s. per ton.

The No. 3 furnace has 12 tuyères, and is 54 in. in diameter. Its capacity is 55 tons of smelting mixture. The pressure of the blast is 28 mm.

The charge for the No. 3 furnace consists of:

| | Kilograms. |
|---|---|
| Copper ore, third grade, 15 per cent. of copper | 70 |
| Copper ore, fourth grade, 7 per cent. of copper | 120 |
| Dressed copper ore | 150 |
| Roasted fine ore, in briquettes | 100 |
| Gold and silver ores | 20 |
| Mabuki-Doko slag | 150 |
| Limestone | 100 |
| Roasted pyrites from acid works | 290 |
| Roasted matte | 100 |
| Coke | 100 |
| Total | 1,200 |

The smelting of lead ores is carried out in a separate cupola in admixture with auriferous and argentiferous ores, so as to produce a rich lead bullion, matte and slags. The lead-carrying matte is roasted in separate stalls and resmelted with the lead products from the cupellation furnace. The furnace is generally operated one week in the month. The lead bullion produced in the cupola furnace is sent to the cupellation furnace to produce doré silver, which is sent to the Osaka refinery. The matte from the second smelting is sent to the Mabuki-Doko.

### *The Mabuki-Doko*

The *Mabuki-Doko* method is equivalent to Bessemerizing in a Japanese forehearth. The forehearths are shown in plan and section in Fig. 2.

In the construction of the Mabukis an excavation in the ground about 9 ft. square is made, and lined generally with square blocks of a fire resisting volcanic stone, care being taken to have a drain in the bottom. The cavity is then filled with a mixture of 30 parts of powdered coke, 6 parts of fire clay, 64 parts of burnt clay, and in a moist state is well tamped in. When dry, crucibles are built up of clay and charcoal forming a series of holes 1 ft. 6 in. in depth, with a diameter at bottom of 2 ft. 6 in., and a diameter at top of 2 ft. 11 in. The cover is also built up of clay on a strong wire netting. It requires considerable experience to build up one of these Mabukis, simple as the operation may appear.

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After drying, hot slag is poured in and the crust removed after cooling. A 2-in. iron pipe is built in vertically for escape of moisture. In front of the dome-shaped cover is a 5-in. opening to feed matte and fuel. The air blast is introduced on one side only on top of the charge and the tuyère is bent downward. The matte is poured in in a molten state and char-

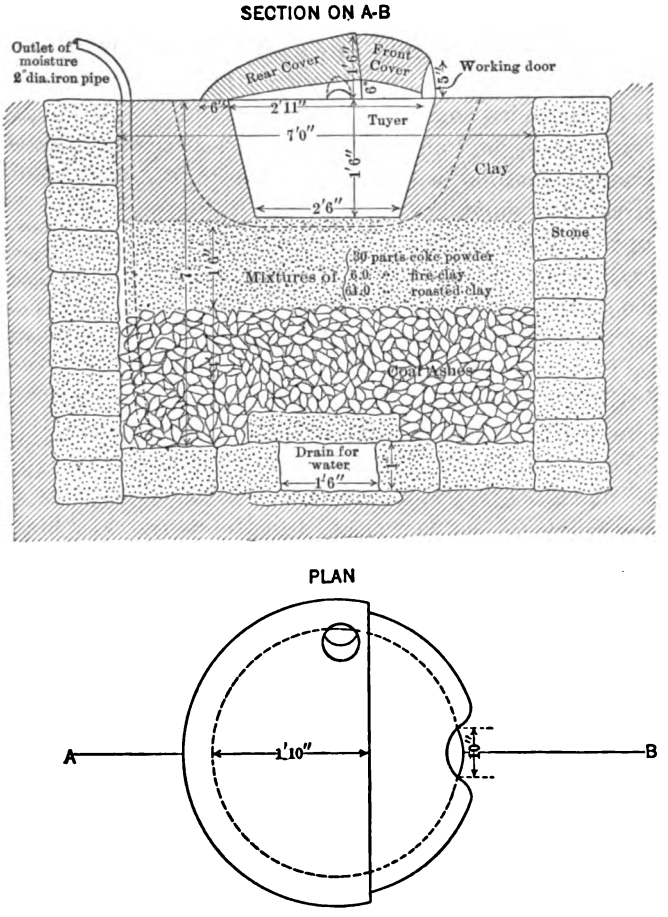


FIG. 2.—SKETCH OF MABUKI-DOKO OR JAPANESE FOREHEARTH.

coal is placed on top; at some mines coke is employed for this purpose. The slag is skimmed every 15 minutes. The idea of introducing the molten matte into the Mabuki originated here, as formerly it was the practic all over Japan to introduce it in lumps and fuse it first.

There are six Mabukis here and the claim is that the loss of copper in this operation is small. The evaporation of the zinc oxide causes silver losses and the claim is that it is recovered in the flues. The copper and

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silver diminish in the flue dust, the further the same is collected from the smelting and refining department.

The cost of treatment in the Mabuki is given at 2.5 yen, or 5s. per ton of matte. There are smelted 3 tons of matte daily in each hearth, and generally three of them are in operation dealing with 9 tons. I am told that 6 per cent. of this weight is consumed in charcoal, or only a little over half a ton daily, which seems rather low. As the matte contains 23 per cent. of iron and each forehearth treats 6,000 lb. daily, this would give 1,380 lb. of iron which had partly to be oxidized and partly slagged off by the silica contained in the clay lining. Although the slags produced are very basic, I am mystified at the successful operation of this strictly Japanese method, as the crucibles are not eaten away as fast as it would seem they should.

The cupels of the cupellation furnaces are made of cement. Some iron is introduced during cupellation to absorb sulphur from the press cakes. The slagged material is drawn out through the working doors. The daily capacity of the cupellation furnace is 1.5 tons; one hearth will treat 5 tons before requiring renewing.

The flue dust is collected in a flue 1,560 ft. in length, built on the slope of the mountain. At the point where the flue connects with the chimney, the vertical distance is 600 ft. above the smelting floor level. The dimensions of the flue are 4 ft. wide by 5 ft. high. The chimney is 30 ft. high. In the upper portion, the flue is cleaned twice a year. The lower portion, between the Mabuki-dokos and the briquette drying chambers, is cleaned once a month. The quantity of dust collected in the flue is 0.6 per cent. of the quantity of ore treated. Near the smelting works are dust chambers which are cleaned every two weeks, 5 per cent. of the quantity of ore treated being collected in this section. The flue dust from the Mabuki furnaces contains 4 dwt. of gold, 40 oz. of silver and 24 per cent. of copper; the dust from the chambers near the ore smelting furnaces contains 1 dwt. 8 grains of gold, 5 oz. of silver and 1.5 per cent. of copper. Experiments to ascertain these values and quantities were carried out during two months. The flue connecting the Mabuki-dokos with the main flue is 200 ft. in length.

The production for 1905 was 850 tons of copper, 7,300 oz. of gold, and 200 oz. of silver.

The machinery in the smelting works includes two Baker blowers operated by a 4-ft. Pelton wheel at a water head of 100 ft. and one Root blower.

A 12-in. pipe conveys the air to a reservoir of boiler iron, and from it a 14-in. main takes it to the furnaces to be distributed by 10-in. pipes. The angles of these pipes are of cast iron. There are three Chilean mills for grinding the ash which is used in the Mabukis.

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*Analysis of Ores Treated at the Tkuno Works*

| | Au, Oz. | Ag, Oz. | Cu, Per Cent. | Pb, Per Cent. | Zn, Per Cent. | Fe, Per Cent. | S, Per Cent. | CaO, Per Cent. | $SiO_2$, Per Cent. |
|---|---|---|---|---|---|---|---|---|---|
| Gold and silver ores from Tasei mine | 8.8 | 22.4 | 0.045 | 0.01 | Trace | 4.966 | 4.739 | 1.730 | 85.70 |
| Gold and silver ores from Kasei mine | 0.7 | 88.0 | 1.870 | 0.57 | ..... | 6.730 | 3.440 | 4.140 | 75.85 |
| Copper ore, 3d grade | .... | 23.0 | 16.400 | 1.57 | 6.21 | 16.640 | 25.400 | 2.910 | 30.46 |
| Copper ore, 4th grade | .... | 10.0 | 7.000 | 1.01 | 11.59 | 7.620 | 11.700 | 3.890 | 67.56 |
| Concentrates from copper ore | .... | 11.667 | 7.900 | 2.11 | 14.70 | 13.500 | 15.100 | 1.100 | 49.23 |
| Raw briquettes | 0.25 | 14.0 | 3.400 | 2.70 | 13.927 | 15.680 | 16.900 | 5.397 | 40.36 |
| Roasted briquettes | 0.3 | 16.0 | 3.680 | 2.81 | 10.309 | 17.020 | 8.200 | 6.111 | 42.84 |
| Slag from ore smelting | .... | 0.7 | 0.412 | ..... | 5.770 | 22.830 | ..... | 7.500 | 42.20 |
| First matte from ore smelting | 1.7 | 78.0 | 41.200 | 5.50 | 4.200 | 22.600 | 26.400 | ..... | ..... |
| Residues from sulphuric acid works | .... | ..... | 2.600 | ..... | ..... | 57.420 | 5.300 | ..... | 7.50 |
| Limestone | .... | ..... | ..... | ..... | ..... | ..... | ..... | 50.900 | 2.65 |
| Mabuki slag | 0.15 | 10.0 | 5.700 | 2.40 | 2.60 | 42.10 | 6.740 | ..... | 22.40 |

## III. Smelting at the Ashio Works of the Furikawa Mining Co.

The mines occur in eruptive rocks, such as andesite and trachyte, which have here found a vent through the Archean and Paleozoic rocks, the greater portion granite, which surround on all sides the cupriferous zone embracing the complete system of the Ashio copper veins.

The first-class ores as they come from the mine contain 13 per cent. of copper, and they are enriched by hand picking to 14 per cent. The second-class ores contain on an average 1.5 per cent. and are enriched by concentration to 10 per cent.

There are produced daily on an average 130 tons of first-class ore, which is sorted down to 111 tons; the 450 tons of low-grade ore produce 34 tons of concentrates assaying 10 per cent. of copper. The total sent daily to the smelter is therefore 145 tons. Hence it takes 13.24 tons of low-grade ore to make 1 ton of concentrates and the following calculation shows that the loss in concentration is 50 per cent.:

| | Metallic Copper, Tons |
|---|---|
| 450 tons of ore, 1.5 per cent. copper, contain | 6.75 |
| 34 tons of concentrates, 10 per cent. copper, contain | 3.40 |
| Loss in concentration | 3.35 |

The ore going to the smelter averages 13 per cent. of copper and therefore the daily production should be 18.85 tons copper, minus the loss in treatment.

The proportion of mine ore treated daily is:

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| | Tons |
|---|---|
| Lump ore, 15.7 per cent. Cu | 61.0 |
| Grain ore, 13.3 per cent. Cu | 21.0 |
| Fine ore, 11.3 per cent. Cu | 61.0 |
| Pulp ore, 6.0 per cent. Cu | 2.5 |
| Cement copper, 50.0 per cent. Cu | 1.0 |
| Total | 146.5 |

Pulp ore is obtained from settling ponds connected with the concentration plants, where the richer portions of the slimes are collected.

### *Six Blast Furnaces in Operation*

There are six smelting furnaces with dimensions as follows:

No. 1. 17 ft. long by 40 in. wide, with 24 6-in. tuyères.
No. 2. 8 ft. long by 36 in. wide, with 14 4-in. tuyères.
No. 3. 8 ft. long by 36 in. wide, with 14 4-in. tuyères.
No. 4. 8 ft. long by 36 in. wide, with 12 3.5-in. tuyères.
No. 5. 67 in. long by 36 in. wide.
No. 6. 67 in. long by 36 in. wide.

The wind pressure at the furnaces is 35 mm. The feed floors are 8 ft. above the tuyère levels. The main air pipe is 36 in. in diameter. Five Root blowers are driven by a Pelton wheel, with a water pressure of 63 lb. per square inch; they work at a wind pressure of about 40 mm. In winter the blowers are driven electrically. It takes 110 h.p. to run the five blowers. A vertical air compressor is also operated by a Pelton wheel and requires 134 h.p. to furnish air up to the required pressure. The compressed air is required for Bessemerizing. There is a separate Root blower for No. 2 furnace.

The fine concentrates are made into briquettes. These briquettes, while damp, are covered with matte by a patented machine invented by Mr. Kondo. On an endless chain working horizontally is a series of small cast-iron pots. Into these the briquettes are placed and passed under a stream of matte. The iron cups are washed with lime water to prevent adhesion of the molten material.

The furnaces discharge into a square forehearth of wrought iron lined with blocks of liparite. The forehearth is divided in two unequal compartments by a water-jacketed partition, which does not reach to the bottom. The matte passes under this partition into the smaller compartment and runs into molds fitted on a small platform car, each car holding four molds. The lining of the forehearth lasts two months and the furnace crucible three months.

The charge for No. 4 furnace is 1,000 lb. of ore to which are added: Limestone, 230 lb., or 23 per cent.; converter slag, 200 lb., or 20 per cent.;

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raw matte, 100 lb., or 10 per cent.; and coke, 130 lb., or 13 per cent. This furnace is used for pyritic smelting. About 50 charges are treated in 24 hr. the capacity being 25 tons of raw ore or 45 tons including fluxes and coke. The coke consumption is 7 per cent. The matte contains 34 per cent. of copper and the yield or extraction is about 98 per cent. of the copper contents of the ore.

Furnace No. 2 is for roasted ores. The discharge into the forehearth is on the long side of the furnace. The air is led into a wind box made of the iron girders which support the upper portion of the furnace. The crucible is 14 in. deep. The slag from this furnace is not granulated, but made into plate slag to be used as flux. The capacity in roasted ore is about 28 tons. The charge is as follows:

| | Lb. |
|---|---|
| Roasted ore | 900 |
| Raw matte from No. 4. furnace | 413 |
| Briquettes from roasted fines | 413 |
| Coke | 130 |
| Total | 1,856 |

The capacity is 50 charges daily or a total of about 45 tons. The coke consumption with roasted ores is 13.3 per cent.

*Furnace No* 1 is the largest of the six. The air pressure is 30 mm. The air pipe around the furnace is 22 in. in diameter and main air conduit 36 in. in diameter. The depth of the crucible in front where the discharge takes place into the forehearth is 18 in. and at the back 12 in. Smelting was started with a forehearth on each end, but the matte fall was not sufficient to keep up a steady flow so the back forehearth was abandoned. The matte produced in this furnace contains 31 per cent. of copper. It is poured on to the floor from the pots, cooled by water and resmelted.

The coke consumption in this furnace on the total charge is 8.2 per cent. The charge is composed as follows:

| | Lb. | |
|---|---|---|
| Raw ore | 6,600 | |
| Limestone | 1,552 | or 25 per cent. based on ore |
| Converter slag | 1,321 | or 20 per cent. based on ore |
| Matte | 660 | or 10 per cent. based on ore |
| Coke | 925 | or 9 per cent. based on ore |
| Total | 11,058 | |

The capacity is 23 charges in 24 hr., 76 tons of ore or about 129 tons of mixture, being smelted daily.

*Furnaces Nos.* 5 *and* 6 have a capacity each of about 21 tons, ore and mixture. The spouts from the forehearths are made of blocks of liparite, covered with a hemispherical dome.

The cost per ton of concentrates and ore is estimated at 36.5 yen,

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and as 7.63 tons of ore make one ton of copper, the mining and concentrating charges per ton of copper are 278.49 yen. The daily mining and concentrating expense would be 5,291.31 yen; and the cost per ton, based on the 528 tons raised, is 10.02 yen. The average value of the 528 tons of ore hoisted is 23 yen per ton. The cost of producing a long ton of metallic copper is given as 90.78 yen; the cost of mining and concentrating to produce a ton of copper as above, 278.49 yen or a total cost of 369.27 yen per ton, or about £37, but I think that these figures do not include general and administrative charges, nor the electrolytic refining, which is being done at Nikko, about 25 miles from the mine. The cost of coke is 26.70 yen per ton.

*Bessemerizing in Converters*

The converters are lined first with blocks of liparite, which are 8 in. square, and then decomposed liparite is plastered over them. The exterior diameter of the converters is 5 ft. The bottom lining is 12 in. thick and the top lining 7 in. The lining lasts 6 to 7 hr. and during that time 10 tons of matte is blown and as it contains 38 per cent. of copper, this should give 3.8 tons of copper. But there are produced at each blow only 2.9 tons of copper so that the rest, 0.9 ton, goes into the slag, skimmings, flue, etc.

For an output of 19 tons daily, there must be produced from the 146 tons of ore smelted daily, about 50 tons of matte. The matting furnace is circular. It has ten 3-in. tuyères and works at 22 mm. pressure. The crucible under the water jackets is on wheels and is tapped when matte is required for the converter. The charge in the furnace is: Matte, 660 lb; converter lining, 132 lb. or 20 per cent.; coke, 52 lb. or 8 per cent.

The remelting cupola smelts 50 tons of matte daily. The ladles for holding the copper are lined with powdered charcoal and clay (*Subai*). Slag is poured into these ladles to dry them. The ingots weigh 51 lb. A separate Root blower is used for the remelting cupola. In the converter the air pressure employed is 10 lb. per square inch. There are four converters in the building; while two are in use, two undergo repairs. The cost of bessemerizing, including smelting the matte, is 17 yen, £1 14s. per ton.

*Cement Copper.*—There are 10 large cement tanks through which the mine waters pass under similar conditions as explained for the Besshi mine. The cement copper is briquetted.

The water issuing from the water jackets is used for granulating the slag. The total quantity of water required by the six water jackets is 30 cu. ft. per minute. From the hollow box girders around the furnaces the iron bustle pipes connect with the tuyéres.

The limestone used for fluxing purposes is quarried about 4 miles distant from the mines and the quarried material is sent down by an

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aerial ropeway to a dumping place, where it is broken up by hand labor and then sent by horse traction on a tramline to the smelter.

The dimensions of the main flue are 6 ft. in width by 11 ft. in height. Before connecting with the chimney the smoke has to pass through a desulphurizing tower. The temperature of the gases and fumes is 200° C., before entering this tower and on leaving only 87° C. This of course impedes the draft and a Guibal fan is placed before the tower to overcome this defect. In this tower a shower of lime water absorbs the sulphurous gases. The water passes through a filter after issuing from the tower so as not to vitiate the river water. About 25 per cent. of the sulphurous gases are absorbed.

The waste rock from the concentration works and the granulated slag are carried by aerial ropeways across the mountain range into another valley and there impounded by large dams, so that the storm water shall not wash the fines on to the rice fields below.

*Analyses of Ores, Ashio Mine*

| | Lump Ore, Per Cent. | Grain Ore, Per Cent. | Fine Ore, Per Cent. | Pulp, Per Cent. | Cement Copper, Per Cent. |
|---|---|---|---|---|---|
| Copper | 18.23 | 13.20 | 12.00 | 5.09 | 50.00 |
| Iron | 23.00 | 30.00 | 23.10 | 11.40 | 14.60 |
| Sulphur | 26.70 | 28.40 | 25.50 | 8.90 | 0.09 |
| Silica | 29.50 | 21.40 | 33.60 | 53.60 | 12.07 |
| Aluminum | 2.20 | 5.80 | 6.50 | 10.50 | ..... |
| Lime | 2.50 | 1.30 | 1.30 | 1.60 | ..... |
| Arsenic | 0.19 | 0.55 | 0.25 | 0.12 | 0.04 |
| Zinc | 0.33 | 0.65 | 0.63 | 1.06 | 0.06 |

*Analyses of Mattes, Ashio Works*

| | Matte from Raw Ore | Matte from Roasted Ore |
|---|---|---|
| Copper, per cent | 35.84 | 42.78 |
| Iron, per cent | 36.80 | 31.24 |
| Sulphur, per cent | 26.25 | 24.14 |
| Arsenic, per cent | 0.45 | 0.58 |
| Gold, grams | 1.00 | 1.00 |
| Silver, ounces | 30.00 | 30.00 |
| Specific gravity 4.96 | ..... | ..... |



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*Analysis of Slag, Ashio Works*

| | Per Cent. | | Per Cent. |
|---|---|---|---|
| Copper | 0.36 | Magnesia | 1.71 |
| Silica | 39.80 | Sulphur | 1.30 |
| FeO | 35.67 | Zinc | 1.44 |
| $Al_2O_3$ | 11.75 | Specific gravity 3.51 | .... |
| Lime | 11.57 | | |

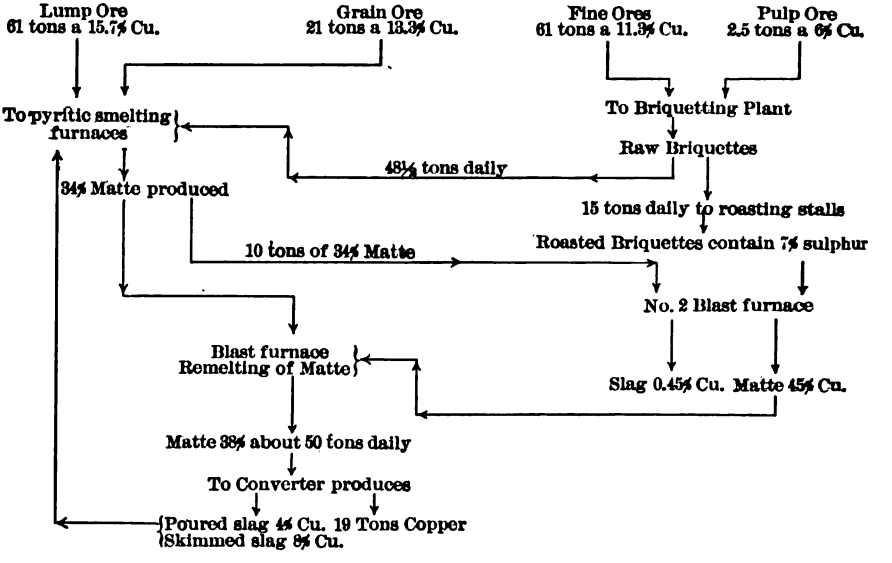


FIG. 3.—SCHEME OF TREATMENT AT ASHIO SMELTING WORKS.

*Analysis of Converter Copper, Ashio Works*

| | |
|---|---|
| Gold, dwt | 0.208 |
| Silver, oz | 35.000 |
| Nickel and cobalt, per cent | 0.0249 |
| Selenium, per cent | 0.119 |
| Tellurium, per cent | 0.040 |

The percentage of copper is not stated.

*Analysis of Speiss, Ashio Works*

| | Per Cent. |
|---|---|
| Copper | 62.800 |
| Iron | 7.725 |
| Arsenic | 14.187 |
| Antimony | 0.488 |

In the bottom of the forehearth speiss collects occasionally, rich in gold and silver.

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*Analysis of Converter Slag, Ashio Works*

| | Per Cent. |
|---|---|
| Silica | 19.86 |
| Iron oxide | 60.05 |
| Copper | 4.13 |
| Alumina | 7.81 |
| Sulphur | 1.26 |
| Zinc, | 0.95 |
| Magnesia | 0.87 |
| Lime | 0.60 |

*Analysis of Converter Lining, Decomposed Liparite, Ashio Works*

| | Per Cent. |
|---|---|
| Silica | 74.47 |
| Alumina | 14.72 |
| Lime | 2.86 |
| Magnesia | 1.03 |
| Other alkalies | 1.31 |
| Iron oxide | 1.06 |

## IV. The Kosaka Mine of the Fujita Company

The deposit of ore at the Kosaka mine is evidently of volcanic origin; it occurs near the junction of the tuffa and andesite. In its widest portion the orebody is 780 ft. wide and is proved for 2,000 ft. in length. The whole mass is mineralized.

A characteristic feature of this mine is that it contains three classes of ore. These form three distinct zones, and it seems as if nature had so prepared them, that the metallurgist by their proper commingling should have his furnace charges without having to draw his fluxing materials from outside sources. All ores containing under 1 per cent. of copper are left standing in the mine as they are not considered valuable.

The black ore or *Kuromono* forms the eastern portion of the orebody and is the richest section of the mine. The silver and gold content increases with the rise in the percentage of copper in the ore. The vein material of this ore is chiefly heavy spar with small proportions of gypsum and quartz. The ore averages 2.5 per cent. of copper. The yellow or pyritic ore forms a band around the black ore on the western side and averages 2 per cent. when extracted for smelting. The largest part of the deposit on the western side is siliceous ore and averages when extracted 1.8 per cent. of copper. There is no zinc in this part, nor in the yellow ore, but the black ore contains over 13 per cent. of this metal.

In ancient times the Kosaka was worked as a silver mine and the following method of treatment was carried out there prior to 1868. The ore was broken into 1-in. cubes, and roasted with charcoal on a hearth about 2 ft. in diameter and 1.5 ft. deep. The latter was dug in the ground and lined with a mixture of slag and powdered charcoal. While roasting,

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a light blast from a Japanese bellows was turned on, which was increased as the roasting advanced. Argentiferous lead collected in the bottom of the well and was ladled out. The small ingots thus produced were smelted on a shallow hearth made of wood ashes, a light blast turned on to oxidize the lead, which was removed as litharge; a silver slab remained behind. The matte and slag produced in the first operation, on account of the large percentage of zinc they contained, could not be utilized.

In 1870 Mr. Oshima introduced the following method:

1. The black ore was broken into 1-in. pieces and heap roasted.

2. The roasted ore and some earthy ore were mixed in certain proportions and smelted with charcoal, sand and slag in a blast furnace. The molten matte was ladled into a kettle and mixed with molten lead which had been smelted in another hearth. The mixture was well stirred and as the matte solidified the round disks were taken off. The argentiferous lead in the bottom was ladled out into molds.

3. The matte desilverized as above was broken into 2-in. pieces and roasted in heaps.

4. The roasted matte was smelted in another hearth furnace together with suitable proportions of litharge and slag and argentiferous lead regulus and a matte rich in copper obtained. This matte was roasted and smelted for crude copper, the resulting matte being again worked in the same manner.

5. The argentiferous lead regulus was treated by the English cupellation process.

*Present-Day Metallurgical Treatment*

There are smelted 1,200 to 1,300 tons of ore daily, or about 450,000 tons a year. There are produced 7,200 tons of copper per year showing an extraction of 1.66 per cent. of copper per ton of ore, therefore it takes 61.66 tons of ore to make a ton of copper. The smelting and refining charge is given as 4s. per ton of ore, or £12 0s. 8d. per ton of copper; the cost of mining at 3.2s. per ton of ore, or £9 1s. 8d. per ton of copper, making the total cost £21 2s. 4d. per ton of copper.

The smelting and mining charges seem small, especially since I am told that 12,000 people besides 400 officials are employed on the property. It is nevertheless quite possible that copper is being made here at about £35 per ton. The smelting cost for the first matte is given to me at 1s. 9d. per ton. The extraction on 2 per cent. ore is 83 per cent. There are seven smelting furnaces in this establishment.

1. Furnace, 60 ft. long by 3 ft. 7 in. wide, smelts 375 tons of ore daily.
2. Furnace, 19 ft. long by 4 ft. wide,
3. Furnace, 25 ft. long by 4 ft. wide,
4. Furnace, 25 ft. long by 4 ft. wide,
5. Furnace, 25 ft. long by 4 ft. wide,
6. Furnace, 25 ft. long by 3 ft. 4 in. wide,

smelt 875 tons of ore daily, or 175 tons per furnace.

7. Furnace, 25 ft. long by 3 ft. 4 in. wide, smelts 110 tons of matte daily.

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The blast pressure employed in the furnaces is 36 to 40 mm. The 25-ft. furnaces are 8 ft. high to flue bottom and 11 ft. 6 in. to charging floor. The 60-ft. furnace is 11 ft. high to flue bottom and 19 ft. 6 in. high to charging floor. There are eight Root blowers, operated by electricity, and requiring 400 h.p.

The furnaces discharge into semicircular forehearths of cast iron with a matte collecting trap, from which the matte discharges upon revolving tables. The 25-ft. furnaces have 15 6-in. tuyères on each side and two tuyères at the back. The wind pressure used is 35 mm. A series of dust chambers connects with the smoke stack, which is 200 ft. high and 16 ft. internal diameter. The diameter of the flues leading to dust chambers is 8 ft. The main air pipe is 6 ft. in diameter, and the branch pipes leading to these furnaces are 3 ft. 6 in. in diameter.

Bituminous coal to the extent of 2 per cent. is introduced through the tuyères. The 60-ft. furnace has 80 6-in. tuyères arranged in three tiers, so as to conform with the inside sloping bottom of the furnace.

Each of the 25-ft. furnaces requires 3,500 gal. of water per hour for cooling the jackets. The 60-ft. furnace is provided with a hydraulic ore-feeding device.

The mine ore after passing through grizzles is classified into lump ore and fines; the latter are hand sifted through 0.5-in. screens, the undersize being briquetted by means of wooden stamps. The daily production of briquettes is 150 tons. These are not furnace dried, but go direct to the smelter.

The mine ore is sent to the furnaces in the following proportions: Black ore, 5 tons; siliceous ore, 3 tons; yellow ore, 2 tons; total, 10 tons.

The siliceous ore is poor in gold and silver and contains 10 per cent. of sulphur. The average contents of the three ores are 24 per cent. of sulphur and 5 and 6 per cent. of zinc. As a metallurgical undertaking, I consider that the Kosaka mine is a difficult problem when the large percentage of baryta is considered. To the ability of Mr. Takeda and his able assistant, Mr. Takanuchi, is due the success of pyritic smelting introduced here by them.

The ore beds for the ore-smelting furnaces are composed as follows:

| | Pounds |
|---|---|
| Black ore | 4,300 |
| Pyritic ore | 1,000 |
| Siliceous ore | 2,700 |
| Briquettes raw | 2,700 |
| Slags from second matte smelting | 2,000 |
| Total | 12,700 |

The matte fall on an average is 9 per cent. of the quantity of ore

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smelted or 112.5 tons per day, containing about 30 per cent. of copper. The first matte is reconcentrated in a special furnace, and the second matte contains 45 to 50 per cent. of copper.

The slags produced average 0.33 per cent. of copper. No coke or fuel is introduced into the furnaces at the feeding floor. The slags from the ore-smelting furnaces are granulated.

No. 7 furnace, which is used for matte concentration, is situated in a separate building. In the matte smelting 40 per cent. of siliceous mine ores and 1.5 per cent. of coke are added and 2.5 per cent. of bituminous coal is introduced through the tuyères. The slags produced here go back to the ore-smelting furnaces. The concentrated matte averages 50 per cent. of copper.

The second matte is crushed in rock breakers and then goes to Krom rolls which have a daily capacity of 30 tons. The pulverized material is roasted in Herreshoff furnaces down to about 4 per cent. of sulphur. There are 10 of these furnaces erected but only six are in operation. Each furnace has a daily capacity of 5 tons, so that they handle the 30 tons of second matte produced every day. Cord wood is used as fuel and 10 per cent. of the weight of ore roasted is used. The roasted ore is smelted in reverberatories to bottoms and white metal.

*Smelting for White Metal.*—For this purpose two reverberatory furnaces are employed. One 25 ft. long and 16 ft. wide, outside dimensions, the bridge plate of which is water jacketed; the other 20 ft. long and 12 ft. wide. The product here is white metal with 70 per cent. of copper and copper bottoms. It takes 4 hr. to melt a charge. The slag produced contains 4.87 per cent. of zinc.

*Blister Copper.*—From the two reverberatory furnaces the cold matte goes to the blister furnaces, of which there are two, and the blister copper produced goes to two refining furnaces. The copper from these is cast into anode plates which weigh 200 lb. The copper bottoms from No. 2 furnace are cast. In the same building are two furnaces for drying the mud or slime from the electrolytic work.

All the slags and skimmings produced in the reverberatories, liquation, blister furnaces, etc., go to the slag-smelting furnace.

*Slag-smelting Furnace.*—In the same building with the matte concentration furnace is a furnace for melting the slags produced in the reverberatory furnaces. This blast furnace is 10 ft. high and 3 ft. 4 in. wide, having a capacity of 65 tons.

In the slag-smelting furnace the charges are composed of: Slag, 100 lb.; scrap iron, 16 lb.; siliceous ores, containing gold, from Matsuoka mine, 13 lb.; coke, 15 to 20 per cent.

In this operation metallic lead collects in the bottom with a lead and copper matte on top; this separation is effected in a forehearth. The matte which is produced contains 10 per cent. of lead and copper. There

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are produced 42 tons of lead monthly. The lead bars are liquated in five Japanese furnaces. The liquated lead after being enriched by the Parkes process is smelted in small reverberatories with the slimes produced in the electrolytic refinery. The bullion then goes to two English cupellation furnaces. Before the lead is cupelled, it is enriched by the Parkes process, two kettles being used for this purpose. The zinc employed is distilled in a special furnace with a retort attachment. The rich lead only is cupelled; the poor lead is sold. The lead-copper matte goes back to the matting furnace.

The electrolytic refinery is equipped with four continuous current generators, manufactured by the General Electric Co., of 220 h.p. each, operating at 75 volts and 2,000 amperes with a speed of 360; each machine is provided with an exciter. In the refinery are 500 tanks arranged in 10 rows of 50 tanks each. Each tank holds 20 anode plates, each of which weighs 200 lb., as stated.

It takes 30 days for the anode plates to dissolve and the daily production is 20 tons of cathode plates or 600 tons monthly. These plates weigh 20 lb. each. There is a special department for the purification of the electrolyte.

*The Value of Ores and Production*

| | £ | s. | d. |
|---|---|---|---|
| Average assay value, copper, 37 lb. at £60 per ton. | 0 | 22 | 0 |
| Average assay value, silver | 0 | 6 | 0 |
| Average assay value, gold | 0 | 1 | 6 |
| Total assay value of ore | 1 | 9 | 6 |

The monthly production as given to me amounts to:

| | |
|---|---|
| 600 tons of copper at £60 per ton | £36,000 |
| 93,750 oz. of silver at 2s. per ounce | £ 9,375 |
| 750 oz. of gold | 3,000 |
| Total | £48,375 |

For the 37,000 tons smelted this would show a return of £1 6s. per ton, or a difference of 3s. 6d. per ton on the ore valuation. These figures also show that each ton of copper contains £20 worth of gold and silver and that the precious metals play a very important rôle in the profit earning capacity of this mine. The total costs per ton of ore treated are 11s. 4d. and the profit 14s. 8d. per ton, so far as I know the best on record in copper smelting. This does not include depreciation nor the cost of marketing the copper.

All machinery is driven by motors and in the various smelting departments 1,550 h.p. are employed. In the mine and in the other depart-

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ments 1,450 h.p. are utilized. The generating station, which is 6 miles from the mine, develops over 3,000 h.p.

*Analyses of Ores, Matte, Slags, Copper and Byproducts, Kosaka Mine*

| | Copper, Per Cent. | Lead, Per Cent. | Iron, Per Cent. | Zinc, Per Cent. | Silica, Per Cent. | Alumina, Per Cent. | Barium Sulphate, Per Cent. | Sulphur, Per Cent. | Bismuth, Per Cent. | Arsenic, Per Cent. | Antimony, Per Cent. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Black ore | 2.44 | 2.18 | 14.41 | 11.36 | 4.57 | 4.17 | 38.06 | 20.82 | ..... | ..... | ..... |
| Sulphide ore | 3.21 | 0.58 | 27.69 | 5.20 | 10.59 | 4.08 | 14.30 | 33.00 | ..... | ..... | ..... |
| Siliceous ore | 1.72 | 0.17 | 18.69 | 1.20 | 49.01 | 4.90 | 4.51 | 18.92 | ..... | ..... | ..... |
| Briquettes | 2.89 | 0.48 | 26.30 | 3.51 | 14.74 | 8.30 | 10.60 | 29.90 | ..... | ..... | ..... |
| First matte | 30.35 | 5.48 | 28.03 | 8.18 | 0.95 | 0.53 | 0.11 sulphide | 24.97 | ..... | ..... | ..... |
| Ore slags | 0.33 | 0.48 | 25.35 | 7.90 | 36.00 | 5.90 | ..... | 0.99 | ..... | ..... | ..... |
| Second matte | 49.83 | 7.57 | 12.04 | 4.00 | 0.50 | 0.50 | ..... | 23.65 | ..... | ..... | ..... |
| Matte slag | 0.74 | 2.42 | 33.11 | 7.82 | 33.06 | 4.20 | ..... | 0.64 | ..... | ..... | ..... |
| Copper bottoms | 96.93 | 1.25 | 0.015 | 0.023 | ..... | 0.009 metallic | ..... | 0.65 | 0.02 | 0.046 | 0.033 |
| Anode copper | 98.96 | 0.106 | 0.003 | trace | ..... | 0.002 metallic | ..... | 0.01 | 0.036 | 0.04 | 0.05 |
| Electrolytic copper | 99.966 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 0.003 | 0.0028 | 0.003 |
| Electrolytic slimes | 4.085 | 11.837 | 0.156 | 0.870 | 5.610 | 3.230 | ..... | 3.65 | 2.009 | 2.059 | ..... |

The first matte from ore smelting contains 16.48 per cent. of barium oxide and the slag from matte smelting 5.8 per cent.

The specific gravity of black ore is 4.132; siliceous ore, 3.289; first matte, 4.762; first slags, 3.676.

## V. Smelting at the Kano Works

The ore deposit at the Kano mine occurs in liparite and as far as present developments permit to judge, the ore formation took place near the line of contact with the schist rocks, which seem to form the hanging wall of the deposit. Therefore it is possible that future explorations may prove it to be a contact deposit. It is evident that the mineralization of the liparite took place by replacement or metasomaticism and that the orebody was not formed by the gradual filling in with mineral matter of a large cave or crevice. Subsequent earth movements exerted a crushing effect on the mineralized portion, which is evidenced by the large amount of brecciated material met with.

This mineralization took place along a wide zone, which, however,

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narrows down in depth; reconcentration acting from the surface downward has been the cause of the formation of large lenses of enriched material.

The principal minerals found are iron pyrites, copper sulphides, blende, galena with some silver ores, whose nature has not been determined yet, and barite. The richest portion of the mine is that in which zinc predominates, forming lenses of black ore. In its main mineralogical features this deposit bears a close resemblance to the Kosaka mine, but the ore down to the 250-ft. level is not so rich. The open-cast workings show the orebody for 540 ft. in length and an average width of 300 ft., but numerous shallow pits prove that the orebody is over 1,000 ft. in length. In the absence of proper developments, however, no judgment can be formed as to the ore value in the unexplored portions. I consider that all ores over 1 per cent. in copper pay to work.

Owing to an insufficiency of iron in the surface ores, fluxing material has to be drawn from outside sources, but no doubt when the undecomposed or sulphide zone is reached, this defect will remedy itself and the mine ores will become self fluxing, as is the case with the Kosaka mine. Owing to the larger percentage of silica and earthy materials it has been possible to adopt, with the siliceous ores that predominate here, a system of concentration, which, in spite of the losses coincident with this process, effects a great saving as against the cost of direct smelting without concentration; the large quantity of iron required to slag the silica would more than likely make direct smelting prohibitive.

The present concentration plant shows a loss of 37 per cent., whereas when the new plant with better appliances is in operation this loss should be reduced to 30 per cent.

In regard to the smelting operations, I believe that the loss of about 14 per cent. in the smelting and refining department can be considered as normal, in view of the difficult character of ores dealt with, as they contain a much larger quantity of zinc than the Kosaka ores and at the latter mine after several years of experience a loss of 15 per cent. in the smelting and refining operations is acknowledged. On an average the ores going to the furnaces at Kano contain 12 per cent. of zinc, whereas at Kosaka they contain 7 per cent.

Considering that in the new plant 104,220 tons of ore will go to the smelter annually containing 12,506 tons of zinc, besides the zinc contained in the tailings from the concentration works, it is a matter of great regret that no process has been discovered by which at least a portion of this valuable metal can be recovered.

### *Ore Treatment*

For 18 months, during the construction of the new plant, there were extracted daily from the mine 150 tons of ore. To the furnaces are sent

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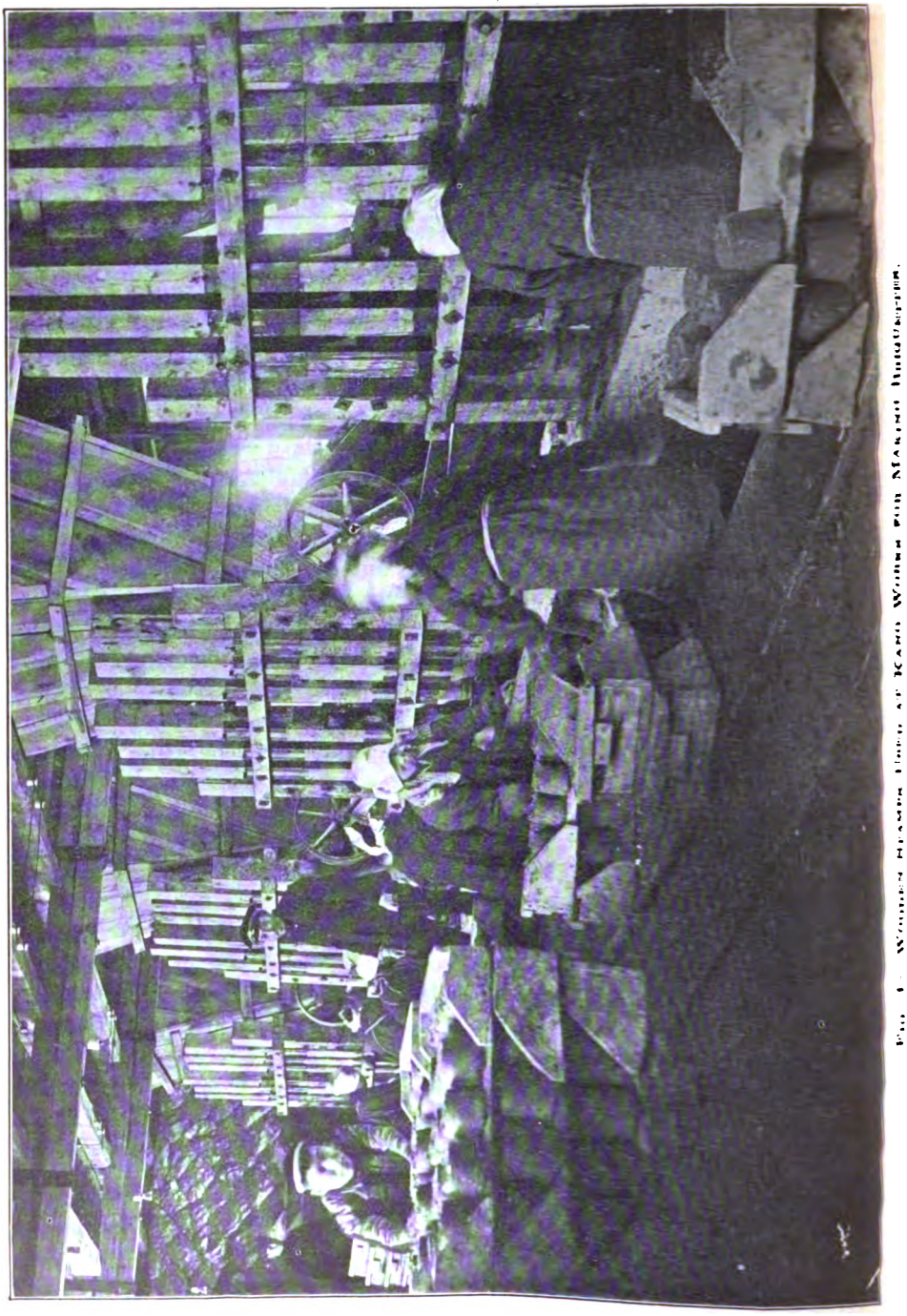

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direct 40 tons of lump ore and to the concentrators, 110 tons. The concentrator produces 30 tons of heading and 15 tons of middling making 45 tons of concentrate sent to the smelter. The total treated by the smelter is, therefore, 85 tons per day.

The concentrate is mixed in pug mills with clay ores from the mine, and by means of wooden stamps made into briquettes, as shown in Fig. 4.

As roasted pyrite from the sulphuric acid works is bought as a fluxing material, the fines are sifted out and added to the briquettes, together with the pyritic fines from the mine. There are 24 molding stamps.

The briquettes are dried in two furnaces, each having a capacity of 40 tons. Each furnace has three shelves. The flame from the fireplace passes over the first shelf, up a flue and back over the second, then through a flue in front and over the third shelf. The fuel consumption per furnace is 2 tons of a rather poor quality of bituminous coal.

The two furnaces are attended by 14 men working in 12-hour shifts or 28 in all per day.

It has been found that it is cheaper to utilize scrap iron than the burnt pyrites from the chemical works.

The present smelting works consists of two blast furnaces 9 ft. high by 3 ft. 4 in. wide, with six tuyères 6 in. in diameter on each of the long sides of the furnaces and forehearths in front of each furnace on the short side. From the tuyères to the bottom of flues the height is only 7 ft. The air main is 20 in. in diameter. The blast is furnished by two No. 3 and one No. 5 Root blowers run by a 40-h.p. dynamo.

From the forehearth the slag flows into pots to allow any matte particles in suspension to settle.

The matte concentration furnace is 40 in. in diameter and has six tuyères, 6 in. in diameter. The first matte produced contains 20 per cent. of copper and the concentrated matte 43 per cent. The slags from ore smelting are granulated. The ore bed in the first smelting is composed as follows: Briquettes, 826 lb.; mine ore, 495; slag from concentration furnace, 330; Mabuki slags, 82; scrap iron, 216; total 1,949 lb.

Each furnace smelts daily: Briquettes, 74,800 lb.; mine ore, 53,680; concentration furnace slag, 29,920; Mabuki slag, 7,480; scrap iron, 18,726; total, 184,606 lb.

The quantity of ore smelted in each furnace is about 50 tons and of fluxing material about 30 tons, making a total of 80 tons. The quantity of iron contained in these 80 tons is as follows: 50 tons of ore, 20 per cent. Fe, 10 tons; 17 tons of slag, 35 per cent. Fe, 6 tons; scrap iron, 8.5 tons; total, 24.5 tons. The ore and briquettes contain on an average 20 per cent. of iron, 21 per cent. of silica, and 12 per cent. of zinc. Air pressure at the furnaces is 26 mm.

It is well known that blende is decomposed by iron oxides and silicates, the resulting zinc oxide entering the slag; metallic iron liberates metallic

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zinc, but most of the zinc sulphide entering the blast furnace remains undecomposed and enters the matte as well as the slag. Generally the percentage of zinc found by analysis in matte and slag will be about equal.

It will be seen from the analyses that both at Kosaka and Kano this rule holds good for the first matte and the first slag, but in the concentration furnace a greater proportion was driven into the slag, which is very important, as blende makes matte less fusible, obstructs proper settling and also carries other sulphides into the slags.

If zinc oxide has to be slagged off, its reduction to metallic zinc must be prevented, consequently the temperature in the furnace must not be too high, and the smelting must be done quickly, which no doubt accounts for the low ore columns in these blast furnaces; and a slag obtained not too high in silica, but rich in iron. In the presence of lime, it is customary to figure that one-half of the zinc oxide replaces one-half of the lime. The ore columns in these furnaces are 7.5 ft. in height.

It will also be seen that the slags contain about the maximum amount of zinc they should carry, namely up to 8 per cent.; if the percentage was higher the losses in metals would be large. If zinc oxide is reduced to metal in the lower part of the furnace by carbon, or by metallic iron, it becomes volatilized and forms accretions in the upper part of the furnace. For this reason an excess of iron must be avoided and the calculation of furnace charges carried out to a nicety. The zinc vapors will carry along lead and silver and becoming oxidized higher up, carry off metal as flue dust. If zinc oxide is not reduced to metal on its downward course in the furnace and then comes in contact with lead sulphide or sulphate, it is converted into sulphide in the presence of carbon. If sufficient iron is present, the iron will decompose the lead sulphide and the zinc oxide will remain unchanged.

I am not aware if similar ores to the Kano are being at this moment treated in the United States, and if not, these clever Japanese metallurgists have solved a problem in pyritic smelting which can be placed to their credit.

The matte fall in ore smelting is 10 per cent. of the quantity of material smelted.

The concentration furnace charges consist of:

| | Lb. |
|---|---|
| First matte, 25 per cent. zinc | 500 |
| Mabuki slag | |
| Siliceous ores | 235 |
| Middlings from concentrator, 15 per cent. zinc | 85 |

With the present plant the daily production of second matte is 8 tons.

An analysis of the products from the concentration works gave the following results:

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| | Jig Concentrate | Wilfley Tables |
|---|---|---|
| Silver, oz. | 4.00 | 4.00 |
| Copper, per cent. | 3.22 | 3.59 |
| Iron, per cent. | 19.65 | 21.09 |
| Zinc, per cent. | 22.31 | 17.17 |

*The Mabuki-Dokos*

Four Mabuki-Doko hearths are used in connection with the matte concentration furnace as shown in Fig. 5. They are 3 ft. in diameter by 1 ft. 7 in. deep, each furnace holding 2.5 tons of matte.

The molten matte is poured into the Mabukis and wind pressure turned on to roast the metal for 4 hr. It takes 30 hr. to finish a charge during which time 675 lb. of charcoal are consumed or 12 per cent. of the weight of the charge. Each furnace produces 30 bars of black copper weighing 50 kins or 66.5 lb. (1 kin = 1.33 lb.)

During the first 16 months of smelting operations there were produced from 38,266 tons of ore: 479 tons of copper having a value, at £60 per ton, of £28,753; 3,812 mome of gold (1 mome = 10s.), £1,906; 442,636 mome of silver (1 mome = 15 sen), £6,639; total value, £37,298 This shows an extraction of 1.25 per cent. copper or 12s.5d.; gold, 1s.; silver 3s. 5d.; total value 16s. 10d. per ton of ore.

The assay value of the ore as coming from the mine is £1 5s.; consequently the loss in concentrating and smelting is 8s. 2d. per ton or 32 per cent. The gold and silver contained in a ton of copper produced formerly averaged £14, but in recent years the silver value shows an increase.

The mine is now being equipped with additional plant, so as to produce 200 tons of copper monthly. When this new plant is in operation, I estimate that the cost of making a ton of copper, including refining electrolytically and marketing will be £50.

The additional equipment in the new concentration plant includes: six Huntington mills, three Hancock jigs, 27 Overstrom tables, two Pindas concentrators.

The new smelting department comprises a smelting furnace 66 ft. in length, with two forehearths and two matting tables; two Root blowers to be run by a 150-h.p. electric motor; a chimney 150 ft. in height, with an inside diameter of 12 ft.; a new briquetting plant of 48 wooden stamps, together with four sets of ore-mixing machines and four drying furnaces; a Gates rock breaker, having a capacity of 300 tons in 12 hr.; and eight additional Mabuki-Dokos.

As the works are located in a highly cultivated country, filter presses were installed to clarify the water from the concentration works so as to avoid pollution of the river. The water is brought in a flume to a little valley and enters the presses under a pressure of 32 ft. In the settling

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ponds some powdered lime is added and the water is clear and drinkable on issuing from the presses. Each press takes 7 cu. ft. of water per minute and every five days the slimes have to be washed from the filtering cloth.

Experience has shown here that iron smoke stacks can be used for the smelting furnaces as a crust forms which protects the interior, whereas in the roasting furnaces, iron chimneys quickly corrode.

The power plant, which is being installed 3 miles distant from the mine, will furnish 500 h.p.

The fuel required in the metallurgical operations is as follows: Drying briquettes, coal, 8.33 per cent.; ore smelting, coal, 2 per cent., and coke 2 per cent. of the quantity smelted; matte concentration, coal, 8.8 per cent., and coke, 2.8 per cent. of the quantity concentrated; Mabuki-Doko, charcoal, 14 per cent. and coke, 2 per cent. of the quantity refined. When mining 16,850 tons of ore monthly, the cost of fuel per ton of ore will be (80 sen) 1s. 7d.

*Analyses of Ores, Mattes and Slags at the Kano Mine*

| | Copper, Per Cent. | Lead, Per Cent. | Iron, Per Cent. | Zinc, Per Cent. | Silica, Per Cent. | Alumina, Per Cent. | Barium, Sulphate Per Cent. | Sulphur, Per Cent. |
|---|---|---|---|---|---|---|---|---|
| Black ore | 2.37 | 3.34 | 10.50 | 20.94 | 27.74 | 6.98 | 2.18 | 17.40 |
| Sulphide ore | 2.27 | 1.07 | 19.74 | 12.82 | 19.70 | 5.78 | 3.98 | 22.26 |
| Siliceous ore | 1.36 | 0.37 | 9.97 | 7.81 | 52.30 | 7.42 | 1.46 | 13.87 |
| Briquettes | 2.47 | 1.11 | 17.00 | 13.16 | 22.53 | 8.16 | 3.21 | 22.50 |
| First matte | 19.95 | 1.18 | 37.24 | 9.82 | 1.50 | ..... | ..... | 26.62 |
| First slag | 0.39 | ..... | 32.75 | 8.32 | 36.00 | 1.56 | ..... | 1.89 |
| Second matte | 42.94 | 1.21 | 21.36 | 6.81 | 0.86 | ..... | ..... | 24.18 |
| Second slag | 0.54 | ..... | 34.53 | 8.68 | 31.30 | 6.64 | ..... | 1.73 |

The ore-smelting charges are made up of 53,680 lb. of mine ore and 74,800 lb. of briquettes or a total of 128,480 lb. Taking the average silica contents as 40 per cent., the charge contains 51,392 lb. of silica. The iron in the concentrates and Mabuki slag amounts in round numbers to 15,000 lb. and in the ore and briquettes to 18,000 lb., a total of 33,000 lb. This is not sufficient to slag such a difficult ore, and scrap iron to the amount of 18,726 lb. is added making the total amount of iron in the charge 51,726 lb.

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## VI. Smelting at the Tsubaki Mine

At the Tsubaki mine there is a large body of silver ore. This silver ore is smelted principally with copper ores purchased from outside mines. The orebody, which occurs in andesite, is divided by a shale dike 150 ft. wide; the northern orebody is called *Homa* and the southern, *Sozen*. The ore occurs in a crushed zone and looks like a mass of breccia which crumbles to pieces very easily; the ore carries lead, silver, sulphide and metallic silver.

The mine sends 116 tons of ore daily to the smelter. The fine ore is briquetted in a similar arrangement to the one described at Kano, 45 tons of briquettes being made daily.

The imported copper ores contain 4 per cent. of copper and 45 per cent. of sulphur. In the pyritic smelter 0.8 ton of Tsubaki ore is smelted to 1 ton of this 4 per cent. copper ore.

There are also brought here ores from the Karatoya mine of the following composition: Zinc, 28 per cent.; silver, 5.5 oz.; copper, 5.43 per cent.; iron, 12 per cent.; silica, 6.65 per cent.; BaO, 9.23 per cent.; and sulphur 29.31 per cent. The black zinc ores all carry barite in Japan.

Three smelting furnaces were in operation at the time of my visit but a new furnace 60 ft. in length was in course of construction.

The No. 1 furnace is 30 ft. in length by 40 in. in width and is used for reduction smelting; it has 20 tuyères on each of the long sides. The quantity of water required for the water jackets is 25 cu. ft. per minute. The daily ore and material smelted amounts to 173 tons and in the following proportions:

| | Tons | Parts |
|---|---|---|
| Tsubaki silver ore | 75 | 100 |
| Purple ore from acid works | 34 | 45 |
| Limestone | 23 | 30 |
| First matte from same furnace | 23 | 30 |
| Sulphide ore | 3 | 4 |
| Coke | 15 | 20 |
| Total | 173 | 229 |

The so-called purple ore is the residue from roasted pyrite furnished by the acid works.

The daily matte production is 40 tons which is re-treated in the same furnace and in the matte concentration furnace. The matte produced contains 20 per cent. of copper.

The No. 2 furnace, which is used for pyritic smelting, is 9 ft. long by 3 ft. 3 in. wide. The furnace has a forehearth with two matte siphon taps. The quantity of ore smelted here is 60 tons daily and the charge consists of:

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| | Tons |
|---|---|
| Tsubaki silver ore | 30.0 |
| Sulphide ores, outside mines | 30.0 |
| Zinc ores | 9.0 |
| Metallic iron (scraps) | 7.5 |
| Slag from concentration furnace | 12.0 |
| Coke | 1.5 |
| First matte | 10.0 |
| Total | 100 |

There is introduced through the tuyères 2.5 per cent. of bituminous coal. The cost of pyritic smelting in this furnace is given at 5s 6d. per ton. The matte, of which 10 tons are produced, contains 18 per cent. of copper. This matte is sent to the concentration furnace.

The height of the matte concentration furnace from tuyères to the flue is 8 ft. The charge consists of:

| | Tons |
|---|---|
| First matte from reducing and pyritic smelting | 20.0 |
| Second matte, circulating | 4.0 |
| Slags from Yamashita and Mabuki | 5.0 |
| Tsubaki silver ore | 11.0 |
| Limestone | 2.0 |
| Coke | 0.5 |

This furnace produces 10 tons of concentrated matte containing about 39 per cent. of copper; the slag contains considerable iron and 3 per cent. of copper and is sent back to the pyritic smelter. The cost of smelting in this furnace is given as 4s. 6d.

The blast is furnished by two Root blowers operated by a Pelton wheel. The pressure on the Pelton is 110 ft.; volume of water 15 cu. ft. per second; diameter of water pipe 30 in.; power developed 140 h.p. The two blowers require 85 h.p., and give a wind pressure of 30 mm. The remaining 55 h.p. operates a dynamo furnishing power for the locomotives on the electric railway. The main air pipe is 40 in. in diameter; branch air pipe to big furnace 30 in. The pyritic furnace has six tuyères on each side, the air pipes around the furnace being 12 in. in diameter.

### *First Matte Desilverized in Lead Bath*

The matte produced is drawn from the forehearth, as shown in Fig. 6, into a cavity, lined with brasque and covered with a dome of fire clay, in the bottom of which is 0.5 ton of molten lead. The lead bars which are placed in this cavity are produced from litharge, which results from the cupellation process. After some lead bars are introduced and melted by the hot matte which covers them, more bars are introduced until the requisite quantity is completed. A log of wood is then placed on



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top of the bath and pushed into the bottom by means of an iron bar introduced through a hole in the cover and kept in position by a cross piece held down by two men. This causes an active ebollution in the bath, and a thorough stirring of the lead with the matte. The bath is then allowed to settle and the matte, containing 45 per cent. of iron and some lead, drawn off and put back into the furnace. The excess matte goes to the other two furnaces. After drawing the matte, the lead bath is skimmed clean and the metal ladled into molds.

To keep the bath in a fluid condition a few pieces of wood and charcoal

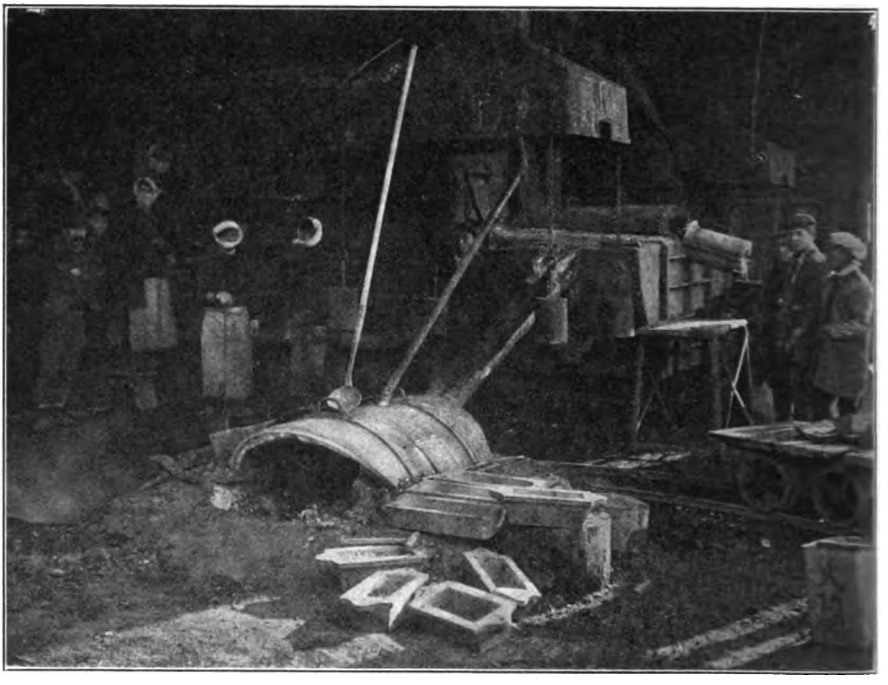

FIG. 6.—SMELTING FURNACE AT TSUBAKI WORKS, SHOWING MATTE-DESILVERIZING WELL WITH DOME-SHAPED COVER.

are kept burning on top of it. There are two cavities, one on each side of the forehearth, which are used alternately for treating the matte as above described. The lead bars produced here contain 570 oz. of silver to the ton and as the lead is impure it is first sent to the liquation furnace.

The "smelting in" or desilverization of matte (in German, *das Eintränken*) in a metallic lead bath is an old process and has for its object the decomposition of the silver sulphide and the absorption of the liberated silver by the lead. As the Tsubaki ores contain a certain proportion of metallic silver, it is hardly to be supposed that any would remain in that state after passing through the furnaces. As silver sulphide is not de-

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composed completely by the lead in one operation, the "smelting in" has to be repeated, and it is only after several operations that the complete desilverization of the matte is accomplished.

In some establishments the matte is smelted first in a reverberatory furnace and lead added and stirred in. In that case one part of lead is added to three parts of matte. After thorough stirring, the matte is allowed to settle and as it cools it is lifted off in disks; the lead, owing to its greater specific gravity, separates very nicely in the bottom of the hearth. The hotter the furnace is kept the more complete the desilverizing action. The matte is then re-treated in the blast furnace.

The operation is made more complete if in the molten matte and lead bath a log of wood is introduced and by means of iron bars is held on the

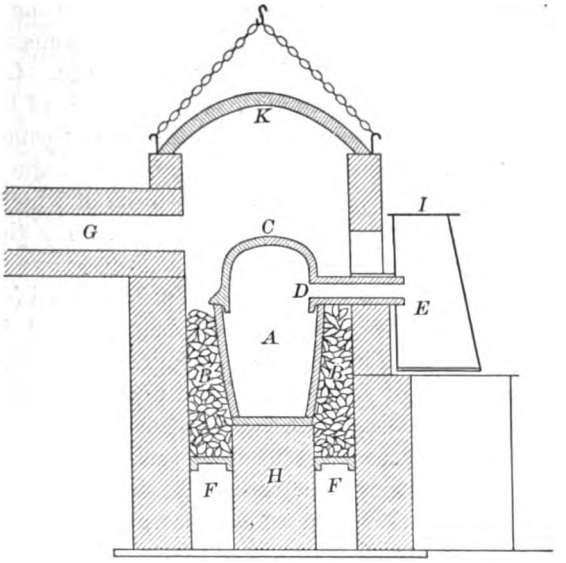


FIG. 7.—ZINC DISTILLING FURNACE AT TSUBAKI PLANT.

bottom of the hearth and kept there for several minutes, as the gases developed from the wood bring all of the molten and metallic particles into intimate contact. After the matte is drawn, the lead can be ladled out or it can be tapped. When matte is treated in this way in small reverberatories it is necessary to deal with the same matte at least four times, as just explained, to effect its complete desilverization.

The lead produced in the first operation may be rich enough to go direct to the cupellation furnace. After the fourth operation the matte now poor in silver goes to the ore-smelting furnace.

In Tsubaki the matte after treatment contains 45 per cent. of iron. The lead ingots are taken now to the liquation furnace.

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The liquation of these ingots is effected on a cast-iron plate built in on top of a brick square. This plate is turned up to form a rim 3 in. high and in the front a spout is provided. Hot charcoal is used and the ingots placed on top. The iron plate has an inclination of 12° and the molten lead passes out through the spout, collects in a little cavity in the ground and is cast into small square ingots which are sent to the cupelling hearth. The dross which remains goes to the open-hearth furnace or the Yamashita Buki.

In this process the largest portion of the silver and lead which is in alloy with the copper is smelted out, leaving a skeleton of copper behind with some silver and lead, whereas a small proportion of copper liquates with the lead. So far as I know this process, known in Germany as the *Saigerprocess*, is no longer in use there.

The matte from the concentration furnaces goes to the five Mabukis in a separate building. Each Mabuki holds 3 tons of concentrated matte, containing 40 per cent. or 1.2 tons of copper, per charge. Coke is used as fuel. There is added in the Mabukis 7 to 8 per cent. of litharge. The black copper produced here goes to special liquation furnaces where the lead is separated out. It is interesting to note that the black copper produced contains 3 per cent. of silver, two-thirds of which liquates out with the lead while one-third remains in the copper. The time of treating one charge in the Mabukis is 30 hr.

The Japanese *Nanban-Buki*, or liquation furnace, used for treating the black copper ingots, is a very small affair, being only 1 ft. 6 in. deep, about 2 ft. wide and 1 ft. 6 in. high. The four furnaces in use are built of brick lined with tiles of fireclay, and are provided with iron smoke-stacks which connect with the main flue of sheet iron leading to the dust chamber. Coke and charcoal are used as fuel. The blast at 10 mm. pressure is introduced through the top. The air upon striking the back wall is deflected upward and then takes a downward course to the front opening. Coke and charcoal are used as fuel.

Each shift of 8 hr. works up 0.25 ton of black copper ingots in each furnace. When the ingots become red hot and soften, the workman presses them with a piece of wood fastened to an iron rod. A charge consists of three ingots weighting 200 lb. The speiss which collects separately is pressed and passed on to the cupellation furnaces. The Mabuki black copper which is treated in these liquation furnaces contains: Lead, 4 to 5 per cent.; silver, 3 per cent.; copper 89 per cent.

The lead produced here goes to the cupellation furnaces and contains 10 to 13 per cent. of silver. The speiss contains 35 to 40 per cent. of silver. The black copper which remains behind is cooled in the furnace with a spray of water, and is then taken out and broken with a sledge hammer.

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*Black Copper Smelted, in Open-Hearth Furnaces*

The Yamashitas, or open-hearth furnaces, like the Mabukis, are built in the ground. The blast is introduced at the back. Coke and charcoal are first burned to heat the cavity and then the copper is flowed into a ladle suspended on a chain from which it is poured into molds on a moving platform car underneath. This copper contains 0.5 per cent. of lead. The copper from the Yamashitas assays 96.5 per cent. of copper and 1 per cent. of silver. The iron ingot molds are heated, by burning charcoal in them before the metal is poured.

The cupelling is done in two furnaces. The cupelling hearths are made of cement and can treat 2.8 tons of work lead in one furnace in 24 hr. The loss in cupelling is 7 per cent. The litharge as it flows from the cupel passes through incandescent charcoal placed in a cylindrical stove and is reduced to metallic lead, which collects in a cavity in the ground and is cast in molds. The pressure for the blast is 3 to 5 mm. The antimony speiss from the liquation furnaces is added during cupelling. The quantity of metallic silver obtained here is 51 kg., and the monthly production averages 1,600 kg. The silver contained in the black copper amounts to 480 kg. making a total of 2,080 kg., or in round numbers 70,000 oz. of silver. If all the silver came from the 3,480 tons of Tsubaki mine ore this would be an average of 20 oz. to the ton, but there are no data available on the subject.

A new plant is in course of erection in which the concentrates will be briquetted and the tailings lixiviated by some process to be determined after experiments are finished.

*Analyses of Ore, Slag, Matte and Furnace Products, Tsubaki Mine*

| | Silver Oz. | Lead Per Cent. | Iron Per Cent. | Zinc Per Cent. | Silica Per Cent. | Alumina Per Cent. | Barium Per Cent. | Copper Per Cent. | Sulphur Per Cent. |
|---|---|---|---|---|---|---|---|---|---|
| Homa silver ore | 33.0 | 1.0 | 2.5 | 4.0 | 29.5 | 4.5 | 43.3 | 20.00 | ..... |
| Soeen silver ore | 31.0 | trace | 4.5 | 2.0 | 40.0 | 2.2 | 30.0 | ..... | 25.0 |
| Matte, reducing smelting | 187.0 | 7.0 | 36.0 | 2.0 | ..... | ..... | ..... | ..... | ..... |
| Slags, reducing smelting | 1.4 | trace | 25.0 | 2.5 | 37.0 | 4.5 | 17.5 | 0.20 | ..... |
| Slags, pyritic smelting | 1.8 | trace | 22.0 | 3.5 | 38.0 | 5.0 | 14.0 | 0.12 | ..... |
| Matte, pyritic smelting | 117.0 | 5.0 | 38.0 | 3.0 | ..... | ..... | ..... | 18.50 | ..... |
| Concentrated matte | 503.0 | 4.0 | 29.3 | 1.2 | ..... | ..... | ..... | 38.80 | 23.7 |
| Black copper from Yamashita | 400.0 | 0.5 | ..... | ..... | ..... | ..... | ..... | 96.70 | ..... |
| Slag from Mabuki | 6.0 | 2.3 | 32.0 | ..... | 8.0 | 5.0 | 0.7 | 15.90 | 8.0 |
| Concentration slag to pyritic smelter | 2.0 | ..... | 28.0 | ..... | 34.5 | 7.5 | 10.0 | 0.30 | ..... |

*The Parkes Process in Use*

As a certain portion of the lead which is produced in the matte-desilverizing wells is not rich enough in silver to go direct to the cupellation furnaces, it is submitted to the Parkes process.

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The lead which has to undergo the Parkes treatment is melted in cast-iron kettles built in over fire places. Zinc is then added in portions of 1 to 2 per cent. With lead poor in silver two additions of zinc only are necessary, but with lead rich in silver and also gold, three, four or five additions may become necessary. It is necessary to employ pure zinc for this operation. When the first portion of zinc is added it absorbs any copper and gold which is in the lead bath; and there is a great advantage in this, as the further additions of zinc when collected are free of copper. The zinc after it is introduced in the lead bath is melted by raising the temperature of the lead bath to the smelting temperature of the zinc, and the contents of the kettle are stirred. The lead bath is then slowly cooled, and as the zinc, lead, silver, gold and copper scum rises to the surface it is carefully skimmed with a perforated ladle. When it is noticed that the lead begins to freeze, the temperature is raised again and a second portion of zinc added, the same operation being repeated three or four times until the assay shows that there are only 1 or 2 dwt. of silver left per ton of lead.

If there is any copper in the lead it goes over with the silver into the zinc. Any gold in the lead goes over into the first zinc which is added and forms a zinc-gold crust with a little silver. Nickel and cobalt are also absorbed by the zinc. Antimony remains in the lead; if present in large proportion it retains silver in the lead; if present in quantities not exceeding 0.7 per cent. it does not interfere with the zinc process. Arsenic and tin remain in the lead. Arsenic delays the desilverization and prevents a good separation of the zinc crust. Bismuth remains in the lead. Tellurium, platinum and palladium go into the zinc. Therefore if lead contains these impurities it should be purified before going to the zinc process.

The zinc scum obtained from the above process is liquated. This operation has for its object the removal of a portion of the lead, as the melting temperature of the lead is lower than that of the scum. If the zinc scum is heated to the melting temperature of the lead, the lead will liquate out, but the temperature must not be so high as to cause the oxidation of the lead and zinc, as a sort of a mush or pasty alloy will be formed which is infusible and resists further separation. After separating a portion of the lead, the remainder is called rich scum. This liquation is generally carried out in kettles, each provided with an inclined bottom and a spout so that the liquated lead can run out and collect in a sump to be submitted anew to the desilverizing process.

The furnace employed at Tsubaki for treating the rich scum is shown in Fig. 7. Graphite crucibles *A* are used which are placed in a round wind furnace, *B*. The covering consists of a dome-shaped hood, *C*, provided with an outlet, *D*, which connects with a condenser, *E*, made of cast iron. The grate bars are at *F* and the flue is at *G*. The crucible is placed on the brick foundation, *H*. The heating is done by means of

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coke. The cover is of fire clay and can be raised and lowered by chain and counter weights.

The scum is heated in the retort; the zinc evaporates and is collected in the condenser as oxide and metal, whereas the lead and silver remain in the retort.

The rich scum is mixed with 1 per cent. of charcoal powder and introduced in portions of about 500 lb. into the crucible. After the dome is luted on with fire clay and the pipe *D*, inserted and also luted, coke is placed around the crucible and glowing coals are placed on top. As soon as gases begin to escape from the pipe *D*, the cover is placed on the condenser.

After the operation is finished the condenser contains metallic zinc, about 30 per cent. of the charge, and zinc dust, about 8 per cent. The residue in the crucible, composed of lead, precious metals and copper, goes to the cupellation hearth. The coke consumption is about 80 per cent. of the weight of the charge.

## VII. Old Methods of Treating and Smelting Ores

*The Crushing and Pulverizing of Ores.*—A stone wall was built, 3 ft. square and 3 ft. high, in the center of which a square block of granite or other hard rock was placed and well tamped in. Upon this rock the ore was broken with heavy hammers and the pulverized material passed through horse-hair sieves, 40-mesh to the square inch, or through bamboo sieves.

*Concentration.*—The pulverized ore was treated in dolly tubs, the portion which floated being passed through launders while that which settled was further concentrated by vanning troughs. The tailings collected from vanning were ground in stone mortars and washed again, and the slimes were made to flow through a launder named *Neko*. The Neko is a frame over which a coarse cloth is stretched, about 10 ft. long, set obliquely. The portion of the slimes which remained on the cloth was again treated in dolly tubs and vanning troughs.

Granulated or grain ore was put into shallow willow baskets *Zaruage*, where it was subjected to a jigging motion in a basin of water. The fine particles which escaped through the meshes of the sieve were caught in the basin and passed over the *Neko;* that which remained was panned in a dish called "*Yurishita.*"

*Roasting the Concentrated Ore.*—The concentrates were mixed with clay and made into balls, which were heaped on to a charcoal fire in hearths made of earth. The sulphur was burned and the minerals oxidized.

### *Smelting in Open-Hearth Furnaces, Do-Buki*

This furnace is simply a cavity in the ground, of hemispherical form, with a diameter of 0.5 to 1.5 ft., and lined with brasque. The blast is

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supplied by an ordinary smith's bellows called "*Fuigo*," a square box in which moves a piston packed with a badger's skin. The piston rod is moved backward and forward by a coolie, who draws it out with the hand and pushes it back with the foot. Usually two such bellows are used with each furnace. The bellows furnish 4 cu. ft. of air per piston stroke or about 120 cu. ft. per minute.

The tuyères open into the upper border of the furnaces. The direct ascent of the blast is prevented by a vaulted roof of clay which extends above the orifices of the tuyères as far as half way across the furnace. The furnaces are separated from the bellows by a back wall, the products of combustion escaping through a chimney, built of framework and covered with loam, which is supported partly by the back wall, partly by pillars and commences about 7 ft. above the bottom of the furnace.

To blow in, the furnace is filled with charcoal, a fire lighted and the blast turned on lightly. Charges of ore are thrown in and the blast increased until complete fusion takes place.

When the furnace is filled with molten material nearly to the tuyère level, the blast is stopped, the burning charcoal raked out and water sprinkled over the fluid slag. As the slag solidifies, it is removed down to the fluid matte; then charcoal and ore charges are added and the operation repeated until the furnace is filled with molten matte and metal, whereupon the contents are ladled out or removed in disks, as the matte gradually cools. The red hot furnace is then repaired with clay and smelting operations resumed. When the furnace becomes unfit for use, it is cooled down, repaired with brasque, warmed over night and is ready for operation again the next morning.

In such a furnace from 3,500 to 4,000 lb. of ore can be smelted in a day, and some mines have a number of them in operation. The treatment in these furnaces is a very wasteful operation and the fuel consumption ranges from 30 to 70 per cent. of the weight of the ore. When ores are smelted containing gold, a certain proportion of metallic lead is smelted in and the bottom collects auriferous lead and matte.

*Roasting the Matte.*—The matte is roasted in stalls built up of common stones, of dimension suitable to local conditions. Brush wood or split cord wood is placed on the floor of the stalls, the matte piled on top of it, care being taken to leave vent holes, the pile covered with fine ore or fine matte and the fuel ignited at the air hole. The time required for roasting the matte depends on the size of the roast heap. Pieces of well-roasted matte are selected for the next operation, treatment in the Mabuki-Doko.

*Fusion of the Calcined Matte in the Mabuki-Doko.*—About 0.5 to 1 ton of the roasted matte is melted down in a hearth furnace similar in construction to the one above described. When all the charges have been melted down, the slag is removed and the front half of the top of the

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hearth which remained uncovered, is covered with a heavy piece of fire-proof clay tile, supported on short pillars of the same material.

A small opening is left in the front of the cover and an auxiliary tuyère inserted so that the blast is directed on the surface of the molten bath. When the smelting is completed the cover is taken off and the slag scraped away. The slag still remaining is solidified by sprinkling water on the surface. This slag is resmelted in the ore furnace. The matte thus exposed is removed in thin disks after cooling. The whole operation takes about 10 hr. with a consumption of charcoal amounting to 25 to 35 per cent. of the weight of the charge.

When precious metals are to be won, lead is smelted in with the copper in the bottom of the furnace, making an alloy of precious metals, lead and copper; this operation is called *Awase-Buki*. This operation is sometimes carried out in a separate hearth furnace.

Black copper containing more than 30 oz. of silver per ton is melted with about 40 per cent. of argentiferous lead in a hearth furnace with one tuyère which inclines downward. The charcoal is kindled and 270 lb. of black copper is melted down. About 70 to 80 lb. of lead are then added. When the hot coal, slag, and dross are raked out from the hearth, an iron bar with a spherical knob is dipped into the molten metallic bath, on which the metal solidifies and adheres, forming a thick crust. The bar is now removed, the incrusted end immersed in cold water and the shell or crust knocked off by a hammer. This is repeated until all the metal is removed from the hearth. Six or seven charges are operated daily in the same furnace, with a consumption of about 20 per cent. of charcoal.

*Liquation in Namban-Buki, or Namban-Shibori.*—This process has for its object the removal of the lead from the copper, silver and lead alloy, and in this manner most of the silver leaves the copper. This is accomplished by heating the black copper in small furnaces with a light blast to a temperature sufficiently high to melt lead but not copper. The molten argentiferous lead is drawn off leaving the copper skeleton behind. As this copper still contains some silver it is smelted again with lead, during the second liquation becoming very much impoverished in precious metals.

In one of the Osaka refineries the liquation furnace has an oval form and is covered with loam on the outside and lined with brasque on the inside. The top of the furnace is covered with a tile, leaving only a small semicircular opening in front, which is also covered with a tile. This opening is used as a charging hole for the charcoal during the operation. The open front of the furnace is also closed by a movable tile which does not reach to the bottom, but leaves a narrow space for the working of the metal.

An iron blast pipe directs the blast downward. The inclined working

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floor in front, formed by ramming down brasque between the side stones, terminates in a shallow cavity for receiving the liquated lead.

About 116 lb. of the alloy are charged in lumps, charcoal is piled in, the tile covers put in place, and the blast turned on.

When lead begins to flow out, and the mass becomes pasty, it is squeezed with a wooden rod, fastened to an iron handle; when it hardens it is pushed back into the furnace, so that it may become reheated. The process is repeated until almost all the lead is liquated out, the operation lasting 2.5 hr., with a consumption of about 50 lb. of charcoal. The quantity of copper alloy which is treated in one day in one furnace is 350 lb.

*Smelting the Liquated Copper.*—After liquation the copper is subjected to an oxidizing smelting in circular open-hearth furnaces. Charges of 275 lb. are smelted with charcoal and the blast, directed on to the surface of the bath, is turned on. In about 1.5 hr. the metal fuses, whereupon the hot charcoal and slag are drawn out and the surface of the metal bath is blown by the blast causing the phenomenon called copper rain; this lasts about 10 minutes and the mass is then allowed to cool gradually, when the frozen copper is removed in thin disks.

The whole operation lasts 2.5 hr. with a consumption of 165 lb. of charcoal. About 1,500 lb. of copper residues are smelted daily per hearth, producing 1,470 lb. of rosette copper and 33 lb. of slag, which contains 10 per cent. of copper.

*Cupellation, Hai-Buki.*—In some of the old methods this operation is carried out without a blast, by placing the cupel in a sort of a muffle furnace, but in most cases the cupel is placed on a hearth and a blast turned on from a bellows.

This is generally effected on a small open hearth, constructed of a wooden box, 3 ft. square and 2.5 ft. deep set in the ground. The cupel is made of wood ashes, the soluble salts being leached out.

In the center of the cupel is a circular cavity on which a charcoal fire is made and 100 lb. of lead added. The charge is then surrounded with tiles 1 ft. square and the blast turned on. As the metal fuses the charcoal is removed to the periphery and the blast made to play slowly on the metal bath. The litharge formed swims on the top of the bath and is absorbed by the ashes. The operation lasts about 2.5 hr., consuming 20 lb. of charcoal.

At a gold mine which I visited, the concentrates of arsenical pyrites, containing some galena, are briquetted by hand labor with calcined feldspar. After air drying they are smelted down in an open-hearth furnace, the blast being furnished by a small fan. The slabs of matte obtained in this first smelting are resmelted in a smaller open-hearth furnace and the matte, or rather speiss, taken off in thin slabs. During the second smelting a pool of lead collects in the bottom of the furnace, which is

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ladled out into a mold and cupelled in the manner above described giving gold bullion about 700 fine.

*Refining Silver Bullion, Parting.*—Silver bullion is fused on a flat hearth, and lead and sulphur added to convert the silver into a sulphide. The silver matte thus produced is sprinkled with water and cooled, or it is poured off carefully while still liquid, leaving the gold on the hearth.

The silver matte thus produced is again melted with lead, until all the precious metal contained in it is absorbed by the lead. It is then heated strongly to burn away the sulphur, and more lead is added; when the operation is finished, the lead is ladled into a mold and then cupelled.

Any silver and gold produced must come up to the standard of the *Hoji Koban.*

*Toughening the Copper.*—This operation is performed in clay crucibles heated in a hearth lined with brasque; charcoal is packed around the crucible, and a clay tuyère is fixed with an inclination of 20° to play on top of the metallic bath. Charcoal is then piled on top and the crucible heated to redness after which 66 lb. of copper are introduced, covered with charcoal and the blast turned on. In about 25 minutes, the charge completely melts down, and the slag and dross are removed. The molten metal is stirred well with a long hard stick of charcoal in order to toughen the metal or to reduce the cuprous oxide in the molten mass, an operation which is completed in a few minutes; the metal is then cast in molds.

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## The Metallurgy of Zinc

Discussions of the papers of DORSEY A. LYON and SAMUEL S. ARENTZ, RICHARD D. DIVINE, H. A. WENTWORTH, and S. E. BRETHERTON, presented at the Salt Lake meeting August, 1914, and printed in *Bulletins* Nos. 91 and 92, July and August, 1914.

GEORGE W. RITER, Salt Lake City, Utah.—We have every reason to believe that oil flotation will soon come into general use as a final guard against slimes losses in concentrating mills, not only in the case of zinc ores, but also in the case of other semi-precious ores. It also promises much as a primary process for the concentration of minerals that are not adapted to gravity methods. Moreover, modifications of the process involving selective or preferential flotation are gradually becoming understood, and will make it possible to eliminate pyrite and other minerals occurring as undesirable impurities in zinc concentrates. And what perhaps is of equal importance, further modifications promise an effective method of separating zinc blende from ores that are chiefly valuable for copper, lead, and precious metals, thereby making an asset out of zinc which has heretofore been a liability.

The tendency of certain pulverized minerals to float on water, especially after having been in contact with grease, has been known to millmen for a long time, and was formerly something to be deplored and to be striven against. All at once, this tendency has become a saving grace; and now, like tardy converts to a new faith, we are zealous advocates of something we once despised.

I was once employed at a mine in the Tintic mining district of Utah—the old Eureka Hill property—where ores that were too low in value to warrant being sent to copper or lead smelters were treated in a combination mill. The ores contained gold, silver, lead, copper, zinc, manganese, arsenic, etc., in varying quantities and in varying mineral forms, mingled in a quartz and calcite gangue. The material was stamped fine and then concentrated on Frue vanners so as to make a smelting concentrate, after which the tailings were thickened and treated in amalgamating pans for the recovery of non-concentratable precious metals.

In the amalgamating pans, under the influence of heat and agitation, a dark greasy froth or scum sometimes formed on the surface of the ore mixture. This froth or scum was composed mainly of mineral sulphides, and its occurrence might have meant only casual losses in the previous concentration, except that the phenomenon was accompanied by faulty amalgamation of the precious metals and by excessive losses in the mill tailings. Grease and oil, present by accident, were the cause of the trouble.

Whether the intense vibration in the mill, due to the pounding of the heavy stamps, had caused small particles of grease to be shaken loose

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from the heavy machinery, or whether the oil was merely lubricant that had passed through rock drills used in mining the ore, a single drop would spread over a large expanse of water and smear up a lot of pulverized ore. Having an experimental unit in our mill, we undertook to find out how to make the pulverized ore behave to our liking in spite of stray grease; and before we got through, we discovered a lot of facts concerning the behavior of greasy minerals under varying conditions as to heat, agitation, and weak chemical solutions. These facts ought to have pointed our way toward flotation as an economic process. They should have pointed further, toward the separation of flotative minerals from one another; because we observed that all minerals did not behave alike, and that sulphides of zinc were the most erratic of all. At that time, however, our results did not seem particularly significant, because zinc was then a liability rather than an asset, and what we were then trying to do was to prevent all forms of flotation; and our energies were directed toward finding lubricants that would be harmless and inactive in contact with ores, or which could be made harmless and inactive by means of reagents.

Zinc blende, when found in minute crystals mingled with pyrite and other sulphide minerals, eludes the simpler methods of mineral separation because it is not in itself unique as to density, crystallization, fracture, or behavior with chemicals. Its one distinctive difference seems to be in its behavior toward electricity. All of the other sulphides are good conductors; but zinc blende, except when contaminated with iron as an impurity, is a non-conductor. Upon this property rests the electrostatic concentration of zinc ores. A paragraph from Henry A. Wentworth's paper on Electrostatic Concentration or Separation of Ores[1] is worth repeating:

"There is an old experiment in physics where an electrified rod is brought close to a suspended pith ball. The pith ball is first attracted, clings for a moment to the rod, and is then vigorously repelled. As the rubber rod approaches the pith ball, a charge of opposite kind, so called, is induced on the side of the pith ball nearest to the charged rod, and as unlike charges of electricity attract one another and as the pith ball is very light, it moves to the rubber rod. But pith, though not a good conductor of electricity, does, because of the moisture contained, conduct electricity appreciably, and it soon becomes, as a whole, charged similarly to the rubber rod, and away it flies. This is the principle which is utilized in electrostatic separation, and to accomplish separation the differential property is the conductivity of the minerals involved."

In the same paper, Wentworth gives a list of more than 50 important conductive minerals; but zinc blende is placed in a list of non-conductors, along with quartz, feldspar, silicates, carbonates, sulphates, etc.

Whether or not it is anything more than a coincidence, experiment indicates that electrically non-conductive minerals are nearly neutral

[1] *Trans.*, xliii, 412 (1912).

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toward oils and gases in the flotation process, and that minerals having the highest electric conductivity are most amenable to oil flotation. Mercury, galena, graphite, nickel ore, pyrrhotite, etc.—this list of conductive minerals,[2] in descending order, is the list of minerals most easily recovered as a flotation concentrate when in finely pulverized state. Using the coincidence as a working hypothesis and carrying the experiments further, we find that oil flotation is not restricted to sulphide minerals, but that tellurides, chlorides, oxides, native metals, and other native compounds, can be brought into the flotation class under the right conditions. Impure zinc blende, because of its iron impurity, also comes into this class. The flotation class is enlarged still further if useful carbonates be first converted into oxides by means of calcination. For example, zinc oxide derived in this way from smithsonite can be recovered by oil flotation if a proper oil is used. This bit of research is one that I have not yet carried to a final conclusion; but a mixture of linseed oil and turpentine is one combination that seems to be effective; and whether we say that we are floating the material as a froth or as a paint, the important point is to do the trick economically and to overcome interfering agents that occur in the worthless gangue.

It is my impression that before we finally get to the underlying principles of oil flotation, including the separation of flotative minerals from one another, the depths of electro-chemistry and electrostatics will have to be sounded pretty thoroughly. In commentating on the migration of particles suspended in liquids under the influence of electric currents, LeBlanc[3] reviews the work of Helmholtz and others, tending to show that the migration is due to the presence of an electric charge upon the portion of matter in question; that the direction of migration is influenced by the addition of small quantities of foreign matter, such as alkali or acid, in solution; that at the surfaces of contact of two dissimilar media—for instance, the contact surface of water and glass—an electrical charge or double layer must form; that the arrangement is entirely analogous to an ordinary electric machine, with only this difference, that whereas in one case a liquid rubs past a solid, in the electric machine a solid rubs past a solid; and finally Coehn's answer:

"If two substances are brought into contact with each other, the one possessing the higher dielectric constant will become positively charged."

Here again we have a striking parallel between the phenomena of electrolytic migration and the phenomena of oil flotation; and a question that arises is whether we are not justified in considering a flotation tank and its agitating device as a dielectric machine; the solution, with its

[2] Landolt-Börnstein-Meyerhoffer. Quoted in *Bulletin No.* 548, *U. S. Geological Survey*, Electric Activity in Ore Deposits, by Roger C. Wells, p. 25 (1914).

[3] LeBlanc: Text-book of Electro-Chemistry, translation by Whitney and Brown, pp. 157 to 160 (1907).

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dissolved foreign matter, as an electrolyte; and the film of oil surrounding the mineral particle and any attached bubble of air or gas, as an insulator that enables the particles to retain their electrostatic charges, even in the presence of the liquid.

As between dry electrostatic separation on the one hand, and selective flotation on the other hand, the relative advantages may depend somewhat on the fineness to which the ore must be reduced in order to separate unlike minerals from one another. Oil flotation has the advantage of being adapted to mineral particles in a state of fineness approaching colloids, without calling for preliminary drying of the material, and without excessive installation and operating costs.

JAMES M. HYDE, San Francisco, Cal.—The flotation process being a new one in American ore-dressing practice, it may be well to outline briefly some of the factors that have to be taken into account in its application. The process depends upon certain physical phenomena quite different from those made use of in the ordinary water-concentration practice. Certain minerals having a metallic luster, particularly the sulphides, sulpharsenides, sulphantimonides, and tellurides, together with native sulphur and graphite, are easily wetted by oil, whereas the usual gangue minerals are not. There is an exception to this rule in that certain of the carbonates, particularly calcite and siderite, are rather easily oiled by the fatty oils, apparently because a certain degree of actual saponification of the oil by them takes place. Both the minerals which have been mentioned and oils also exhibit the phenomenon of readily attaching themselves to gases even in the presence of water.

The first successful application of flotation concentration on a large scale along anything like the lines now practiced was made in Australia by what was called the Potter process, in which the Broken Hill sulphide middlings, consisting of pyrite, blende, galena, garnet, calcite, and certain other gangue minerals, were introduced into a hot solution containing about 5 per cent. of sulphuric acid. The acid reacted upon the calcite, causing the evolution of carbon dioxide gas, to which the sulphide particles attached themselves and were thus buoyed to the surface and removed as a froth of concentrates. Following upon this work, installations were made in which a small amount of oil was added to the pulp, the amount of sulphuric acid was lessened, and the gases necessary for flotation were in part supplied by air beaten into the pulp by the violent agitation necessary to distribute evenly throughout the pulp the small amount of oil used and allow of a film coating of the sulphide particles with oil. This practice followed very closely along the lines disclosed earlier by Carrie J. Everson, and by Alcide Froment.

The process was first applied in this country on a large scale for the treatment of the ores of the Butte & Superior mine, at Butte, Mont. The original installation was made in a mill at Basin, Mont., where this

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company was temporarily milling its ores. It was found both by this company and by another company working upon similar material from the same vein, that ordinary milling practice, making use of jigs, tables, and vanners, could not profitably make a higher recovery from these ores than from 50 to 60 per cent. of the zinc which they contained. Some further recovery might be possible by larger installations of vanners, but the product yielded by vanners was so low grade as to be salable at very small profit. The ore contains its principal value in zinc in the form of zinc blende, the average content of the ore ranging from 18 to 21 per cent. zinc. With this occurs a small amount of lead in the form of galena, and several ounces of silver—probably in the form of silver sulphide. The principal gangue minerals of the ore are quartz, rhodonite and rhodochrosite, and more or less altered inclusions of granite. The high content of zinc of this ore and the low recovery possible by water concentration alone made it necessary to find some new method for its treatment. Laboratory tests, accompanied by screen analyses and microscopic examinations, showed that the material was successfully treatable by flotation concentration, but that, because of the peculiar nature of the ore and the intimate association of the blende with the quartz, a special method of treatment would have to be worked out for it. The principal modifications that proved necessary were the tube milling of the tailings resulting from vanning and table operations; preliminary treatment of the slimy portion of the pulp with sulphuric acid and lime or some other coagulant; and a novel method of roughing and cleaning by flotation concentration. This ore presented a peculiar problem in that even the coarse crushing produced a high percentage of slimes which were particularly difficult to settle. Tests made upon them have shown that after a slimy pulp had remained in a quiescent condition in a Callow cone for 3 hr. the solids still in suspension contained as much as 14 per cent. zinc.

The practice that has been adopted by the company makes use of jigs and tables for the recovery of as much high-grade concentrate as is possible by the use of these machines and involves the treatment by flotation of all the slimes produced incidental to the crushing throughout the mill combined with reground tailings from the jigs and vanners.

The process which was worked out for the ore has been so efficiently administered by the staff of the company in their new mill, working at a capacity of 1,000 tons per day or more, that in the first six months of 1914, 193,000 tons of ore were treated with an average recovery of 89.3 per cent. of their zinc content, made into a concentrate containing 52.4 per cent. zinc. The elasticity of the process is shown by the fact that whereas in January this recovery was at the rate of 90.67 per cent. of zinc contained in the ore, the concentrate produced running 51.3 per cent. zinc; in June it had been altered so that the recovery went 88.7 per cent. zinc but the product had been raised to a grade of 54.6 per cent. zinc. There



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is much greater profit in marketing high-grade zinc concentrate than low-grade material. On the basis that water concentration alone would profitably recover but 60 per cent. of the content of zinc in this ore, the figures given out by the company indicate that in this mill alone, during the first six months of this year, approximately 17,500,000 lb. of zinc were recovered which would have been lost, in large part at least, had it not been for the use of the flotation concentration process. A further advantage to the company has been gained by the ability to market its whole product in a higher-grade concentrate than would otherwise be possible.

S. A. IONIDES, Denver, Colo.—The dry chlorination treatment was started as a comprehensive scheme for saving zinc in these complex sulphide ores, and it proposes in one building to start with the ore and end with metals. The first step is drying, and crushing practically parallel to any other kind of drying and crushing, but with this distinction: it is not necessary to crush so fine. The only necessity is to expose one surface of each metallic particle, and not necessarily to isolate it, and the chlorine which comes in the second stage will attack that and be able to pierce it. Working at low temperature the sulphur gets no chance to coagulate, but the reaction between chlorine and the metallic sulphides, resulting in the formation of metallic chlorides, cannot be carried to a conclusion, and so a second stage is necessary, which consists of a light chloridizing roast, between 200° and 400° C. In the first step I should have mentioned that silver and probably gold are chloridized in addition to the base metals, copper, lead, zinc, and iron. The ferric chloride is not wanted in particular, and in ordinary practice it is found better to decompose this by the light roast I have mentioned, and the chlorine, freed from the ferric chloride, attacks the sulphides from the other metals and converts them into chlorides. The roasting ends the dry part.

After roasting, the ore is leached with hot water, and this will dissolve all of the metals with the possible exception of the gold. If there is any ferrous chloride or other reducing agent present the gold will not be dissolved. It will be carried out. Silver chloride is readily soluble in solutions of other chlorides, lead chloride in hot water, and zinc chloride and cupric chloride in water of any temperature. After thorough agitation the gangue is filter-pressed off, and the solution containing the metallic chlorides then goes to the refinery. The refinery follows the ordinary course of wet refining. Metals are precipitated on one another, the gold and silver on copper, the copper on iron, the lead on zinc, and the iron is removed with zinc oxide. This leaves a solution containing pure zinc chloride, which is then evaporated to dryness, fused, and electrolyzed in a fused condition, giving zinc and chlorine ready for re-use. The one point I would make about the process is that the loss should be very low. There is just one point in the process where loss can occur in-

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stead of three, four, or more, as in the ordinary retort and other processes. The one point is in the actual chloridizing. If that is not complete some zinc will go out with the gangue, but with the rest the only chance of loss is mechanical, and we hope to be able to keep that low.

S. S. ARENTZ, Salt Lake City, Utah.—During the past six months I have met a number of men more or less interested in the Rankin-Westling process for the wet treatment of all classes of sulphide ore. These men appeared to be so enthusiastic about this process that a word here will not be out of the way. The process consists in treating the ore with nitric acid in a closed retort. Steam is applied under pressure. The ore must contain about 7 per cent. sulphide. The gases and vapors driven off are caught in various receptacles. The gases caught include NO and $NO_2$ and contain nearly or all the nitrogen present in the nitric acid used in the treatment. The solution will contain, as sulphates, all the metals present in the ore. The treatment of zinc is particularly interesting to us to-night, and I have copied verbatim an extract from the Rankin pamphlet.

"After silver, bismuth, arsenic, iron, aluminium, copper, cobalt, nickel and cadmium have been removed, one can

"(a) if metal is desired, add to the cold sulphate a little less than theoretical amount of CA ($OH_2$) necessary to precipitate all zinc decant, filter and wash and treat the mixed hydroxide of ZN and CA $SO_4$ precipitate, electrolytically with an E. M. F. of less than three volts in a Rankin-Westling cell, or

"(b) if the oxide is desired treat the ZN ($OH_2$) and CA $SO_4$ ppt. with $SO_2$ in a closed tank to get ZN ($H\,SO_3$)$_2$, decant, filter and treat the acid zinc sulphite solution as given below for last part of our improved method, *i. e.*,

"(c) to the sulphate solution in a closed tank add the proper amount of CA ($H\,SO_3$)$_2$ plus some free $SO_2$, to obtain a precipitate of CA $SO_4$ and a solution of sulphates and ZN ($H\,SO_3$)$_2$ in dilute $H_2SO_3$; decant, filter, wash under pressure, into a closed precipitating vessel. Heat and from this solution release the first molecule of $SO_2$, thus obtaining a solution of remaining sulphates and a precipitate of zinc sulphite. Wash with $H_2O$; decant and filter; press the ZN $SO_3$. Remove zinc sulphite to a Rankin-Westling rotary tant-iron retort and drive off $SO_2$, and obtain technically pure zinc oxide. While still hot, drop into cold distilled water to render less crystalline, if desired.

OLIVER C. RALSTON,* Salt Lake City, Utah.—I am asked to take up in general the hydrometallurgy of zinc. I will try to state the situation rather briefly. At the present time at least 17 different corporations or individuals are conducting experiments larger perhaps than the test tube—that is, tests of 100 lb. or more—and doubtless a great many of the large zinc corporations are quietly carrying on work, although it is very hard to find out. Of those 17, over half are processes consisting in general of leaching with sulphuric acid, with electrolytic precipitation in view, and a few are leaching with hydrochloric acid, or metal chlorides, with

*Non-member.

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electrolytic precipitation in view. The other processes, two of which you have heard to-night, the Rankin process, just discussed by Mr. Arentz, and the Bretherton process, as well as the dry chloridizing processes, are somewhat different. There seems to be no difficulty in getting zinc into solution. There is one plant in Russia operating commercially, I think, on about 30 tons of ore per day, consisting of a limestone having 6 per cent. of zinc as carbonate, and they actually leach that out with sulphuric acid and make money on it. They have a special method of precipitation with a depolarizer of iron. In England is a notable plant, the Bruner-Mond Alkali Works, which has been going for years, using up a calcium chloride waste as a source of chloride ions, and getting the zinc into solution as a chloride from certain roasted zinc ores. They have been precipitating electrolytic zinc supposedly 99.6 to 99.96 per cent. pure, and I was informed to-day that they get 1½c. to 2½c. per pound extra for that pure zinc. In Germany for years there has been one plant, designed by Hoepfner, precipitating zinc electrolytically. The exact details of the industrial electrolysis of zinc sulphate, or of zinc chloride solutions, have never been published. They are of such a nature necessarily that if they were to be published these companies would have too much competition, and destroy their own trade—supposedly. Very recently work has been going on in California at Bully Hill, in which very good cathode deposits of zinc have been made, but we understand that to precipitate electrolytically requires a great deal of capital, and it is practically a rich man's process; it does not apply to any small work in zinc. That I think covers pretty well the present situation in regard to zinc hydrometallurgy. Recent work in England has shown that zinc can be precipitated in the presence of iron by an application of certain retardation and over-voltage phenomena, getting a very good deposit of zinc, and, moreover, the over-voltage prevents many bubbles of hydrogen, which formerly caused trouble in the way of trees, spongy zinc, and other such deposits of the zinc, which are undesirable.

D. A. Lyon, Salt Lake City, Utah.—The idea of igneous concentration is illustrated by the Fink process, with which some of you who are present are familiar. In this process, the furnace is a revolving cylinder which somewhat resembles a trough copper converter. After it is charged, the furnace is revolved while being heated, and air is blown through the charge. In this way, the zinc is oxidized. As before stated, this process illustrates what is meant by igneous concentration. This manner of recovering zinc from its ore appeals to a great many. They believe it is the ultimate solution of the zinc problem. However, one of the disadvantages of igneous concentration is that after you get your precipitate, or whatever form you get your zinc in, it is liable to be very fluffy and hard to handle.

Mr. Swart.—This morning I was introduced to a gentleman in the lobby, and he looked at me and said: "Oh yes, you are the zinc man," in

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the same sort of a tone as though I might be the ice man or the milk man. I suppose that means that they feel sorry for the man who is in the zinc business. We feel sorry for ourselves, and we have good reason to do so. The fact is that no one can predict just what is going to happen in the metallurgy of zinc. The losses discussed in the paper just read are not all of one kind and cannot all be corrected in the same way. Some of them are hopeless losses, I am afraid. The loss of zinc in smelter slag is one decidedly important loss, but in that same slag iron is lost as well as zinc. Is it fundamentally any worse to lose the zinc than the iron? This iron may not be worth as much per pound, but there is more of it and it is probably just as hopelessly lost. We ignore this loss because our iron supplies so enormously exceed our zinc supplies that the loss of the one seems trivial compared with the other, yet it is all a matter of degree, not of kind. We have had to make a slag from iron to get our lead metal and we have had to use zincky ores for the same reason.

There are some new things in the metallurgy of zinc, and I was glad Mr. Lyon mentioned them. I feel pretty well satisfied, for example, that igneous concentration is going to work well, but it is not going to be universally applicable. There doesn't seem to be anything universally applicable to zinc ores. Igneous concentration is being tried out now, and in my opinion a decided step in advance is being thus taken.

LAWRENCE ADDICKS, Chrome, N. J.—I noticed one of the speakers in the discussion said that the very pure zinc commanded a premium of 2½c. a pound. I had always understood that the difficulties in the way of producing electrolytic zinc were that the cathode was so rough that it could not be marketed without remelting, entailing a metal loss almost as high as that of the old fire process, and that you couldn't get any more for the product than you could get for ordinary spelter. The price of zinc is so low that it is very difficult to justify the cost of an elaborate process. I would be interested in knowing for what purpose the zinc which commanded so high a premium was used.

MR. SWART.—There is a limited market for high-grade zinc at an excess price, but it is a very limited market. In case any considerable amount of spelter were produced and thrown on the market by such processes as have been described, the premium would surely disappear. The premium is not 2½c. unless in exceptional cases in retail or special lots, but is considerably less than that on larger contracts, and even these contracts do not constitute what may be called the metal trade. There is also another thing of interest, and that is, that spelter made by electrolytic processes, while it may be chemically pure, or very nearly chemically pure, may still not have all of the necessary physical qualities. If you go into the market to-day to buy spelter, for instance, for making a special grade of spinning brass, and are willing to pay a premium for it, you can get it, but it won't be an electrolytic spelter. It will be made by the New Jersey Zinc Co. from ore found in Virginia or Tennessee, and

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electrolytic spelter, of which quite a little has been made and tried in Europe, does not apparently give the same results, in spite of its greater purity. This may be only one of those peculiar trade prejudices based on custom and inertia, but it exists and will be difficult to overcome.

S. E. BRETHERTON, San Francisco, Cal.—In regard to the special price of zinc made by the electrolytic process, I was informed by the chief chemist at Syracuse, N. Y., two years ago, that parties were making zinc in England by electrolytic precipitation, for which they were then getting from 1½c. to 2c a pound premium, which I think verifies, or helps to verify, the statement made by Mr. Ralston, and I took his statement for granted because such a man could not afford to make a misstatement. At that time I was looking into the recovery of ammonia, and the losses of ammonia in the Solvay process, where, as you know, they use carbonates of ammonia for making carbonate of soda and chloride of lime. The last product is mostly thrown away. I wish to state further that to-night I simply described the ammonia-carbon dioxide process very briefly, but it is fully described in the *Bulletin* of August, 1913. I thought I would mention this in case any one wished a detailed description of it. In that particular article I laid stress on the fact that the process I described briefly to-night is not suitable for ores containing silicate of zinc, or silicate of copper, or ores containing an appreciable amount of arsenic.

G. B. WILSON, Salt Lake City, Utah.—I would like to ask what this particularly high grade of zinc is used for. I presume it is for the manufacture of the salts of zinc.

MR. BRETHERTON.—That I cannot say. I simply know that the gentleman made that statement.

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## The Annealing of Cold-Rolled Copper

Discussion of the paper of EARL S. BARDWELL, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 92, August, 1914, pp. 2075 to 2094.

LAWRENCE ADDICKS, Chrome, N. J.—Eleven years ago I presented a paper before the Institute of Electrical Engineers covering this same field, but less comprehensively than it has been presented here, and at that time I made a study of the correspondence between tensile strength and conductivity, and found a straight-line relation between the two as you soften a wire by annealing, or harden it by drawing. Recalling this, there is one thing I want to ask about, which I think must be a misprint. I notice on p. 2081 that at the 700° temperature there was obtained a soft wire apparently without a rise in conductivity.

I might also add that a great deal of unpublished work along these lines has been carried out at some of the Eastern brass works, an that

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it has been found that the temperature of annealing is affected by impurities. Also that the time of exposure plays a part.

W. McA. JOHNSON, Hartford, Conn. (communication to the Secretary*)—In the production of a metal there is a certain loss. The losses of zinc, as scheduled so clearly by Mr. Lyon and Mr. Arentz, are generally higher than the losses of other metals, such as gold, silver, copper, and lead, for two interdependent reasons: zinc has sold for a 30-year average of but 5c. per pound, and is plentiful geologically as compared with the other metals.

The peculiar nature of its reduction necessitates a commercial minimum of 22 per cent. for such carbonates and of 30 per cent. for such sulphides as go to the retort plant, since the cost of coal and labor per ton of ore retorted is high. Accordingly, concentration processes—wasteful in metallic values, since blende is hard to concentrate, but cheap in operation and in capital cost—must be used to raise the grade of the ore so that it will be rich enough to stand the $20 to $30 aggregate cost of freight, metallurgical losses, and treatment charge per ton of ore. It is a rigid principle that it pays to waste mineral to attain the maximum commercial usufruct.

In short, we see that if zinc were not so abundantly distributed geologically that it could stand heavy losses in mining, milling, and retorting, and other commercial and adverse factors, it would cease to exist as a metal of prime commercial importance; for any increase in price cuts off consumption more markedly in the case of zinc than in the case of other metals, since, generally speaking, its use is not indispensable.

At any selling price, however, below 6c. per pound, society can afford to use spelter for many purposes, chief of which is galvanizing. There are two main reasons for the enormous growth of consumption of spelter in galvanizing, for which 60 per cent. of the spelter is used: People are building things in more permanent fashion each year and are therefore using galvanized iron and steel. Parenthetically, let me state my belief that some day steel bridges and structural steel in a large way will be galvanized. Also our use of sheet zinc for roofing material will be comparable to foreign consumption. Furthermore, the increase in the selling price of lumber has enormously increased the use of galvanized sheets. Considering conditions from several angles, we can expect zinc to increase in consumption in the future as it has done in the past provided the price does not exceed 6c. per pound. Confirmatory of this *à priori* reasoning, we find that if we extrapolate Mr. Siebenthal's curve of compensated spelter consumption,[8] the consumption of spelter in the United States by 1924 should equal about 850,000 tons. It would be impossible to produce all this amount of metal in the retort furnaces as now operated

*Received Aug. 20, 1914.

[8] *General Report Zinc and Cadmium in* 1912, U. S. Geological Survey.

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(where, indeed, are the laborers to come from now?), and besides the ore for this stupendous tonnage can come only by improved mining and milling and metallurgical practice.

Let us consider now the influence of the Johnson process in helping to furnish this large tonnage. I had best introduce this side of the question by saying that with power conditions and coal conditions as they are in North America, the Johnson process will not try to compete with the retort process on ores high in zinc and low in other values. The reason is plain: the cost of retorting a ton of Joplin ore is $10, and the cost of retorting a ton of Leadville or other Western sulphide ore is $12.50. In electric smelting the reverse is true and the cost of treatment is less as zinc tenor decreases. That is, the cost of treatment increases with increasing tenor of zinc. Let us assume as exactly true the probable truth that 100 lb. of slag-making materials can be smelted for 5 kw-hr., lead for 5 and matte for 10 kw-hr., and that 100 lb. of zinc can be smelted electrically for 125 kw-hr. Then two typical zinc ores of extreme type would show up as follows:

*Low Zinc-Lead Ore, Roasted*

Zinc, 20; lead, 15; matte, 10; slag, 35 per cent.

| | | |
|---|---|---|
| Zinc | 400 lb. @ 125 | 500 |
| Lead | 300 lb. @ 5 | 15 |
| Matte | 200 lb. @ 10 | 20 |
| Slag | 700 lb. @ 5 | 35 |
| Kw-hr. per ton of ore | | 570 |

*High-Grade Zinc Ore, Roasted*

Zinc, 70; lead, 2; matte, 1; slag, 7 per cent.

| | | |
|---|---|---|
| Zinc | 1,400 lb. @ 125 | 1,750 |
| Lead | 40 lb. @ 5 | 2 |
| Matte | 20 lb. @ 10 | 2 |
| Slag | 140 lb. @ 5 | 7 |
| Kw-hr. per ton of ore | | 1,761 |

To simplify calculation I have assumed 20 per cent. of oxygen in each case.

If we pay 3 mills per kilowatt-hour, energy cost will be in the first case $1.70 and in the second $5.28; at 5 mills per kilowatt-hour, in the first case $2.85 and in the second $8.80. The prohibitive charge of $8.80 for electrical energy in the case of Joplin ore can be seen at a glance.

I was fairly well convinced of the correctness of my own reasoning as exemplified above, but when it was corroborated by C. A. H. deSaulles, of the American Smelting & Refining Co., who has extensive practical knowledge of the zinc business, I felt absolutely sure that the proper line of development of the Johnson electric zinc furnace lay in zinc-lead smelting. If we add to the above the fact that there are plenty of zinc

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fluxing ores containing lead, copper, gold, and silver, with lime and occasionally fluorspar, and consider that our lead recovery will be higher than that of the lead furnace, we see the wisdom of starting zinc-lead smelting rather than electric zinc smelting pure and simple. It is indubitable logic. The fact that there are possibilities of saving labor in electric smelting has also bearing.

Exactly how all this electric zinc-lead smelting will affect the concentrating of ores is unknown. But it can be prophesied when crude ore is not used that the procedure will be to make roughly a zinc-lead middlings with considerable included galena, subjecting the tailings to the flotation process after grinding, then make other sulphide concentrates, giving clean tailings. Any part of the milling process that would naturally give a product high in zinc would be used, and this would go to the retort plant, whereas any stuff high in total zinc plus lead would go to the electric furnace plant. The marvelous perfection that the flotation process has reached at the Butte & Superior mill as a "clean-all" on tailings inspires belief in its large future sphere of usefulness. The electrostatic separator has several avenues of further success, especially when the copper sulphate treatment to make zinc sulphide superficially conductive is employed. Igneous concentration, as talked about by Mr. Clerc and done by the late Mr. Truax, has certain possibilities, but I do not look for it to come about at once nor to be of much importance.

Hydro-metallurgical processes will be used some day when zinc ore is scarce and chemical knowledge is so far advanced that laborers can make a "Group II A and Group II B separation" respectively, but that day is far distant, for quantitative analysis writ large does not spell metallurgy.

The improvement in the retort practice will surely become manifest, as can be judged by the increased tonnage that the plants in Kansas, Oklahoma, Illinois, and elsewhere have attained in the past 10 years. A charge of 16,000 lb. of roasted Joplin ore to a 320-retort furnace would have been unthinkable in 1903, when 13,000 lb. was the standard charge, yet to-day 16,000 lb. is normal. Unquestionably machine charging and discharging is about to be practicalized. I have always been inclined to larger retorts, but other practical retort men differ with me in this respect.

Along with this improvement in retorting and the advance in electric smelting and in milling methods, we can expect a combination of processes, each doing its own duty at a maximum efficiency. Considering the probable increase in demand in the United States and the fact that the European output, amounting to 600,000 short tons, is produced in centers now distressed by war, we can foresee that an enormous production will be in demand in the United States. This can only be met by the united effort of all zinc producers on this continent. The paper under discussion will contribute to the carrying out of such a task, and as such it is worthy of respectful attention and praise.

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## Nodulizing Blast-Furnace Flue Dust

Discussion of the paper of LAWRENCE ADDICKS, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 91, July, 1914, pp. 1671 to 1674.

JAMES H. PAYNE, Baltimore, Md. (communication to the Secretary*). —Mr. Addicks's account of the behavior of the nodulizer at Chrome on blast-furnace flue dust leads me to outline what has been done so far upon the application of the same apparatus to fine sulphide concentrates.

As Mr. Addicks states in his paper, the conditions prevailing in the test kiln at Yorktown in the demonstration I made for him upon Chrome material were strongly oxidizing; so much so that the sulphur was roasted down to less than 1 per cent. It is therefore to be expected that the first test made upon sulphide ore a short time after this, in which Ducktown green ore fines were used, did not show very promising results. Although the Ducktown ore contained only 13.52 per cent. of sulphur its gangue was quite fusible, and the intense heat generated by the oxidation of the extremely fine sulphide particles slagged the mass, bringing about a condition approximating that of a Brückner roaster in distress.

During the spring and summer of 1912, the writer kept working upon different schemes to cut down this slagging. Two successful methods were finally worked out embodying opposite principles, one promoting oxidation of the finer particles and the other retarding their oxidation.

In the early part of the present year the writer succeeded in interesting the Braden Copper Co. in nodulizing, with the result that an extensive series of tests were made at Yorktown upon oil-floated concentrates covering quite a wide range in analysis. These tests led to trials in the Chrome rotary on the part of the U. S. Metals Refining Co. of Minerals Separation concentrates which they had in stock, and later to test runs to check up the Yorktown runs on Braden concentrates. The large-scale tests checked up with the Yorktown tests in every way (except fuel consumption, which was, however, correctly predicted), and led to the Braden company's decision to adopt the process at Braden.

The fuel consumption upon oil-floated concentrates is not over 6 gal. per ton and in many cases is less. The action in the furnace is independent of the sulphur content, and there is actually less tendency to form nose rings in the Chrome rotary than upon flue dust. It is believed that the improvements entering into the design of the Braden kilns will cut down the nose-ring trouble to where it will no longer be a drawback to the process.

The flue dust produced by the kiln itself is practically *nil*, although some of the material treated has been as fine as 82 per cent. through 100

* Received Aug. 5, 1914.

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mesh. This is unbelievable to metallurgists familiar with the old Brückner roaster; but the conditions are entirely different. The material is fed wet and what dust there is precipitates in the atmosphere of steam at the exit end. The material, furthermore, is constantly moving downward to the hot end and is in a dry, dusty state but a short time, soon changing to a densified condition which is no longer dust.

The amount of sulphur in the nodules is under perfect control. It can be as high as 15 per cent. or as low as 5 per cent., as may be desired. The upper limit for good working seems to be about 15 per cent., as the nodules are very sticky with higher sulphur content. A product running 13 to 15 per cent. sulphur has been readily obtained on all oil-floated concentrates so far tried, where the object was to retain all sulphur possible. This makes pyritic blast-furnace smelting of sulphide concentrates possible.

The size and character of the product has varied with the different concentrates tested so far, and also with the sulphur content sought for. When high sulphur is desired in the product the nodules run smaller than the flue-dust nodules as produced at Chrome. They are, however, entirely free from material that would blow out of a blast furnace. A number of samples tested show from 42 to 45 per cent. of voids.

Nodulizing of sulphide concentrates, particularly oil-floated concentrates, opens up a new field, because it makes pyritic smelting of such material possible. Further, the operation appears to be such a cheap one that it may compete with roasting in multiple-hearth furnaces in reverberatory practice. If so, it would be far preferable to roasting in the case of the exceedingly fine oil-floated concentrates, because of the large amount of dust that the roasters must necessarily produce on this material.

R. M. Draper, Chrome, N. J.—The kiln will treat El Cobre flotation concentrates very successfully. There does not seem to be much of a tendency to form ring accretions, and those that form are nearer the discharge end of the kiln, where they may be removed much more readily. The temperature of the kiln is much lower for concentrates than for flue dust and the oil consumption much less. Our consumption of oil was about 6 to 7 gal. per ton of concentrates.

The nodules were very satisfactory from a blast-furnace point of view when charged into the furnace cold. We did have considerable trouble with crusts on the furnace if we put in too much of the hot nodules, due probably to the fact that they were near the smelting point when charged into the furnace and naturally smelted much higher up in the furnace. I think we would have had this difficulty with any hot material that was charged. In fact, we found the same tendency to crust when we tried the experiment of charging hot converter slag into the furnace several years ago.

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The concentrate feed averaged about 30 per cent. sulphur, and several samples of nodules taken at various times showed an average of 14 per cent. sulphur. One sample of nodules chilled immediately in water showed a sulphur assay of 20 per cent.

There is no difficulty in averaging 60 tons of concentrates per day, and I believe that with a proper feeding device this tonnage could be increased to 75 tons per day.

There is no question that the greater the tonnage the less consumption of oil per ton.

*Screen Test on Nodules from El Cobre Concentrates*

| | Per Cent. |
|---|---|
| Oversize (3 in.) and ½ mesh | 25.95 |
| Between ½ and 8 mesh | 44.75 |
| Between 8 and 16 mesh | 24.52 |
| 16 and 20 mesh | 1.64 |
| 20 and 30 mesh | 1.93 |
| 30 and 40 mesh | 0.46 |
| 40 and 60 mesh | 0.33 |
| 60 and 80 mesh | 0.14 |
| 80 and 100 mesh | 0.04 |
| 100 and 120 mesh | 0.06 |
| Over 120 mesh | 0.18 |

## Lead Smelting at East Helena

Discussion of the paper of EDGAR L. NEWHOUSE, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 92, August, 1914, pp. 1801 to 1806.

G. C. RIDDELL, East Helena, Mont.—As nearly as I can recall at the present moment, there are only four or five large plants on the continent where the lead on charge is carried in the neighborhood of 40 per cent. These are at Herculaneum, Collinsville, and Federal, Ill.; Trail, B. C.; and East Helena, Mont. Our object at the latter point is to smelt a maximum amount of Cœur d'Alêne lead sulphides in as compact and self-fluxing a charge as possible, and to this end we are now smelting a mixture that at times carries up to 42 per cent. lead.

The monotony of producing the 7,000 tons of bullion shipped each month to the Omaha and Chicago refineries is relieved somewhat by the fact that the 15,000 tons monthly of Cœur d'Alêne ores that are coming to us carry enough zinc to provide the one-half slag used with from 6.5 to 7.5 per cent. zinc day after day. During the month of March, 1914, three lead furnaces in continuous operation, each 136 in. long, averaged 249 tons per furnace day, exclusive of coke and slag shells, which is smelting at the rate of 5.5 tons per square foot of tuyère area. The

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R. P. M. of the metallurgist himself on this 40 per cent. lead-7 per cent. zinc operation are not a matter of record, but can perhaps be imagined.

The presence of so much lead on charge results in a very light slag fall, and it is this matter of the small amount of slag produced per ton of charge that makes the local metallurgy an interesting proposition at times. One ton of charge makes less than one-half ton of slag, comparing with three-quarters of a ton at most plants, and with the high lead burden carried in the furnace charge there is a great tendency for the metals to crowd into the slag produced. Needless to say, the lead is kept in the neighborhood of 1 per cent. (wet), in the slag, only by the most careful refinement and control in preparing and handling the charge at roasters and feed floor, distributing the feed at the furnace top, manipulating the furnace itself, and by attention to the type and analysis of the slag.

While the slag fall is abnormally light, the matte fall in this operation is exceedingly heavy. The Cœur d'Alêne ores that form the basis of our operation carry in the neighborhood of 10 to 13 per cent. sulphur, and 90 per cent. of the smelting stock is composed of these roasting ores. No other one item must be so carefully watched for its effect on the blast-furnace metallurgy as the matter of preparing in the roasting department a hard, firm, satisfactorily sintered roasted product, low in sulphur.

We have under way at East Helena, at the present time, a very interesting and promising series of roasting experiments, designed to cut the matte fall of the past few years, 15 to 16 per cent., down to 7 or 8 per cent. The blast roasting of these zincky lead ores has hitherto left a residual sulphur of 4 to 5 per cent. in the sintered product, but at the close of a long series of experiments on both the H. and H. and the D. and L. processes we have recently hit upon a combination of the D. and L. and the H. and H. which brings this final sulphur down to from 2 to 2.5 per cent. A trial run with this combination process on 100 tons of charge was entirely successful from the roasting point of view and we are now making a 2,000-ton test, to check up the first run and to demonstrate the behavior of this sintered product on the blast furnace itself. If the test comes up to expectations, and to the preliminary run, a detailed description of the method will perhaps be of interest to those members directly concerned with the smelting of high-lead charges, and will be forthcoming later.

R. C. Canby, Wallingford Conn.—Mr. Riddell says they adhere to the Eilers-type slag on the charge, placing emphasis on that. I would like to ask how closely they adhere to that type of slag?

Mr. Riddell.—We still do that, at the East Helena plant, although in doing so we do not feel that it is a necessity. As we change over from one smelting campaign to another, we generally move from one well-defined slag type to another. That is to say, if the two-fifth type at any time becomes too irony for the ore situation, we go to a 0.444, one-half,

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five-eighth, or a three-fourth slag. The 0.444 is not an Eilers type, by the way, but we feel in general that the closer we stick to a definite type slag, the longer does the furnace shaft remain free from accretions, and the better does the crucible behave. While we recognize the Eilers types as good slags, we feel that we have found a number of other types that can be used to advantage when the occasion requires.

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## Smelting Lead Ores in the Blast Furnace

Discussion of the paper of IRVING A. PALMER, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 91, July, 1914, pp. 1447 to 1460.

L. S. AUSTIN, Salt Lake City, Utah.—With the union of the Guggenheim interests and the American Smelting & Refining Co. in 1900, competition among the lead-smelting companies appeared to have been largely eliminated. This condition prevailed for but a short time, owing to the erection of smelteries by competing companies. Naturally, as competitors, the older companies endeavored to keep improved methods to themselves, not realizing, it would appear, that such knowledge could never be bottled up, but would drift into the common field of practice. Of late it has begun to be felt that more is gained than lost by adopting a franker and more open policy. The line, however, is drawn in giving details of costs, which, fully known, it is felt, would be taken advantage of by a rival.

As Mr. Palmer says, steady and substantial progress has been made, and latterly, I may add, the improvements have been radical.

Sintering by the Huntington-Heberlein method or with Dwight-Lloyd machines has effected an important saving over reverberatory roasting, the cost being, say, 50 to 75c., as compared with $2 per ton by the older method. Moreover, the large use of a sintered product has resulted in doubling blast-furnace tonnage. However, the change has not been an unmixed blessing, as in the machine roasting little time can be given to the breaking up of the blende, and it has been impossible to decompose largely the zinc sulphide, so detrimental in later blast-furnace treatment. The satisfactory roasting of the blende is still an unsolved problem. Nor is this all. For the best reduction it has been found that a part of the change should be crude (unsintered) ore, indicating that the already fused material is only too ready to change to slag, thus lessening the opportunity for better reduction.

So far as the mechanical handling of ores is concerned, the importance of the saving to be effected has perhaps been exaggerated. To be sure such mechanical handling has been reduced to a low figure, but even 15 years ago it was cheaply done. Philip Argall gave the cost of unloading, sampling, and coarse crushing (the ore being delivered in the

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bins) at the mill of the Metallic Extraction Co., Florence, Colo., at 11c. per ton. Cutting such costs in two, or to 5½c. per ton, makes no important saving, especially when we consider the interest on the greater cost of a present-day installation.

Referring to the boshing of the furnace, the introduction of air, the height of ore column, and the feeding, interrelated factors as they are, I would say:

It is a matter of observation for 30 years or more that after a furnace has arrived at a certain shape during its campaign, owing to shaft accretions, it will drive faster. With no extended flue system or bag house, as in earlier practice, it was necessary that the furnace should be expanded at the top of the ore column in order to prevent the formation of excessive flue-dust, and for the same reason the furnace was run slowly. To-day, with the large tonnages put through, this is no longer possible. The air rises through the charge at an estimated velocity of 22 ft. per second, as against 5 ft. per second in early practice. But with the removal of the fines, and with the aid of the flue system and the bag house, such rapid running is now quite possible.

The notion that the shape, size, and direction in which the tuyères are pointed are of importance, seems to me absurd. Of course such openings through the jackets should be ample, but, that penetration is in any way modified, conforms neither to reason nor to practice. The air enters the furnace and rises through the charge by the path of the least resistance, often an intricate and winding way. A partly slagged-over tuyère takes the air, not straight forward into the furnace, but through cracks, openings, and cavities and so to the more open part of the charge. This air is not to be recognized as a gust of wind traversing a free space, but rather as air pent up and seeking to escape where it can most easily pass through.

As respects the height of column, the older notion was "to increase the ore column in order to give more time for reduction." That idea we must give up. Furnace gases, in a well-working furnace, carry no more than 5 per cent. of CO. We know that this gas so diluted has but little reducing power. Moreover, it has less than a second in which to perform reduction even with a high column. Actually, reduction must be due to the action of the glowing coke in the melting zone.

I have already referred to the path of the air by channels of the least resistance. Hence the importance of suitable mechanical feed, supplemented by trimming the furnace by hand, so that the rising air escapes uniformly from the surface of the charge.

The matter of slag composition was threshed out years ago, and the literature of the subject is extensive and conclusive. Clean slags were made then and furnacing was done with as much skill as to-day. In some of the discussions the chemical language of "oxygen ratio and of

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single and bisilicate types" was used, but practically, as now, percentage composition was carried in mind. We speak to this day of a whole, a three-quarter, etc., slag, but it is only a convenient way of designating a type. At no time in the past generation did we think that a slag was other than a complex silicate with alumina and zinc sulphide not a constituent of that silicate, but dissolved in it. Quite in the past, as is also now the case, only fuel enough was used to give a hot slag of the proper fluidity. With our high tonnages the exothermic reactions of slag formation take place more quickly and so more energetically, and hence the lower percentage of coke that can be used. We have the disadvantage, too, owing to the use of mechanical charging, that the metallurgist cannot, as in the hand-charging days, closely see and study the phenomena of feeding, the action of blow holes, the appearance of the surface of the charge and the escape of the rising gases. Theory has had to replace in part actual observation.

In this, the Utah smelting center, the complaint is that, as the oxidized surface ores have been replaced by sulphides (especially blende), the difficulties of smelting have increased. To take care of this zinc sulphide, a more irony, less limy, and consequently less siliceous, slag has been made. The high-lime slags are better suited to ores low in zinc. The disadvantage of the more basic slags is, however, that, due to the lessened difference between them and the matte, separation is less complete. This brings into consideration the question of the use of the settling furnace; because of high-zinc slags, it has been tried and rejected in numerous instances.

It is the practice to send back to the blast furnace slag shells and other foul slag, with the idea that such additions aid in the smelting operation —viz., that slagged material would promote the formation of new slag, while incidentally its contained silver and lead would be largely recovered. Slag additions displace new charge, and it now appears that it will pay better to waste all the slag. We have to remember that it is now high in zinc and in sulphur, both elements we are glad to be rid of.

It is a fact that may be forgotten that when the slag is running cold, analysis shows that the iron content is low, and the metallurgist proceeds to increase it, when, were he to increase fuel, he would find that the iron would rise to the required figure.

Were I to predict in what direction we may hope for radical improvement in lead smelting, I would say that it would be in the use of the reverberatory furnace, principally because of the saving in fuel, or a reduction to 60c. per ton of materials smelted, against 90c. per ton as now prevailing. One of the first objections to suggest itself would be the difficulty of preventing the hearth from coming up and the leakage of lead into the foundation. However, in refining both lead and copper such hearths can be maintained. The next point would be, that should escap-

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ing gases be filtered in a bag house there would be their high temperature to overcome. However, since in passing the waste-heat boiler that temperature has been largely reduced, the necessary further cooling should not be difficult. Moreover, we have the Cottrell system as another method to fall back upon. In operation coal would have to be added to the charge and no doubt the fire carried with a neutral or reducing flame. Reverberatory smelting has the advantage that to correct the charge carbonaceous material, fluxes, oxidized ores, or roasted material could be added at any time, and that the charge need be tapped only when ready. There would be no tap-hole difficulties, no slagged tuyères, no hearth and shaft accretions to interfere. The molten contents of the furnace could be inspected and tested. There would be little or no flue dust, though no doubt considerable fume would have to be caught.

L. D. ANDERSON, Midvale, Utah (communication to the Secretary*).—Some further light has recently been obtained on certain points mentioned in Mr. Palmer's paper. With respect to the angle of the bosh in the furnace lines, a rebuilt furnace of only about 10° or 11° has been running beside furnaces of the old angle of over 20°. As far as observed, the new furnace seems to run much hotter and obtain a larger percentage of lead contents as direct bullion output. The matte fall, however, is considerably smaller, and it is necessary to carry more sulphur on the charge in order to maintain this fall high enough to insure clean slags. Judging from slag analysis and percentage of matte fall, the narrower furnace burns off half, and even more, of the sulphur as $SO_2$ as compared to only a third, or less, for the wider furnace. In these instances both furnaces are 48 in. at the tuyères, the difference in width being in the stack. The narrower furnace would appear to be superior to the wider one in that it has less tendency to form awkward hanging crusts, its crucible stays hotter, and, considerable heat being supplied by the combustion of the sulphur, a noticeable saving in fuel consumption can be effected. These remarks apply to rather coarse charges; for fine charges the differences would not be so marked.

I feel that the matter of the correct furnace shape is yet far from being settled. Every metallurgist has undoubtedly noticed that often after a furnace has been in blast for some time its work improves and the slags become surprisingly clean. Furnaces have been observed with crusts in them so heavy as to make it seem almost imperative to take them out of blast, whose slags nevertheless were so clean as to make the metallurgist reluctant to disturb them, until finally blowing out becomes imperative on account of the reduced tonnage. Experiences of this kind lead to speculations as to whether better reductions might not be obtained

* Received July 31, 1914.



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by making the shaft walls converging instead of diverging, somewhat after the lines of the iron blast furnace. In other words, why not make the furnace of the ideal shape at the outset, instead of waiting for it to form its own "natural" shape and running all the chances of irregularity in the formation of the crusts?

In this matter of furnace shape there also comes in the question of the amount of lead which it is reasonable to permit in the flue dust. An old, heavily crusted furnace tends to "blow through" and to produce an excessive amount of flue dust, high in lead. It has been found that, other things being equal, the percentage of lead in the flue dust or the bag-house dust is a good index of the work accomplished in smelting as a whole. Theoretically the bag house catches all the metals escaping from the blast-furnace tops. In practice, however, it does not actually save them entirely. Bag-house and flue dust have to be rehandled, and in the course of the rehandling several losses occur. Careful observation shows that when the lead in the various dusts is maintained at the minimum the percentage of recovery will reach its maximum. For this reason daily assays of the bag-house dust and adjustment of furnace conditions so as to maintain a low lead content in the dusts will prove of great value in the problem of keeping the metal losses at a minimum. The experiences leading to these conclusions appear to corroborate Mr. Palmer's remarks on the high lead losses experienced at plants whose slags were most admirable for cleanliness, but which had apparently not given full attention to the matter of volatilization losses.

Mr. Palmer's remarks on the desirability of a freer exchange of blast-furnace experiences are warmly seconded. Independent smelters are freely visited by officials of the larger corporation. The industry as a whole suffers when a policy of uncommunicativeness prevails. The handling of the blast furnace, after all is said and done, becomes finally a matter of personal judgment. Exchange of ideas on details can therefore have scarcely any effect on competition, while it will promote the tone of the entire profession.

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## The Leaching of Copper Ores

Discussion of the papers of FREDERICH LAIST and HAROLD W. ALDRICH, FREDERICH LAIST and F. F. FRICK, W. L. AUSTIN, and STUART CROASDALE, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletins* No. 91 and 92, July, and August, 1914.

R. C. CANBY, Wallingford, Conn. (communication to the Secretary*). —Apropos of the experimental reduction of copper from cuprous chloride by fusion with ground limestone and coke, as described by Messrs. Laist and Frick, and also proposed for use at Chuquicamata, it might be of interest to refer briefly to the experience at Argentine, Kan., in the smelting of the suboxide of copper (as produced in the Hunt and Douglas process) containing appreciable quantities of chlorides.

In the *Engineering and Mining Journal* of June 10, 1911, I gave some account of what had appeared to be crucial difficulties with the Hunt and Douglas process, and how the filter-press difficulty had been so rapidly overcome after my discovery of the effect of the sodium salts in the leaching solutions. I also described the effect of the cupola furnace gases upon the lime rock with which I had replaced the coke in the absorption tower, which apparently showed the presence of considerable chlorine, due to the imperfect conversion of the subchloride to suboxide, as well as that due to the difficulties of washing thoroughly the suboxide with the means provided, and also to the presence of chlorides in the cement copper which was smelted with the suboxide.

The fumes from the cupola, when issuing direct into the atmosphere, were very dense and white, except when a hot furnace top caused a reddish tinge due to dust. These fumes had a peculiarly suffocating effect, so that when they were blown over into the town the population suffered the greatest actual distress, not the mere annoyance which would have been due to a disagreeable odor. The absorption tower had been built at the instance of the city authorities to abate this nuisance, and not with the idea of making any additional metal savings.

A serious feature in connection with the Argentine experience with the Hunt and Douglas process was, that it had been implicitly assumed that there could be no metal loss in the Hunt and Douglas process itself, and that all attempts to locate the ruinous copper losses which were accumulating in the metallurgical accounts were devoted to the roasting department. It is possibly unfortunate for metallurgical advance along these particular lines that just about the time of this discovery of this previously unsuspected volatilization loss, in the smelting of the suboxide, the Kansas City company made a contract with the Western Union Telegraph Co. for its supply of copper sulphate and the Hunt and Douglas

* Received Aug. 7, 1914.

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process was abandoned, the plant being remodeled into a copper sulphate works. I regretted this, as I had confidence that by a more suitable manipulation these volatilization losses could have been overcome, thus giving an ideal treatment for lead-furnace matte, in which the lead, silver, and gold values were so promptly converted into bullion, and in which the copper was equally prompt in being placed in marketable form.

It is obvious that the conditions attending the proposed reduction of the cuprous chlorides by admixture of lime and smelting with coke are very different from those which obtained in the smelting of the suboxide, but the metallurgical experience at Argentine was such as to indicate that in the practical application of the proposed treatment of the cuprous chloride the greatest care must be taken to maintain the proper conditions, otherwise there undoubtedly will be very considerable volatilization losses.

LAWRENCE ADDICKS, Chrome, N. J.—I think this question of whether a process shall be wet or dry is a good deal of a temperance one. Some ten or fifteen years ago there was a good deal of intoxication over wet methods, which was not justified, and the result was that the use of liquor became rather unpopular for a while, and we had a prohibition period. I think now it is largely local option. We have to realize that every one of these cases has to be dealt with as a particular problem, and there is no fixed process that will apply to all cases.

I felt, in looking over these papers, that electrolytic deposition was in need of a champion, and I want to say a few words on that subject. I have been particularly interested in this subject the last three months in doing some work for Phelps, Dodge & Co., and I have been astonished to find under what adverse conditions it was possible to get good deposits. A coherent deposit premises a sufficient number of copper ions in contact with the cathode to satisfy the current. You can get this by having a solution very heavy in copper, or a solution considerably lighter in copper, but with a very rapid circulation. The ordinary electrolytic refiner has a handicap which is entirely absent in leach liquors: namely, the presence of gold and silver slimes at the anode, which cannot be disturbed by any violent circulation, or there will be prohibitive silver and gold losses in the cathode. I believe also that we are wrong in looking at this problem as a cyclical one, rather than a continuous one, and that it would be better to consider the use of liquor quite high in copper, say 3 per cent., take it down to 2 and 2½ per cent., run that back into the tanks and make up the half per cent. that is missing, and then back to the electrolytic tanks. In this way you get better depositing conditions, and, when studied over, it is not so bad from the leaching point of view. Of course, that means counter-current washing of the ore in order to bring the liquor up to the necessary strength

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for the depositing tanks, and indicates the use of Dorr thickeners, or something of that kind.

The question of the wash water bringing up the bulk is very easily dealt with in the very dry climates out West, by using a cooling tower and evaporation, to get reduction of bulk.

Now as a depolarizer, using a carbon anode iron is the real depolarizer in most cases, even though sulphur dioxide may be introduced. Iron is almost always present, and it is a very efficient depolarizer. We found in the experiments at Douglas that it is quite possible to consider working voltages as low as $\frac{6}{10}$ and $\frac{7}{10}$ volt. The depolarizer requires circulation in order to bring fresh iron sufficiently close to the surface of the anode. With such voltages we aim to obtain a recovery of 3 lb. every kilowatt hour.

There were one or two questions that I wanted to ask, in case anybody could give me any information. One is regarding the apparently remarkable wood out in this country, which seems to hold acid liquors, which wood won't do in the East. I am used, in all my experience in electrolytic work, to see wood after an exposure to sulphuric acid reduced to a soft charcoal, which you can poke your finger through. If redwood will stand acid to the extent which my brief observation indicates, I don't see that there is any problem entailing the use of asphalt mixtures and other protective coatings for this work.

The other question I wanted to put was whether any one here has had experience in the restraining of the solution of impurities by carrying the liquor with which you leach very high in those impurities. The real problem in Arizona is one of alumina, and what to do with it after it gets in solution. My idea was that we might take a solution so high in alumina that the amount dissolved from ore would be very little additional. I might also say we have worked with solutions and electrolytes as high as 7 per cent. alumina, and that we are using in our standard work 3 per cent. $Al_2O_3$ and 3 per cent. Fe in the liquor without any trouble with the deposit.

STUART CROASDALE, Denver, Colo.—Messrs. Laist and Frick have made an excellent summary of the conditions which govern the selection of a precipitation method in leaching processes for copper ores. The importance of a careful consideration of these conditions is apt to be overlooked by investigators who are over-anxious to develop one idea. The conditions in Anaconda are quite different from those in Chile or even those in Arizona, and a method of precipitation that is adapted to one set of conditions might be quite impracticable in another.

From my own experience, I believe the authors could have obtained a better absorption of hydrogen sulphide by introducing it through a diaphragm of burlap, or of perforated sheet lead like that used for fil-

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tration in chlorination barrels, or through a "silica sponge" that was on the market a few years ago for cyanide vats. Any sort of a diaphragm is much better than perforated pipes, provided the column of solution is not too high. With this arrangement, the depth of solution should not be over 6 in., and 3 in. is better. The gas bubbles then have a chance to hit each other and collapse, thus forming a foam with the solution, whereas if they have to go through a greater depth of solution they are forced together into large bubbles and of course escape like so many balloons. A launder, through which a shallow stream of solution could flow over the perforated bottom, would no doubt be a more efficient precipitating apparatus than a tank.

I believe it is possible to make hydrogen sulphide in quantity by a more direct method than by the decomposition of a solid sulphide with an acid. I have produced it quite readily by passing illuminating gas over pyrite at a low red heat, but this probably utilized only one atom of sulphur.

When pyritiferous lignite or bituminous coal is used in a gas producer, considerable hydrogen sulphide is generated.

These facts suggest that both hydrogen sulphide and power gas might be obtained from a gas producer by charging pyrite with the fuel and introducing steam in the proper proportion. A modification of the Hall process for desulphurizing might also yield the desired results.

In making sponge iron it is unfortunate that these gentlemen wasted time and money in duplicating my experiments with the multiple-hearth furnace without making any progress in that direction.

During August, 1912, I made a number of laboratory experiments on the reduction of calcines and iron ore, by using coke, coal, and gas for reducing agents. These were so encouraging that I went East in September, 1912, with Dr. Ricketts and the officials of the Calumet & Arizona Co., to investigate Jones's methods and those that had been used by the various iron companies for making sponge iron.

While visiting the Pennsylvania Salt Works, in Philadelphia, we learned that Professor Crabtree, of the University of Pittsburg, had been experimenting along this line a year or two before with the Wedge muffle furnace, but he had neglected to make his furnace air tight, and, after securing a reduction on the upper hearths, his product oxidized on the lower hearths.

After explaining to Mr. Webb, Secretary of the Furnace company, that we wished to roast pyrite for acid making and then reduce the calcines to sponge iron for a precipitant, we decided to order an experimental furnace of the Wedge double-function type, whereby we could roast the pyrite on the three upper hearths and deliver the hot calcine on the upper of three muffle hearths, where it would be mixed with

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powdered coal from a feeder attached to the side of the furnace, and be reduced to sponge iron on the three lower or muffle hearths. This furnace was set up and operated in Douglas, Ariz. We reached the limit of its possibilities, and completed this set of experiments by August, 1913. We learned what not to do and where our furnace was weak. The only thing to do was to build a new furnace along the lines I have already suggested in another paper before the Institute.

It seems that several months later Messrs. Laist and Frick duplicated our furnace in every essential detail, obtained the same results, and stopped their set of experiments where we were forced to stop. Could they have taken advantage of my experience, the time and money spent in duplicating my work would have made substantial progress in making sponge iron from calcines with a continuous furnace. I have gone somewhat into detail on this subject because it emphasizes once more the advantage of more centralized research among companies in the same line of metallurgy.

The days of secrecy have passed, and every metallurgical company in the country keeps open house for the continuous stream of managers, superintendents, and engineers of other companies that go the rounds "swapping information." Some engineers still entertain an imaginary value of patents, but these values have less chance of becoming real as the exchange of information becomes more common. It seems to me, therefore, that under these conditions much greater and more intelligent progress could be made in all branches of metallurgy, whether it be copper, lead, or zinc, if metallurgical companies, or at least closely allied companies, would segregate their research work into common departments in their respective lines and employ a competent force of engineers, metallurgists, electro-chemists, physical chemists, and others, as might be needed to meet the requirements.

This idea has been objected to as being destructive of individuality, subject to the petty jealousies of human nature and subject to the loss of unbiased viewpoints from independent research. It seems to me these are all offset by lack of progress and loss of time and money from the detailed duplication of work which it has been my experience to observe.

To secure cheap iron from calcines, I think efforts should be directed to the development of a continuous furnace. Encouraging results are now being obtained from a shaft type of furnace and some experiments of my own, made in September, 1913, lead me to believe that a cheap and efficient continuous furnace can be made after this pattern.

I do not see how sulphur dioxide can ever become a commercial precipitant for copper from solutions on a large scale. It may be, as the authors state, "highly attractive theoretically," but in leaching large tonnages of low-grade material, as now planned, I do not see how it

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can be made practicable to introduce a precipitant into cold lixivium under pressure, then heat the entire lixivium to boiling under pressure, and again cool to ordinary temperature, in order to get a product that has to be treated by another reagent in order to precipitate the copper. I need not mention the other difficulties that have been encountered when using this precipitant in straight copper sulphate solutions. It is, of course, an acid maker, but I believe acid can be made more cheaply by ordinary methods.

In fact, with the average leachable ore, where more or less iron, aluminum, and other elements pass into solution with the copper and the lixiviants soon become foul, I doubt if much advantage can ever be obtained by any method of dual precipitation. The regeneration of the lixiviant is seldom complete, and such a method involves the construction and operation of two distinct plants where cheapness and simplicity of operation are of primary importance.

DORSEY A. LYON, Salt Lake City, Utah.—The following statement appears on p. 1818 of the August *Bulletin:* "When the copper to be extracted is found mineralized as a sulphide the use of sulphuric acid as lixiviant necessitates breaking up the sulphur combinations by roasting in order that the metal can be brought into soluble form. This operation, however, instead of being a drawback, presents advantages. The expense of roasting is light, for it is very effectively and cheaply carried out in modern mechanical furnaces." I would like to inquire from those who have had experience along this line whether they believe this to be true.

FREDERICK LAIST, Anaconda, Mont.—I would like to reply to Mr. Lyon's question, and also to say a few words in reply to Mr. Croasdale's and Mr. Canby's discussion.

I hardly think that any saving of lixiviant due to roasting would balance the cost of roasting in the treatment of low-grade ore. The amount of soluble copper formed during the roasting operation would not cause the saving of much sulphuric acid. It would certainly not save an amount of sulphuric acid which would balance the cost of roasting, particularly when the leaching plant is located at a smelter where acid can be made around $3 per ton.

Regarding Mr. Croasdale's remarks on the absorption of hydrogen sulphide, I am quite certain that a much better absorption than we obtained could be gotten and that a more suitable apparatus than we used could be designed. The question with us, however, was whether hydrogen sulphide could be made cheaply enough from a low-grade copper matte to justify its use as a precipitatant. We never questioned the possibility of getting a complete absorption.

When we undertook to solve the problem of profitably recovering

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the copper from our sand tailings by leaching, some three years ago, we decided to do the work thoroughly and to try out every method of precipitation that offered hopes of success. Along with our $H_2S$ experiments we carried on experiments with various other methods, such as precipitation by sponge iron, electrolytic precipitation, sulphur dioxide precipitation, etc. It was quite early in the game that we came to the conclusion that while hydrogen sulphide could undoubtedly be used and developed into a commercial precipitatant, the method would not be as satisfactory as some of the others which were tried. Experimentation, therefore, along this line was dropped and hence there was no incentive on our part for spending the money necessary to develop a more satisfactory type of absorption apparatus than that used in the preliminary work.

Methods for the production of sulphuretted hydrogen, such as passing illuminating gas over pyrite at a low red heat, were tried by us, but did not give sufficiently satisfactory results to compete with the other method. Reactions of this kind are generally very incomplete and for this reason the expense of heating the pyrite and making the illuminating gas is high per unit of $H_2S$ produced.

As regards the making of sponge iron, Mr. Croasdale sees fit to criticise us for wasting time and money in duplication of experiments with the multiple-hearth furnace without making any progress in that direction. Both the making of sponge iron and its use for precipitating copper from solutions are old. Therefore, no one could claim originality for proposing the use of this reagent in connection with the lixiviation work now being carried on by various companies. Professor Lunge describes in considerable detail the making of sponge iron and its use for precipitating copper in his work on sulphuric acid and alkali. The iron oxide which is reduced to sponge iron is cinder or calcine which results from the roasting of Spanish pyrite after the copper has been removed by lixiviation. Soon after we actively commenced experimentation here it occurred to us that the iron which is so abundant in our calcines might be converted into a metallic form and used for precipitating copper. We looked up the subject of sponge iron and could find records of no apparatus devised for making it which fully met our requirements.

During the Butte meeting of the Institute, in the course of conversation with W. McA. Johnson, he suggested the use of a muffle-fired MacDougall furnace. At that time, we had not arrived at any clear idea as to what form of reduction furnace could be used on a large scale, and we were building a small retort furnace in order to obtain several hundred pounds of sponge iron for experimentation. Mr. Johnson's proposal sounded pretty good to us and we decided to fix up one of our MacDougall furnaces and give the method a trial. We did not know at that time exactly what work Mr. Croasdale was engaged on, nor

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would such knowledge have materially influenced our actions. The cost of making the experiment was not great, and while it was an absolute failure, it clearly showed us wherein lay the defects of the MacDougall furnace for this kind of work and served to put us on the right track.

The MacDougall furnace is admirably adapted for oxidizing, and for that very reason it is not adapted for carrying on operations where a powerfully reducing atmosphere must be maintained. It is almost impossible to entirely exclude the air, and it is impossible to heat the charge to the reduction temperature by means of muffles without the consumption of an excessive amount of fuel. Moreover, the expense of keeping the muffles in repair is prohibitive.

We, therefore, turned our attention to the Brueckner cylinder as a more promising type of furnace for this work. We rapidly carried out a series of experiments, starting with a small hand-turned cylinder, and finally ending up with the present 8½ by 12 ft. cylinder, in which we are making about 20 tons of sponge iron per day, with a moderate consumption of fuel and the production of a high grade of sponge iron.

Mr. Croasdale further criticises the use of $SO_2$ as a precipitatant and thinks the expense of working under pressure, heating the solutions, etc., must be very formidable obstacles. We naturally investigated these phases quite thoroughly before we did much work along this line. The pressures required are moderate and involve no more difficulties than blowing a converter charge; in fact, the pressure is the same (16 lb.) as is used in converter practice. The saturating vessel need not be closed provided it is made sufficiently high so that the lower portions of the liquid will be subjected to this pressure. The most expensive part of the operation is heating the solutions. This expense can be materially lessened by the use of counter-current apparatus in which the hot precipitated solutions are made to heat the cold solutions from the absorber. Our calculations indicate that precipitation by means of $SO_2$ can be accomplished for around $10 per ton of copper, without taking any credit for the regenerated acid.

I am strongly of the opinion that where electrolysis cannot be used, owing either to high cost of power, rapid fouling of the solutions, or the necessity for maintaining chlorides in the solutions owing to the presence of silver in the ore, precipitation by $SO_2$ is not only the best way out of the difficulty but is an entirely practical method.

As Mr. Canby states, the reduction of copper from cupreous chloride is accompanied by a certain amount of volatilization. This is more marked when cupreous chloride is first decomposed by milk of lime, as was done in the Hunt-Douglas process, the resulting cupreous oxide, so called, being smelted. We found in our experiments that the prod-

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uct obtained in this way was a mixture of oxychloride and suboxide of copper. When you try to smelt this the volatilization losses are very heavy. The method that we used, and which is also proposed by Mr. Smith for Chuquicamata, does not give the copper much chance to volatilize because the limestone present undergoes double decomposition with the cupreous chloride long before the volatilizing point is reached. Our experiments indicate that about 5 per cent. of the copper is volatilized, 75 per cent. of which could be recovered from the fume, so that the net loss of copper would be quite small.

W. L. AUSTIN, Riverside, Cal. (communication to the Secretary*).—Replying to the question raised by Mr. Lyon: If a sulphide ore is to be leached with sulphuric acid solution it must of necessity be previously oxidized, and roasting will usually prove to be the most economical method of oxidation. Roasting puts the ore in excellent condition for leaching. For example, a slime that cannot be leached in the state in which it is found in a tailing pit, can be readily treated after roasting. Roasting furthermore sulphatizes lime present, and converts the iron into more or less insoluble ferric oxide. Sulphur dioxide used in the electrolytic vats as a depolarizer can be obtained from roaster gases. Roaster gases when used in this way may furnish all the acid required for leaching, and may considerably reduce the amount of current consumed in electrolytic deposition. For these reasons, roasting, "instead of being a drawback, presents advantages" in many instances; but of course an ore both economically and metallurgically suitable for the purpose indicated is presupposed. The author had no intention of advocating roasting all low-grade ore with a view to saving lixiviant by production of sulphates. Ore which is amenable to leaching is unfortunately seldom found where sulphuric acid is obtainable "around $3 per ton."

W. L. AUSTIN, Riverside, Cal. (communication to the Secretary†).—On p. 1907 of Mr. Croasdale's paper, Leaching Experiments on the Ajo Ores, a table is given showing the amount of acid consumed when leaching said ore by percolation. Analyses made of the liquors during the second and third applications of lixiviant indicate approximately the combinations entered into. The following figures are taken from the tables mentioned:

First and last analyses; second lixiviant:

| | $H_2SO_4$, Per Cent. | Cu, Per Cent. | $Al_2O_3$, Per Cent. | Fe (Ferric), Per Cent. | Fe (Ferrous), Per Cent. |
|---|---|---|---|---|---|
| Second lixiviant..... | 10.31 | 0.9 | 0.57 | 0.49 | 0.13 |
| 9 p.m..... | 0.72 | 3.0 | 0.94 | 0.40 | 0.49 |
| | − 9.59 | + 2.1 | + 0.37 | − 0.09 | + 0.36 |

* Received Sept. 18, 1914. † Received Sept. 25, 1914.

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Assuming that these several metals were present in the lixivium as sulphates, the following quantities of acid may be accounted for:

| | Per Cent. |
|---|---|
| $H_2SO_4$ taken up by the copper | 3.24 |
| $H_2SO_4$ taken up by the $Al_2O_3$ | 1.06 |
| $H_2SO_4$ taken up by the Fe (ferrous) | 0.63 |
| | 4.93 |
| $H_2SO_4$ released by the Fe (ferric) | 0.24 |
| Total acid consumption accounted for | 4.69 |
| Free acid unaccounted for | 4.90 |

With the third application of lixiviant the account runs as follows:

| | $H_2SO_4$, Per Cent. | Cu, Per Cent. | $Al_2O_3$, Per Cent. | Fe (Ferric), Per Cent. | Fe (Ferrous), Per Cent. |
|---|---|---|---|---|---|
| Third lixiviant | 10.00 | 0.4 | (0.58) | 0.16 | 0.26 |
| 9 p.m. | 1.44 | 2.54 | 0.98 | 0.55 | 0.54 |
| | −8.56 | +2.14 | +0.4 | +0.39 | +0.28 |

| | Per Cent. |
|---|---|
| $H_2SO_4$ taken up by the copper | 3.3 |
| $H_2SO_4$ taken up by the $Al_2O_3$ | 1.15 |
| $H_2SO_4$ taken up by the Fe (ferrous) | 0.49 |
| $H_2SO_4$ taken up by the Fe (ferric) | 1.02 |
| Total acid consumption accounted for | 5.96 |
| Free acid unaccounted for | 2.6 |

In the analysis given on p. 1884 the oxidized ore is shown to contain 0.56 per cent. CaO, but it is reasonable to assume that this was neutralized by the first lixiviant, and as apparently the three lixiviations were carried out on one charge, the small amount of CaO present in the ore could not have influenced acid consumption in the second and third lixiviants. There are no other elements mentioned in the analyses which might account for the excess acid consumption. As free acid in large quantities was removed from the lixiviant, it must have gone into combination with something, and consumption of acid has an important economic bearing on leaching porphyritic ore with sulphuric acid. On p. 1912 it is stated that 3.6 lb. $H_2SO_4$ (not commercial acid) was consumed on the average per pound of copper dissolved.

Furthermore, it is shown in the diagram on p. 1908 that in the initial stages of the treatment free acid is used up very rapidly, out of all proportion to the copper going into solution. It would add to the value of Mr. Croasdale's excellent paper if he would throw further light upon the points raised.

In general, when a comparatively weak sulphuric acid lixiviant is used, the copper in oxidized ore is for the greater part removed before the other elements are seriously affected, whereas a strong (10 per cent. acid,) lixiviant attacks all bases simultaneously. The objection made by Mr.

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Croasdale to the use of a weak lixiviant—namely, that neutralizing alkalinity in the ore needlessly consumes time—might be overcome by making the lixiviant stronger in the beginning, and then carrying on the operation with a weaker solution.

The writer has found in some cases that by reducing an oxidized ore to 16 mesh, and violently agitating with a strong acid solution, the copper can be dissolved in a surprisingly short time without undesirable quantities of other elements appearing in the lixivium. In this way it is sometimes possible to dissolve the metal with an acid consumption of less than 1 lb. $H_2SO_4$ per pound of copper extracted.

Some tests made to determine the effects of lixiviants of varying acidity in dissolving copper from oxidized ore showed that with a certain percentage of acid the lixiviant acted practically on the copper alone. When a certain point was passed the acid was rapidly consumed without correspondingly effective action on the copper.

In connection with acid consumption at Ajo, it would be interesting to know what proportion was neutralized by CaO in the concrete of which the vats were constructed. Concrete will stand a cupriferous solution carrying 1 or 2 per cent. $H_2SO_4$ fairly well, but it is rapidly eaten into when 10 per cent. solutions are used.

STUART CROASDALE.—The table on p. 1907 referred to by Mr. Austin was prepared only for the relative value of the results shown.

In my leaching operations it was impracticable to remove each lixiviant completely before adding another, or even to allow a space between the two, on account of the danger of entrapping some air, which would have a tendency to produce channels or imperfect leaching after the ore had been once saturated. Therefore, as soon as the first lixiviant was drawn down to the top of the charge, the second lixiviant was applied. I then continued to draw off the first lixiviant until its estimated volume was removed, before I began the circulation of the second lixiviant, but there was always an indefinite and considerable amount of the first lixiviant retained by the ore, which became mixed with the second lixiviant as soon as circulation began. The same would be true of the second and third lixiviants, so it would be impossible to account for the acid consumption from these percentages alone, as Mr. Austin has figured it.

The acid consumption per pound of copper dissolved, as given on p. 1912, was found by determining the percentage of acid, the specific gravity, and the volume of all solutions and wash waters. It was also estimated by determining the combined acid from the analyses of these solutions at the end of a test or clean up from 75 to 100 or more tons of ore.

As shown, these results are 3.6 and 3.15 respectively, which is a very close check, considering the slight but unavoidable errors in making free-acid determinations in solutions containing salts of iron, aluminum, and copper.

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## Electrical Fume Precipitation at Garfield

Discussion of the paper of W. H. HOWARD, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 92, August, 1914, pp. 2029 to 2046.

EDGAR M. DUNN, Anaconda, Mont.—Mr. Howard's contribution to the literature on the Cottrell method of electrostatic precipitation of flue dust and fume at Garfield has been read by us with peculiar interest for the reason that we have been experimenting with the same process at Anaconda for the past eight or nine months, with a view to adapting it, under the direction of Dr. Cottrell and the supervision of J. O. Elton and D. R. Kellogg, representing the Anaconda Smelter Commission, to the treatment of very large volumes of gases, using large pipe treaters and various forms of electrodes, with about 150,000 volts rectified current. For the amount of gas to be treated here the 5-in. pipe means a cost for installation that is practically prohibitive, due to the tremendous number of pipes required, and we have been trying to cut installation costs by increasing the size of the pipes. All of our work is still in the experimental stage, but we have obtained good clearance on our mixed roaster and converter gases with a 3-ft. pipe treater 20 ft. long, and with 5 ft. velocity therein; and with a discharge electrode of a single No. 29 nichrome wire set in the middle of the 3-ft. pipe, with a consequent electrode spacing of 18 in. The current used has been 150,000 to 160,000 volts, and of course there have been troubles galore, relating particularly to insulation and rectifying. Blast-furnace gases did not give so good clearance, probably due to relative poverty in moisture and $SO_3$ content of blast gases to roaster-converter gases; making the blast gases the poorer in conductivity. Our experience has been so satisfactory that in our next experimental plant—built on a somewhat larger scale—we shall attempt a 4-ft. treater with greater electrode spacing, and a somewhat higher voltage—up to 220,000.

As to fractional precipitation, mentioned by Mr. Howard, this idea was tried out at Anaconda last year (1913).

Formerly our furnaces (Bruntons) roasting flue dust for the recovery of white arsenic, passed so great an amount of dust to the settling kitchens that refining of this arsenical product was necessary to eliminate the 5 to 20 per cent. dust contained therein. Consequently the manufacture of $As_2O_3$ was effected in two stages, with a reverberatory roast of the first product to obtain the final 99.5 per cent. $As_2O_3$.

Part of the dusting trouble was due to too high a temperature and draft in the first roasting furnace, and we are now installing new furnaces of another type to volatilize the arsenic trioxide in the dust recovered from our flues. This process is simply one of volatilization (physical

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change) of the $As_2O_3$ present in the flue dust; no chemical change of an appreciable nature occurs in the best practice, the arsenates contained in the dust not being commercially reducible below a smelting temperature.

Fractional precipitation was used last December to separate the $As_2O_3$ from the dust carried along from the first roasting furnace in the gas stream. Two treaters were used. In the first—gases entering hot, 310° C.—practically all of the dust was precipitated; while in the second —gases cooled to 90° C. by admission of air—white arsenic of 99.7 per cent. was directly thrown down; fractional precipitation thus eliminating the refining furnace formerly necessary.

Our new flue-dust roasting furnaces will be equipped with fractional precipitation treaters, thus saving the cost of refining the crude product; their operation must still be regarded as experimental, due to change in type of roasting furnace. We hope to furnish complete results to the Institute at an early future date.

Finally, we note with regard to Mr. Howard's report of experiments at Murray on fractional precipitation, that "as high as 95 per cent. of the total precipitate was collected" in the first treater at an average entering temperature of 125° C.; while in the second treater (average entering temperature 70° C.) "most of the remaining dust, acid, and water, approximating 4 per cent., was collected."

This might lead one to believe that the first treater was but 95 per cent. efficient. A moment's thought, however, will bring out the fact that the efficiency of the first treater, *at its temperature*, must have been close to 100 per cent.; the change in vapor tension between 125° and 70° rendering additional acid and water precipitable at the lower temperature of the second treater. In other words, had the gas contained vapors capable of precipitation between 70° and atmospheric temperature, a third treater (or a lowering of the temperature in the second treater) would have recovered those vapors in the liquid or solid state.

F. G. COTTRELL, San Francisco, Cal.—There is very little I can add to Mr. Howard's paper except a word of appreciation for his energy and ability in carrying this work through to completion. As most of you doubtless remember, the work on this same general subject which was under way several years ago at the Balaklala smelter, in California, was terminated by that smelter shutting down on account of fume troubles, in which at the time no very definite segregation was attempted as to the part played respectively by dust and sulphur dioxide.

Although that installation ran for only a few months, and was really never thoroughly completed, it served to bring the process forward to a scale large enough to give an idea of what could be done in the collection of values, and Mr. Howard was the man among the members of the smelt-

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ing profession who took enough interest in the work, saw the possibilities with sufficient clearness, and had the courage, in the face of the skepticism of many of his associates and official superiors, to take the work up purely as a matter of saving values. The process thus entered on this second stage which he has so admirably carried through at Garfield.

Looking backward on the history of the work, it appears doubtful whether it would ever have lived through the difficulties encountered in its early stages had it not appeared as a possible solution of the life and death struggle some of the plants were then making in the courts on fume-damage suits. But once these early difficulties were passed and the work was placed on a reasonably definite and certain basis as an established method, its general importance to the industry and the interest now taken in it have rapidly come to center chiefly around the collection of values previously lost in the smoke.

The work at Garfield, as you know, is aimed primarily at the saving of values from the converter flue. The degree of recovery attained at the Balaklala plant appealed to Mr. Howard as amply warranting, even at that stage, its operation on converter flues, aside from any question of fume nuisance. In the present plant, with the lack of any direct disturbance from that cause, it should be possible to gain a thorough knowledge of what can be done on the recovery of values. In the meantime, of course, as Mr. Dunn has told you, the work has gone on at other places, largely due, I feel sure, to the stimulating effect of Mr. Howard's own confidence and example. A plant nearly the size of that at Garfield is already running at the Trail lead smelter in British Columbia.

I feel that Mr. Howard is to be very heartily congratulated on the successful way in which he has carried out this work in the face of a great many difficulties, some of which I have had opportunity to see and know as the work progressed.

I have with me a few samples of the material collected on the Anaconda work which I thought might be of interest in this general connection, as they illustrate in a rather striking manner the possibilities of fractional precipitation suggested by Mr. Howard in his paper and further emphasized by Mr. Dunn's discussion. In this set [exhibiting samples] there are three bottles numbered 1, 2, and 3. No. 1 is the original feed to the arsenic furnaces in one of the experiments of Anaconda; No. 2 is the dust precipitated in the first precipitating unit—that is, precipitating the dirt out of the arsenic at a temperature still retaining the greater part of the arsenic in the form of a gas; and No. 3 is the refined arsenic precipitated in a subsequent electric treater after chilling by admixture of cold air. The gray of the original flue dust, the almost jet black and the snow white colors respectively of the impurities removed and the refined arsenic obtained serve to make this, as you notice, a very good illustration of the principle.

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## A Comparison of the Huntington-Heberlein and Dwight-Lloyd Processes

Discussion of the paper of W. W. NORTON, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 92, August, 1914, pp. 1993 to 1999.

ARTHUR S. DWIGHT, New York, N. Y.—Mr. Norton's chief argument is directed against the idea that the D. & L. sinter is any better material for the blast furnace than the H. & H. product. He supports this by citing two experimental furnace runs.

*Reducibility.*—To discuss a furnace record like that quoted by Mr. Norton for the five days' run in August, 1912, is manifestly difficult for an outsider, but a possible explanation for the poor reduction on Furnace No. 5, smelting D. & L. sinter, may lie in the fact that both sets of furnaces carried the same fuel charges. Other things being equal, if 11½ per cent. coke was right for the furnaces smelting H. & H. product, it was too much for the furnace smelting D. & L. product, as I shall endeavor to show later, and incipient overfire, accompanied by higher leads in slag and matte, was induced. It may sound like a paradox to say that excessive fuel sometimes produces poor reduction, but it is a sad fact.

In certain plants where practically the entire charge to the blast furnaces receives a preliminary treatment by being sintered on the D. & L. machines the peculiarities of this problem have had to be very carefully studied, for serious metallurgical difficulties presented themselves when the change was first made to D. & L. sinter in large quantities. Some of these plants had blast pots before they had D. & L. machines, and many opportunities were afforded for comparative study of this very point. There is reason to believe that, due to the long period of blowing and the higher temperatures developed in the pots from greater mass action, the iron in the H. & H. product is present chiefly as $Fe_2O_3$, perhaps in the form of ferrites, while in the D. & L. product the iron is present as FeO. The fuel charge in the blast furnace must be so proportioned as to exactly meet the chemical and thermic requirements of reduction and melting. If any of these functions have been performed before the material enters the furnace, a corresponding allowance should be made or the fuel balance will be disturbed and bad work will result. One of the functions of the coke is to reduce enough metallic Fe for the matte requirements, leaving the remainder of the iron as FeO to go into the slag. In smelting almost any other material than D. & L. sinter the iron must be reduced from the higher oxides, $Fe_2O_3$ or $Fe_3O_4$, to the protoxide, FeO, before a true comparative basis is reached. Experience has shown that with about one-half D. & L. sinter on the charge the coke should be cut about 10 per cent. This theoretical benefit is borne out by the practical blast-furnace results on a large scale in numerous plants. In fact, wherever this very important factor is ignored the evil symptoms of excessive fuel will be indicated,



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one of which symptoms is a diminished furnace speed with tendency to overfire, and, as every metallurgist knows, this is likely to be accompanied by poor reduction in slags and mattes, especially if the mechanical features of the problem, such as size of lumps and distribution in the furnace, have not been correctly adjusted. This general fact has been quite convincingly proved, at least to my own satisfaction, by the numerous cases where I have been called upon to observe difficulties and recommend remedies in connection with the growing use of D. & L. sinter in smelting operations.

To put the matter tersely, and emphasize the important point I wish to make, the successful smelting of D. & L. sinter not only *permits* of economy of coke, but actually *demands* it.

*Furnace Speed.*—In the furnace record of Aug. 12 and 13, 1912, when the furnaces smelting D. & L. sinter ran a trifle slower than those with H. & H. product, it is to be noted that the former carried 580 lb. iron ore on the charge, while the latter required 1,140 lb. iron ore, indicating that the D. & L. sinter was itself very high in iron. It is well established that a very basic sinter does not smelt as fast as one in which the proper proportion of $SiO_2$ and FeO for the slag is approximately maintained. This may or may not be the reason in this case, but it is worth mentioning.

The following figures were furnished me some time ago as giving the comparison of average furnace speeds in a certain plant where the character of the charge sent to the blast furnace was gradually improved by the introduction of modern methods of preliminary treatment. The furnaces in question were 48 by 162 in. (54 sq. ft.) at the tuyères, and the averages were taken over considerable periods of time.

| | Average Tons Smelted per Furnace Day | Tons per Square Foot Hearth Area per Day |
|---|---|---|
| (1) With Brückner roaster product | 180 | 3.3 |
| (2) With Huntington-Heberlein product | 200 | 3.7 |
| (3) With H. & H. product and ore fines largely removed by screening | 227 | 4.2 |
| (4) With Dwight & Lloyd product and ore fines largely removed by screening | 285 | 5.3 |

Case (4) shows a gain of 25 per cent. over case (3).

In another Western plant the addition of D. & L. sinter to the charge increased the furnace speed from 140 tons per day to 175 tons per furnace day, a gain of 25 per cent.

In still another case the increase was from 180 tons to 210 tons, an increase of 17 per cent.

These are from the reports of the managers and can be easily verified. Many others could be given.

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*Low Melting Point.*—Mr. Norton expresses his belief "that a partly fused or 'predigested' combination may tend to poor results rather than to good results when smelted, for the reason that such substances fuse at too low a temperature in the furnace."

I ful y recognize this as a possibility and indeed a probability when D. & L. sinter is treated without due regard to its properties, but I positively assert that such tendency can be quickly and easily met with no change from ordinary standard furnace equipment, and with distinct fuel economy, as has already been pointed out.

A very pertinent citation on this point is the experience of a certain smelter superintendent who had to carry a considerable amount of leady converter slag on his charge. He was much troubled by premature fusion of this ingredient until he discovered recently that by crushing it and adding it to the charge of his D. & L. sintering machines he was able to control this tendency and is now regularly following this practice.

I do not wish to prolong this discussion unduly and therefore will only suggest in the briefest possible manner my passing comments on the other matters I would like to speak of.

*Sulphur Range.*—The general experience has been that higher initial sulphurs can be handled by the D. & L. than by the pots. This is to be expected from the smaller mass under treatment at one time, and the mechanical means for rapidly abstracting the heat as it is developed by the oxidation of the sulphur. As an example a certain plant figures its D. & L. charges between 12 and 14 per cent. sulphur, and its H. & H. charges for direct conversion at 10 to 11 per cent. sulphur.

*Sulphur Elimination.*—There are many data available to indicate that the D. & L. is at least as thorough in this respect as the pots, with a far greater efficiency per unit area of plant.[1]

*Operating Costs.*—The H. & H. plant at Murray has about double the capacity of the D. & L. plant. The operating cost per ton is necessarily influenced by this fact. If the costs were corrected to an equivalent basis the small difference in favor of the large H. & H. plant would probably be more than wiped out. Without going into details, it may also be said that a true comparison should include all the pertinent items, such as interest on investment, depreciation, etc., which would unquestionably throw the balance in favor of the D. & L. In this connection, I wish to say that the D. & L. process is most distinctly on its mettle at Murray, for Mr. Norton has worked out the H. & H. process to an extraordinary degree of perfection and his practice in that department justly ranks among the very best.

*Fineness of Charge.*—As Mr. Norton says, the D. & L. machines have to handle the very fine material that the pots cannot treat. In

[1] A. S. Dwight: Efficiency in Ore Roasting, *School of Mines Quarterly*, vol. xxxiii, No. 1, pp. 1 to 17 (Nov., 1911).

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many plants this includes also the fines from the pots. In other words, the machines have to do the dirty work, because they can, but it is only fair to ask how the figures would be affected if the H. & H. pots had to take their share of the very fine material.

*Lead Losses.*—Mr. Norton does not attempt to present any conclusive data on this point. As the volatilization is influenced largely by the temperature and the time the charge is subjected to the heat and air currents, the advantage ought theoretically to be with the D. & L., for in that process a given particle is undergoing the heat treatment about as many minutes as it is hours in the H. & H. pot. Investigations of lead losses have been made at various plants, some of which will probably be published some day; meantime, it can be stated that while it is possible to establish conditions which will make a D. & L. machine into a very efficient lead burner, it is not necessary so to do, and the lead losses should be less than with the pots. As an example, in one plant under my direct observation where pots have been entirely superseded by D. & L. machines, the lead loss was about 3 per cent. in the pots and is now about 0.75 per cent. as the average over a period of two years with the machines.

In conclusion I wish to again thank Mr. Norton for his interesting paper.

G. C. RIDDELL, East Helena, Mont.—Mr. Norton's interesting paper, comparing the H. & H. and D. & L. processes, no doubt reflects with a high degree of accuracy the behavior of these two roasting methods as he has found it at the Murray plant.

Up in Montana, however, we are working with somewhat different conditions, and as we have been a little more favorably impressed with the relative merit of the Dwight-Lloyd sintered product, it may be of interest at just this time to describe the behavior of these two processes at the East Helena lead plant of the American Smelting & Refining Co.

The reason for a difference in performance of these roasting processes at Helena and at Salt Lake lies perhaps in the rather unique roasting and smelting practice at the former plant, where all operations are on charges carrying an abnormally high lead and zinc content. Helena mixture beds contain approximately 80 per cent. Cœur d'Alêne lead sulphides and average 160 to 180 lb. to the cubic foot. The local significance of this high percentage of heavy leady material can readily be seen when it is remembered that the processes under discussion involve the blast-roasting principle. This density, together with the ready fusibility of such a lead-bearing charge, presents a set of conditions at East Helena that give rise to a number of interesting problems at both roasters and blast furnaces.

Certain it is that with both H. & H. and Dwight-Lloyd products in excellent physical condition, our East Helena blast furnaces run at higher speeds and with distinctly lower coke when dominated by the Dwight-

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Lloyd sinter. All those daily observations, recorded and unrecorded, that the metallurgist makes as he guides his blast furnaces through their various ailments and difficulties, have for several years past confirmed us at East Helena in the feeling that the more Dwight-Lloyd material we can get on our furnaces in place of H. & H. product, the lower is the percentage of coke required and the faster do the blast furnaces run. That we have dealt with extremes as well as averages in these matters, may be seen from the fact that the East Helena furnaces have carried upward of 90 per cent. roast on charge for long periods, with smelting tonnages of 475 to 575 tons per 100 sq. ft. tuyère area. An average of 249 tons per furnace day, maintained during the month of March, 1913, figures 549 tons per 100 sq. ft. tuyère area.

In December, 1912, one blast furnace was run for seven days on a charge containing H. & H. roasted product and no Dwight-Lloyd; a furnace alongside of it running on Dwight-Lloyd product with no H. & H.—the idea being to note the tendencies of the two roasted products as to speed and reduction. The composition of the two charges used is shown in the following table:

| | H. & H. Charge, Pounds | Dwight Machine Charge, Pounds |
|---|---|---|
| Siliceous bed ore | 1,580 | 176 |
| H. & H. roast | 5,264 | .... |
| Dwight sinter | ..... | 7,040 |
| Lime rock | 1,076 | 322 |
| Scrap iron | 80 | 80 |
| Golden Curry iron ore | ..... | 382 |
| | 8,000 | 8,000 |

The results of the week's test are shown in the following tabulation:

This tabulation indicates that 2 per cent. less coke on charge, better reduction, and higher speed attended the use of Dwight machine product. It is to be noted that the Dwight machine charge was run part of the time on one furnace and part of the time on the other, in order to make a fair comparison of the Dwight-Lloyd and H. & H. products regardless of local furnace conditions. In every instance except one, the Dwight-Lloyd product gave a more favorable showing than the H. & H. This exception is indicated in the wet lead in the slag on furnace No. 2, the result here being distorted by hang-over assays, after the H. & H. charge had been changed to furnace No. 1. Eliminating these hang-over assays, the slag wet lead on furnace No. 2 for the H. & H. charge becomes 1.37 per cent. and for the Dwight 1.29 per cent.

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*Huntington and Heberlein*

| | Dec. 5 to 9 1912 | Dec. 3, 4, 1912 | Average |
|---|---|---|---|
| Furnace, No. | 1 | 2 | |
| Per cent. coke | 13.75 | 12.50 | 13.39 |
| Per cent. F. C. | 10.9 | 9.8 | 10.6 |
| Tons wet charge | 210 | 208 | 209 |
| Slag: | | | |
| Per cent. zinc | 5.0 | 4.9 | 5.0 |
| Ounces silver | 0.29 | 0.26 | 0.28 |
| Wet lead | 1.99 | 1.47 | 1.84 |
| Per cent. lead on charge | 30.1 | 30.4 | 30.2 |
| Per cent. roast | 66.2 | 67.9 | 66.7 |
| Per cent. sulphur on charge | 4.2 | 4.2 | 4.2 |
| Per cent. matte fall | 18.3 | 16.8 | 17.9 |
| Matte: | | | |
| Lead | 24.6 | 18.8 | 22.9 |
| Copper | 9.9 | 10.9 | 10.2 |

*Dwight and Lloyd*

| | Dec. 3, 4, 1912 | Dec. 5 to 9, 1912 | Average |
|---|---|---|---|
| Furnace, No. | 1 | 2 | |
| Per cent. coke | 11.57 | 11.39 | 11.52 |
| Per cent. F. C. | 9.0 | 9.0 | 9.0 |
| Tons wet charge | 216 | 224 | 218 |
| Slag: | | | |
| Per cent. zinc | 5.1 | 5.1 | 5.1 |
| Ounces silver | 0.21 | 0.25 | 0.22 |
| Wet lead | 0.72 | 1.48 | 0.94 |
| Per cent. lead on charge | 30.6 | 30.2 | 30.5 |
| Per cent. roast | 88.8 | 87.6 | 88.4 |
| Per cent. sulphur on charge | 3.7 | 3.8 | 3.7 |
| Per cent. matte fall | 15.4 | 15.9 | 15.6 |
| Matte: | | | |
| Lead | 9.8 | 17.8 | 12.1 |
| Copper | 8.1 | 9.4 | 8.5 |

It was found, in fact, that with furnace No. 2 doing poor work on the H. & H. charge it was possible to straighten out both the tonnage and assays by switching to the Dwight charge. In all cases we had good crucibles, bright tuyères, and cool tops, but it was found necessary repeatedly to take off considerable amounts of coke on the Dwight furnace, on account of persistent over-reduction.

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It was found in general that our mechanical feeding arrangements were more favorably adapted to the proper distribution of the Dwight machine product. As fast as they were indicated, however, changes were made in the distribution of both charges, and it was felt that each one of the changes made was in the nature of an improvement at the t me. The test was too short, however, to permit these changes in feeding arrangement to proceed to a point at which the H. & H. product was distributed to absolutely the best advantage. In fact, had we been able to prolong the test, we could undoubtedly have made a more favorable showing for the H. & H. by further changes in feed-floor arrangement. It is true, however, that the over-reduction on the Dwight furnace prevented the large tonnages that were clearly indicated for the Dwight charge under proper conditions.

It should be noted that in these tests both charges involved larger percentages of roasted product than were used in the Murray trials—65.8 and 88 per cent. against 37.5 and 60 per cent.

In Mr. Norton's recapitulation, five items are set down, two in favor of H. & H., two in favor of Dwight-Lloyd, and one with honors even. These same five items viewed in the light of East Helena operations compare as follows:

1. *Cost of Installation.*—Regardless of location, this is, of course, in favor of the Dwight-Lloyd apparatus.

2. *Cost of Roasting.*—During the year 1913, the Dwight-Lloyd plant at East Helena accomplished a sulphur elimination of from 10.9 to 4.6 per cent., while the H. & H. converter pots during the same period reduced from 10 to 5.1 per cent. sulphur. In this narrow range of sulphurs is seen the deterrent effect of the heavy dense lead charge at East Helena on both methods of blast roasting. The roasted material at present turned out in the H. & H. department averages 38 per cent. lead, while the Dwight-Lloyd sinter carries 43 per cent. lead. The H. & H. process is also favored to the extent of receiving the coarser Cœur d'Alêne products, while the Dwight-Lloyd beds take care of the slimes and fine concentrates. The average figures for the 12 months of 1913 show a cost per unit of sulphur eliminated, the same within a fraction of a cent, for both processes. Three hundred tons per day are handled in the H. & H. plant and about 400 at the Dwight machines.

3. *Adaptability of Charge.*—As at Murray, we recognize a greater leeway on the Dwight-Lloyd process in the physical and chemical variations possible in the charge.

4. *Metal Losses.*—Repeated tests on East Helena flues indicate, as at Murray, that lead losses are practically the same on both processes.

5. *Physical Condition of Product.*—An abundance of experience and testimony concerning the important part in reduction played by ascending blast-furnace gases makes it seem entirely logical and proper to us at

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East Helena to attribute the beneficial effect of Dwight-Lloyd product on our smelting procedure to its peculiar cellular structure, allowing the reducing action of the gases to proceed over a maximum area of contact from the moment the charge begins its downward journey from the furnace top.

The danger from the fusing of such predigested material (either Dwight-Lloyd or H. & H.) at too low a temperature in the furnace, should be and is offset, at East Helena at least, by the faster descent of the charges containing these roasted products. Although 85 to 90 per cent. of our smelting stock is made up of roasting product (50 per cent. Dwight-Lloyd, 35 per cent. H. & H.), hot tops are practically unknown.

Recent years have found us gradually and steadily increasing the amount, and improving the character, of the roasted product on charge, at East Helena, and with each step in the process better blast-furnace results have been noted. In short, it is to the roasters and not the blast furnaces that we are inclined to give credit for accomplishing a notable improvement in lead-smelting speed and general metallurgy during the last few years all over the country, and the Dwight-Lloyd process has played such an important part in this at East Helena that we are glad to add our word to the approval expressed by Mr. Norton. In going a little further in our appreciation and acknowledging a distinct superiority of Dwight-Lloyd over H. & H. product, we fully realize that the handling of 35 to 40 per cent. lead charges may present a rather extreme smelting procedure, and that these opinions as to the relative merits of Dwight-Lloyd and H. & H. roasted products may simply reflect local Montana conditions.

A summary of Murray and East Helena experience with these two roasting processes would appear, then, as follows:

| | Advantage in Favor of | |
|---|---|---|
| | (at Murray) | (at East Helena) |
| Cost of installation | D.-L. | D.-L. |
| Cost of roasting | H. & H. | Even |
| Adaptability of charge | D.-L. | D.-L. |
| Metal losses | Even | Even |
| Physical condition of product | H. & H. | D.-L. |

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## The International Lead Refining Plant

Discussion of the paper of G. P. Hulst, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 92, August, 1914, pp. 1865 to 1871.

L. S. AUSTIN, Salt Lake City, Utah.—One question that I would like to ask is in regard to the sampling kettle which stands beside the softening furnace. How is the sample taken or melted?

G. P. HULST.—The sampling kettles in question are of 45 tons capacity and are fired with coal. The lead bullion comes in on a high line in carload lots of 30 to 40 tons. The entire lot is trucked from car, weighed over scales and thrown into the sampling kettle, which has been previously cleaned. The kettle is fired and the bullion melted, then heated up until no frozen bullion remains in the bottom or adheres to the sides of the kettle. The kettle is scraped clean, sides and bottom, and the wet dross is skimmed off clean into molds. The kettle is stirred thoroughly during the period in which 24 to 30 samples of approximately ½ assay ton are taken. These samples are taken in a long-handled iron mold, providing for taking six to eight "gumdrops." The mold is inserted and heated to the same temperature as the molten metal, and gumdrops are dipped out and cooled in the mold by dipping the bottom of the mold into water. The samples should be free from fins. The gumdrops represent the contents of the kettle, and in assaying, they should not be clipped, but weighed up and the result computed. The dross bars are weighed and sawed according to old hammer and punch sample template. The sawdust represents a sample of the dross. In cases of high gold in the bullion there is a slight correction in the assay.

This method of sampling was developed at the Omaha plant of the American Smelting & Refining Co., W. T. Page, Manager, in 1903 or 1904, for adjusting differences in checking smelter sampling.

L. S. AUSTIN.—The base bullion which you treat is very clean. It has been remelted, and skimmed before shipment. How rapidly can you get it through the softening furnace, and how does it compare with ordinary base bullion?

G. P. HULST.—Twenty-four hours is the softening limit on these charges. In other words, it is charged into the furnace, heated up, and the first and second skimmings are taken off, and it goes into the kettle the next day; that is, 24 hr. elapse from the time the bullion is charged to the time it is in the kettle.

L. S. AUSTIN.—It seems to be a long time to hold such a pure bullion. I suppose, though, it comes around conveniently so that you have a time for all the other operations of refining. Isn't that the case?

MR. HULST.—Every refinery is based on those principles.

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MR. AUSTIN.—Well, in other words, you do not tap your softening furnace as soon as it is softened.

MR. HULST.—Absolutely not.

MR. AUSTIN.—Of course it is convenient for all operations to come around at the same time of the day. Now, another question: You say that you ship away the matte you make to the smelter. To what smelter?

MR. HULST.—To the Perth Amboy smelter of the American Smelting & Refining Co.

## Tests of Rock Drills at North Star Mines, California

Discussion of the paper of ROBERT H. BEDFORD and WILLIAM HAGUE, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 92, August, 1914, pp. 1807 to 1816.

W. L. SAUNDERS, New York, N. Y. (communication to the Secretary*). —Messrs. Bedford and Hague have given us a very interesting and practical paper. No better excuse for such a paper could be desired than the statement made that the expense of drilling in this mine is one-third of the cost of delivering the ore at the mill. This statement should be written in red letters, because it applies to a great many mines besides the one referred to and because it is generally true that in a mining plant, as also in papers read before the Institute, more attention by far is given to the metallurgical side of mining than to the excavation of the ore, and no study of the excavation work is properly made that does not involve a careful inquiry into the merits of the rock drill used.

Next in importance to the selection of a good rock drill, in a case where the actual drilling cost is one-third the cost of delivering the ore, is that the drill should be maintained in order and kept up to its highest efficiency. The best rock drill may run down in efficiency through accidents, wear, or other conditions which are sometimes beyond supervision or control. The machine is usually in the hands of men who know how to get work out of it, not men who know how it works. The best drill runners are not machinists, but miners. They could not tell you how the valve operates or the principle of the rotation, but they know how to get the work done. A testing machine, particularly the type described in this paper, may be of the greatest value in maintaining a uniform state of efficiency.

It must not be supposed, however, that this testing machine in itself will distinguish the value of one type of drill from another. This has been referred to in the paper, where it is shown from the record that drills developing the greatest horse power of energy were not the

* Received Sept. 1, 1914.

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most efficient in drilling capacity. This is referred to on p. 1814 and it is corroborated by the tables on p. 1813, where the foot-pound blow of approximately 40 to 45 lb. is shown to be the most efficient for this particular class of work.

At the works of the Ingersoll-Rand Co. testing machines of practically the same principle as that described have been in use for several years. Drills are tested on these machines and afterward operated in an experimental underground mine, where it has been shown that the maximum foot-pound energy does not always mean maximum drilling capacity. The efficiency of any type of rock drill depends mainly upon the proper adjustment of the air pressure to the nature of the rock. In certain rocks a slow-moving, heavy foot-pound blow will show the best results in drilling, while in other rocks the fastest drilling is done through a rapid blow of light foot-pound energy.

The testing machine, however, is a guide in showing comparative results under similar conditions and it is a valuable means by which from time to time a drill may be tested to determine whether or not it has deteriorated. A rock drill that is not up to its maximum efficiency should be examined and the cause discovered and remedied.

It is to be regretted that Messrs. Bedford and Hague did not give us a little more information concerning the tests given in Tables I and II, on p. 1808. These tables at a glance indicate that a comparison has been made between the cost of operating a water Leyner No. 8 drill and a Waugh 12-A. I feel quite sure, however, that there is no comparative test intended, because, in the first place, one machine is a rock drill of the mounted type while the other is an unmounted stoper. In these tests the Leyner drill was used for heading work, which means that a larger proportion of the holes are flat and down-holes. Such work, as every one knows, is more difficult and is harder on the steel than work, commonly known as stoping, with air drills, which is usually up-hole work, the drills clearing themselves of cuttings during operation. This point has a bearing upon the statement showing the amount of steel consumed and the cost per shift for sharpening the steel. There can be no comparative value placed upon tests of steel cost except when used in the same class of work. Furthermore, the Leyner drill was used with round steel while the stoper used the cruciform pattern. The statement is also made that 2-in. bits were used with the stopers and 2¼-in. with the Leyners.

Another point worth mentioning is that the Leyner drill (No. 8) referred to in this paper is no longer made, it being an obsolete pattern, the present type being No. 18.

The point brought out in this paper that the force of the air feed on a stoper has a very material effect upon its drilling capacity is one not generally considered, but it is of the utmost importance in stoper service.

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## The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada.

Further discussion of the paper of JAMES JOHNSTON, presented at the New York meeting, February, 1914, and printed in *Bulletin* No. 85, January, 1914, pp. 107 to 133. See also *Bulletin* No. 91, July, 1914, pp. 1473 to 1496.

THOMAS CROWE,* Victor, Colo.—The interest manifested of late in the treatment of low-grade ores, together with Mr. Clevenger's discussion of the Mill and Metallurgical Practice of the Nipissing Mining Co., prompts me to add a few remarks relative to concentration in connection with cyanide treatment of low-grade ores.

Mr. Clevenger in this discussion does not condemn concentration in this connection, but, nevertheless, the tone of his remarks would lead one to believe that his conclusions are like those of many others: that concentration is often turned to as a last resort in an attempt to improve or obtain an extraction upon an ore by the recovery and sale of the refractory portion of the ore. This, I will attempt to point out, is not always the case. Concentration in connection with cyanidation often performs an entirely different function, *i.e.*, one of saving fine grinding.

Economy being the keynote of successful treatment of low-grade ores, the problem often becomes more commercial than metallurgical, and, as there is generally a definite ratio existing between cost of operation, degree of comminution, and percentage of extraction, the grade of ore under treatment usually imposes a limit upon these factors.

With many ores grinding is the most expensive single item in their treatment; therefore, the degree of comminution is very apt to be governed by the allowable cost of operation. With most ores the degree of comminution controls to a great extent the percentage of extraction. So in the treatment of low-grade ore it often becomes necessary to sacrifice extraction, through coarse grinding for the benefit of cost, in order that the greatest ultimate profit may be obtained, and it is under these conditions that it is possible for concentration to play an important part in overcoming to some extent the effect of mesh.

The precious metals occurring in an ore are usually closely associated with the metallic portion of the ore, and as this metallic portion is generally fairly well liberated from the gangue at comparatively coarse meshes, further grinding of the ore is necessary only in order that the metallic portion may be reduced sufficiently fine that the precious metal part of it may be dissolved by cyanide solutions in a reasonable length of time.

* Non-member.

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A concentrating table under these conditions would have the effect of removing this refractory metallic portion, it being especially efficient in removing that portion which is not sufficiently fine to be readily dissolved by cyanide solution (the coarse), putting the small amount of high-grade concentrate in a separate pile where it can be dealt with by more extensive methods of treatment, it being of sufficient value per ton to justify further grinding, longer contact, and more elaborate methods, at the same time simplifying the subsequent treatment of the bulk of the ore and accomplishing the same result as though the whole mass of ore were ground to a very fine mesh.

An exemplification of the effect of concentration in connection with cyanidation in the treatment of low-grade ores may be found in the mills of the Cripple Creek district, which are treating the sulpho-telluride dump ores. Here concentration performs another function besides that of saving grinding as described above. On account of the peculiar occurrence of the values in these ores, the sulpho-tellurides occurring upon the faces and seams of the rock, when the ore is crushed to 30 mesh it is found that the sulpho-tellurides are liberated to such an extent that, after concentration and classification, the sand product of this operation is of such low value that it can be rejected as a tailing, leaving only the enriched concentrate and slime to receive further treatment.

The low treatment costs allowable by this rejection of 50 per cent. of the ore in the form of low-grade sand can be well imagined. In fact, the success of these mills in the treatment of this low-grade by-product is only made possible through the continual elimination of that material which will not withstand further treatment. This is practiced by other methods of concentration besides table concentration, such as hand sorting, coarse crushing and trommeling, etc., and I would like to make a long-range prediction that in the future low-grade milling selective methods will prevail and concentration become an important factor.

G. H. CLEVENGER, Palo Alto, Cal.—The whole question of ore treatment is, of course, an economic one and frequently our pet metallurgical theories have to be sacrificed upon the altar of greatest ultimate profit. If the recovery of a portion of the gold and silver can be more economically made by concentration than by solution in cyanide, obviously concentration should be practiced. The case cited by Mr. Crowe is an unusual one in that his strongest argument for concentration is that the sand can be rejected without further treatment if the concentrate is removed. It must be remembered, in this connection, that the ore treated runs less than $3 per ton and that a large proportion of the minerals carrying the gold occur along the cleavage planes. Even with the same character of ore, if of considerably higher grade, it would not be possible to reject the sand without incurring a serious loss. The

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character of many of the low-grade ores of other districts would render this type of practice impossible. Lack of suitable mill sites for a large expanse of leaching tanks, together with favorable smelter contracts, are factors in the Cripple Creek district which are not without their influence.

There are a number of possible variations of concentration in conjunction with cyanidation, the more important of which are:

1. Crushing of the ore in water; concentration, either directly or following another recovery operation, as amalgamation; rejection of the tailing, and cyanidation of the concentrate. This method is most suitable for use upon very low-grade ore or tailing. A good example of such practice is the Treadwell, where the tailing from the amalgamation of the very low-grade ore treated could not be profitably treated directly by cyanidation; but cyanidation of the concentrate recovered from the tailing by concentration returns a handsome profit.

2. Crushing of the ore in cyanide solution; concentration, followed by cyanidation of the sand and slime. Concentrate treated by one of three methods:

(*a*) Shipment of the concentrate to the smelter.

(*b*) Special local treatment of the concentrate.

(*c*) Special treatment of the concentrate stream, as, for example, fine grinding or amalgamation, etc., and return of the concentrate stream to the balance of the pulp for cyanidation.

Method (*a*) of concentrate disposal has been very generally practiced in the past, but care must be exercised in adopting this practice, for the reason, as I have previously pointed out, that one may pay the smelter rather dearly for recovering gold and silver recoverable by cyanidation under proper conditions.

Method (*b*), involving special chemical treatment or roasting prior to cyanidation of concentrate, has been practiced and, under certain conditions, may be advantageous.

Method (*c*) has its adherents and, under favorable conditions, may present certain advantages. However, there is at present a tendency to revert from this method to (*b*), even when it is possible to obtain as high a recovery by (*c*), for the reason that, if the concentrate residue is kept separate, it may later become a valuable asset.

3. Crushing of the ore in water or cyanide solution and separation of the pulp into sand and slime; concentration of either one or both; rejection of either sand or slime and cyanidation of the other product. An example of this is the old Homestake practice, where, after crushing in water and amalgamation, the slime was rejected, as it was too low grade for profitable treatment until the development of the Merrill filter press. The sand in this case was treated by cyanidation without

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concentration. At the Portland Victor mill the sand is rejected after concentration and the slime treated by cyanidation.

ALLAN J. CLARK, Lead, S. D.—A point that might be made against aluminum precipitation is that the necessity for free alkali in some quantity would prohibit its use in connection with the treatment of ores where a very low protective alkalinity was necessary for the best results regarding either extraction or cyanide consumption. A good example of this is present Homestake practice, where an extremely low protective alkalinity has been found advantageous.

G. H. CLEVENGER.—Mr. Clark's experience with Homestake ore has also been borne out by my own with certain silver-gold ores, where the best results were obtained with a low alkalinity. However, there are exceptions to this.

## Chloridizing Leaching at Park City

Discussion of the paper of THEODORE P. HOLT, presented at the Salt Lake meeting August, 1914, and printed in *Bulletin* No. 91, July, 1914, pp. 1699 to 1708.

F. S. SCHMIDT, Salt Lake City, Utah.—Any furnace that can make a chloridizing roast to yield an extraction of 92 to 93 per cent. and do this at a cost of less than 24c. per ton in a 10-ton unit opens great possibilities for the treatment of certain classes of ores. This cost does not include the salt and will be reduced to about 16½c. per ton in the 35-ton furnaces which are now being designed. On a trial run the roaster treated 9.6 tons of Ontario stope fillings crushed to pass a ¼-in. opening. The ore is well adapted to chlorination with the exception of insufficient sulphur and occasional days of high lime, but is not adapted to any other process. The following is an analysis of the crude ore: Ag, 9.52; Au, 0.029 oz.; Pb, 0.71; Cu, 0.11; SiO, 79.4; Zn, 0.57; S, 1.00; Fe, 7.9; Mn, 0.89; CaO, 1.20 per cent. The composition of the mix was:

| | Per Cent. |
|---|---|
| Dry crude ore | 83.4 |
| Moisture | 4.4 |
| Salt | 8.3 |
| Coal dust | 2.3 |
| Pyrite, 70 per cent. pure | 1.6 |

To determine the amount of volatilization, the mix was roasted in the usual way; an analysis is as follows: Au, 0.025; Ag, 9.0 oz.; Pb, 0.72; Cu, 0.13; SiO, 72.2; Zn, 0.95; S, 0.7; Fe, 7.3; Mn, 0.93; CaO, 0.73 per cent. From results obtained in an experimental roaster it has been determined that the loss in weight sustained by the mix in roasting

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averages about 7 per cent. The loss may be said to be the weight of the coal, moisture, and one-half the salt. The percentage of sulphur is inconsiderable. Using the factor 93 per cent. on the weight and calculating the analysis from the crude heads for comparison, we have: Au, 0.027; Ag, 8.89 oz.; Pb, 0.67; Cu, 0.11; SiO, 74.5; Zn, 0.54; Fe, 7.4; Mn, 0.84 per cent. The sulphur naturally cannot be figured. It will be seen from this that there is no measurable volatilization of the silver or gold. As a further check, if the assumption is made that the silica is constant in weight and the relation calculated between the percentage of silica and the ounces of silver in both the crude and the roasted mix, it will be found that this again shows there is no volatilization.

To determine the uniformity of roasting, nine 100-lb. samples were taken at regular intervals by cutting the discharge stream. Small samples of each of these were kept separate and a 24-hr. leaching test was made under approximate mill conditions. In all the following leaching tests the extraction can be calculated direct on the roasted mix, due to the absence of volatilization. Experiments were made to determine the loss in weight of the roasted mix in leaching and the average result is 12.5 per cent., which again can be checked by comparison of the silica percentage in the roasted mix and in the leached tails, which in this case gives 13.2 per cent. The extractions on these nine interval samples varied from 91.2 to 93.4 per cent., the tails varying from 0.6 to 0.9 oz. silver; the average roasted mix was 9.06; average tails, 0.78; and average extraction, 92.4 per cent. These results show the extraction to be high and uniform.

The next point is the coarseness of crushing for this furnace. A screen analysis of the raw mix, using 8, 14, 20, 40, and 60 mesh screens, shows that the values increase with the fineness from 6.8 to 12 oz. By weight, 34 per cent. is plus 8 mesh and 9.5 per cent. is through 60 mesh; 28.0 per cent. of the value is plus 8 mesh and 13.8 per cent. is in the through 60 mesh. A screen analysis of the roasted mix shows again that the values increase with the fineness. We now have 41.2 per cent. of the weight on the plus 8 mesh and only 5.4 per cent. in the fines, while the plus 8 mesh has 31.2 per cent. of the values and the fines 8.1 per cent. It will be noted that in the roasting the coarser sizes have increased at the expense of the fines. This is what would be expected from an inspection of the roasted product and is a very desirable result as an aid to the leaching.

A leaching test was made on these sizes separately to determine whether it would pay to do finer crushing by determining the different percentages of extraction. The plus 8 mesh gave an extraction of 89.8 per cent. and the fines gave 96.7 per cent.; the tails in the plus 8 mesh gave 0.8 per cent., while the tails of the fines gave 0.5 per cent. It will be seen that while the fines give the best extraction, the extraction on the coarser sizes is really higher than would have been expected. This result

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is probably due to the nature of the furnace, with its 4 ft. thick bed of ore. It will also be seen that the question of finer crushing is solely one of the value of the ore.

To determine the relative velocity of leaching of the different sizes, a test was made by interrupting the leaching action and making a screen analysis of the imperfect tailings. For this purpose only 3 parts by weight of mill solution to 1 of ore were used instead of the usual 6 of solution to 1 of ore. The coarser products gave tailings running from 1.2 to 1.8 oz., while the fines retained a value of 3.0 oz. The percentage of extraction on the plus 8 mesh was 79.7, and on the fines 80.7. These partial tailings were then leached again to the full extent and the extraction on the plus 8 mesh increased to 87.3 and on the fines to 95.5 per cent. The tailings decreased regularly from 1.0 oz. in the plus 8 mesh to 0.7 oz. in the fines. The second leaching reduced the value of all sizes, but mainly the fines, which resist the leaching the most. The question of fine grinding is therefore important. From tests made on numerous ores it is safe to say that this type of furnace will give more effective chloridizing on coarser sizes than furnaces using a thin bed of ore.

Another point in connection with this roaster is the ease with which it is possible to get an excellent physical constitution of roasted ore for the subsequent leaching. The gathering up of the fines in roasting and their release only as the values are leached out is well shown by a study of the screen test of the tailings. The percentage of value of the fines is 16.1, while if the screen analysis is made after only half the solutions have passed through we have 7.9 per cent. The percentage of weight is the controlling factor, because the tailings run very uniform for the different sizes. In the crude ore the weight of the fines is 9.5 per cent.; in the roasted mix this is reduced to 5.4, showing the effect of gathering the fines. In the partly leached ore some of the fines have been freed, to the extent of 9.7 per cent., while in the thoroughly leached mix the amount has been increased to 15.5 per cent. An analysis of the tailings gave the following: Au, 0.005; Ag, 0.75 oz.; Pb, 0.41; Cu, 0.06; SiO, 83.2; Zn, 0.8; S, 0.2; Fe, 4.5; Mn, 0.62; CaO, 0.52 per cent. It will be seen from the low tenor of copper in this ore that it is very difficult to get sufficiently accurate determinations of the copper for close figures on extraction. But extractions of over 90 per cent. have been obtained on ores of better defined values. The mill solutions contain 8 lb. metallic lead per ton of solution, the profitable extraction of which has not yet been worked out. The lead is in the form of chloride.

It would seem from my results and an analysis of the cost data that this furnace is a cheap and efficient chloridizer; that it is well adapted to the treatment of gold-silver-copper ores having a siliceous gangue, and opens a field of great possibilities.



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OLIVER C. RALSTON,* Salt Lake City, Utah.—Mr. Holt has given some costs for roasting (16.5c. per ton) which are so surprisingly low that they call for comment. Mr. Holt has informed me that this cost does not include the cost of salt, which is a variable factor rather hard to approximate. This, taken in conjunction with the low price paid for coal dust, makes his low figure more easily comprehended. However, it might be well to call attention to the fact that although this coal dust is a waste product, loaded on to cars for the cost of the labor necessary to do it, if chloridizing leaching reaches a very extensive application the coal men will be liable to feel that they are entitled to a little more than 25c. per ton for this product. Still, if the Holt-Dern roaster can roast ore at 25c. per ton, allowing for higher cost of coal dust, there will be nothing to complain of, and Messrs. Holt and Dern are to be congratulated on their solution of the mechanical difficulties in applying this kind of a roast.

Another point about this roaster is that it seems to be designed along the line of good counter-current principles so as to act as a heat exchanger. The fuel is burned in the ore, as in the blast furnace, affording a maximum of heat absorption by the ore. We find the air blast being preheated by the hot roasted ore before it reaches the zone of combustion and then the hot gases of combustion in their turn preheat the ore before it reaches the combustion zone. And, moreover, the machine is simple and inexpensive. It would look as though this roaster should have a future.

Outside of the new roaster, the process will be found to be nothing but a new modification of the old Augustine process, used in days past in the hydrometallurgy of silver. Nothing but the roaster is subject to patent and the rest of the process is old enough to be free to all. In Utah there is a great deal of ore that is amenable to this treatment and doubtless more mills will be built now that we are getting some assurance that the process pays. Likewise there is plenty of such ore elsewhere in the United States. It is an interesting sight to see a process as old as the Augustine. which had seemingly been hurled into Aphelion by the introduction of cyaniding, now coming back to claim its own.

Two drawbacks, however, are to be met with, and one of them Mr. Holt has already mentioned; namely, the deleterious effect of lime. There is plenty of ore within a radius of 50 miles of where we sit which will be found to contain a fatal amount of lime. The other drawback is that, as of old, we cannot recover the zinc of the ore. Mr. Holt informs me that he does not know whether the zinc of the ore is chloridized and goes into solution or not, but it ought to. At one of the plants of the Pennsylvania Salt Co., where they have roasted pyrite cinder for 25 years, the zinc content is definitely known to be chloridized and accumulates in the leach solution, and for years was finally discarded into the sewer at the rate of

* Non-member.

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several thousand pounds of zinc per day. Recently efforts have been made to recover this zinc. The consumption of the salt by this zinc and its fouling of the solutions and ultimate loss, while not serious at the Ontario, is bound to become a serious problem with the further app ication of the process.

There are a number of questions which I would like to put to Mr. Holt and they are as follows:

It is stated in the paper that ores amenable to this type of treatment are those with high percentages of silica, iron, or manganese. Is there a lower limit of silica allowable? Must the iron be as sulphide or oxide? What is the advantage of the presence of pyrite, is it necessary, and if so in what percentages? How can high manganese cause the ore to chloridize well?

What is the maximum allowable amount of lime?

How can one determine the amounts of salt and of fuel necessary for this roast?

The paper states that the driers reduce the moisture to 5 per cent., but that there is considerable variation from this amount. How much is the variation? In other words, how much is the allowable variation in moisture content of the ore on roasting?

How many tons of solution and of wash water are used in the Ontario mill to one of ore?

Does the Holt-Dern roaster handle a finely crushed ore?

THEODORE P. HOLT, Park City, Utah (communication to the Secretary*).—In reply to Mr. Ralston's discussion, I must confess my inability to answer all his questions. Some of them would require generalizations too broad for our present limited experience.

Silica, iron, and manganese are indicated as desirable gangue minerals in an ore to be chloridized. Silica at proper roasting temperatures is for the most part inert. Iron and manganese are generally recognized as active chloridizing agents. The iron is desirable either as oxide or sulphide, preferably as sulphide. Pyrite is very desirable though not always necessary. In the first place, it is an active chloridizer of silver and copper. Secondly, it converts $CaCO_3$ and CaO into inert $CaSO_4$. In the third place, it is valuable as a fuel and when present to the amount of 6 per cent. or over no additional fuel is required.

The deleterious effect of lime in chloridizing metals is well recognized. When the chloridizing roast is followed by an acid-salt leach, as in our case, it is doubly essential that all $CaCO_3$ and CaO be converted into neutral compounds; otherwise the consumption of acid would be prohibitive. We have secured satisfactory results on an ore carrying 23 per cent. CaO by analysis. But since the ordinary analysis reports the

* Received Sept. 21, 1914.

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calcium combined as sulphate and silicate as well as the carbonate, it is of limited value in determination of probable results.

The "amounts of salt and fuel necessary for the roast" can be determined by experiment on a small sample. The amount of both reagents in the mill will be slightly less than in the small laboratory roaster.

The moisture in the mixed ore, while essential, may be subject to considerable variation without serious effects. With the new roaster the variation may be from 4 to 9 per cent. in many cases without seriously affecting the results.

Mr. Ralston asks if the Holt-Dern roaster can handle finely crushed ore. This depends on the characteristics of the individual ore, and also on the size he designates as "finely crushed." We have found some ores to work well when crushed to 20 and even 30 mesh. However, we must bear in mind that one characteristic of this roaster is its ability to chloridize well coarsely crushed ore, thus avoiding the expense of fine grinding. This has been demonstrated in a great many ways. I will mention one by way of illustration. A large sample of ore crushed to pass ¼-in. screen was cut in two samples. Sample No. 1 was passed through the Holt-Dern roaster carrying a 4-ft. column. Sample No. 2 was roasted in a bed only 12 in. deep. Leached under the same conditions, sample No. 1 gave 92 per cent. extraction, while sample No. 2 gave only 83 per cent. extraction. The difference is due to the fact that sample No. 1 was in intimate contact with the chloridizing gases at a chloridizing temperature for over 4 hr., while for sample No. 2 these conditions lasted less than 1 hr. The prolonged heating is effective in opening up the coarse particles of ore, thus exposing any inclosed mineral particles.

Since preparing the paper on Chloridizing Leaching at Park City a successful silver precipitator has been put in operation at the plant. This makes possible the marketing of a high-grade bullion in place of the base silver precipitate, thus materially reducing the marketing expense. Before entering the iron boxes as shown in the flow sheet, the pregnant solution is first pumped to two cement tables, each 8 ft. wide by 40 ft. long. These tables have a slope of 1 in. to the foot and carry a thin layer of scrap copper, upon which the silver precipitates. Practically all the silver remains on the table until clean-up time. At the lower end of the tables is a settling tank. Twice each month the silver product is sluiced into this tank and the excess water decanted. It then passes to a filter tank, where it receives a final wash. The dried product carries 60 to 70 per cent. silver and melts easily to bullion 970 fine.

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## Mining Claims within the National Forests

Discussion of the paper of E. D. Gardner, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 91, July, 1914, pp. 1467 to 1471.

H. V. WINCHELL, Minneapolis, Minn.—There are two questions I should like to ask: First, does the Department still adhere to the decision made in the East Tintic case? Second, does the Department require placer ground to be of such value that, in the opinion of the Forest Service employees, it will pay to work, before admitting the location to be valid?

T. C. HOYT,* Ogden, Utah.—With reference to the first question: the Department does not hold to that decision, as it was reconsidered by the Secretary of the Interior, and the Department is governed by the decision of the Secretary of the Interior, on those points always. The Department of Agriculture makes no decision as to what constitutes a sufficient discovery. The mineral examiners simply report their findings and opinions. These are transmitted to the chief of the field division of the General Land Office, who in turn transmits the reports to the Commissioner of the General Land Office, who decides whether the report raises an issue requiring a hearing. If a hearing is ordered, expert testimony is given on the question of the discovery, as well as the evidence of practical people who operate in the vicinity of the claim in question. As to the extent of placer values considered sufficient to justify a location, my answer will need a little amplification. I couldn't answer it directly for the reason that sometimes people resort to placer claim location as a means of acquiring land for other purposes than placer mining, which makes it absolutely necessary that the government give close attention to the question of whether good faith has been shown, as it is the evidences of good faith that largely determine whether a report adverse to a claim shall be made. That is a difficult point to cover sometimes. For instance, in the region I have recently visited there are some 17 miles of a valley, from $\frac{1}{4}$ to $\frac{3}{4}$ mile in width, all covered by placer locations. It is valuable stock-grazing land, and is stocked at the present time. It also contains a stand of timber, and is on the watershed of an important stream—important from the fact that it furnishes the water supply to government storage reservoirs, and to pending reclamation and Carey Act projects. In this case the placer claims will not be disturbed as mere locations. They will simply remain there, and there will be no action with reference to them until the claimant asks for patent or, before patent, begins to denude that watershed and remove that timber. Then the government would direct that the mineral experts go on to the ground, examine it, prospect it, and submit their report, with a view to seeing if

*National Forest Service. Non-member.

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the claims were really valid and being used for actual mining purposes. Placer claims in the National Forests to-day are covering power sites, homesteads, dipping corrals, sawmills, summer resorts, and a good many other things, but a claim that is located as a *bona fide* placer claim and used for purposes consistent with mining will not be disturbed by the Forest Service.

This is an illustration of our attitude. In one of the National Forests where we had a withdrawal by the Secretary of the Interior for administrative purposes, the tract supported a cabin, pasture, and small field. Subsequently placer locations were made to cover this site and additional land up and down the canyon. The proposition was made to the locators, "If you will proceed with actual placer mining up to the point where you demonstrate to yourselves and to disinterested parties, competent to pass upon it, that you have a paying placer mine, we will get out of the way and allow you to mine all over the area, but we don't feel that we can do that on the showing as now given. Our experts have examined the land, and do not report sufficient values or sufficient prospects on your part to determine whether you have values enough to justify our giving up this land." This land was adjacent to a growing mining camp, and there were various reasons why we desired to know whether there were values there to justify before permitting it to pass to patent. These claims were not disturbed until the application for patent was made. Then it was a question of whether this 1½ or 2 miles should be alienated, or whether we should wait until values were proved. That case is now pending, and our proposition still stands. That is as near as I can illustrate the attitude of the Forest Service on that question.

MR. WINCHELL.—In the case of a quartz locator within the National Forest, where the report of the National Forest employees is to the effect that he has not made a valid discovery, what procedure is taken against him? Does he have further time and opportunity to perfect it, or is some action taken to oust him?

MR. HOYT.—That answer will require a little amplification, too. Recently a mineral examiner of the Land Office submitted a report on a claim on the National Forest which was adverse. Feeling that more expert testimony was desirable before deciding the issue, the examiner transmitted the request to me, and I detailed an examiner to examine this claim. Our examiner hunted up the locator, or present owner of the claim, and asked the claimant to accompany him on the examination, which was done. The owner of the claim said: "Now, I want to have your advice in this matter. If you think that the reports which you will have to submit, under the conditions as they now exist, are such as to jeopardize the probabilities of my getting a patent, I would like to have further time," and that proposition was accepted. The examiner trans-

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mitted that request with his report to me, and I transmitted the request to the chief of the field division, with the recommendation that no action be taken, and that the applicant be not required to defend his claim at the present time, but be given such time as he thought desirable in order to make a better showing, and mind you, this was on an application for patent. It should be borne in mind that these lands have been declared by the President of the United States, under authority of Congress, to be lands valuable for a certain purpose, a certain public purpose; therefore, before alienation is sanctioned, the Department having jurisdiction of the National Forest administration has to look carefully into these questions. The examination of mining claims in the National Forest by the Forest Service came about in 1906, when the Secretary of the Interior was petitioning for additional appropriations to examine lands. Congress did not appropriate to meet the requirements to the extent that the Secretary of the Interior thought desirable, so the President suggested at the time, "Can you not use the Forest officers throughout the National Forests in this respect?" and he directed the Secretary of Agriculture and the Secretary of the Interior to confer on this matter, and see if they could bring about some arrangement. The result was that the Secretary of the Interior instructed the Commissioner of the General Land Office to transmit instructions to the Register and Receiver, in all cases of application to patent lands in National Forests, to notify the Forest Service. The reports as to mining claims may in the first instance come merely from a ranger, who may not be qualified to pass on technical questions, but his report is placed in the hands of the mineral examiner, who is a man of considerable experience throughout the West, the one in my office at the present time being a graduate of the Columbia School of Mines. He examines the report, and if there are sufficient evidences of good faith, the claim is passed without question, the chief of Field Division being advised that there is no protest against the allowance of a patent in that case. If the evidence of good faith is manifest, we don't bother with technical examinations, or don't raise technical questions, as Mr. Gardner indicated in his paper; but if there seems to be a lack of compliance with law then the mineral examiner is directed to examine that claim, but he waives technicalities in every case where evidence of good faith is manifest. The Commissioner acts upon the report, and no location of a mining claim in a National Forest is interfered with until he considers the report, or there is something in connection with it manifesting bad faith, failure to comply with the law, or interference with the administration. Sometimes we have had groups of claims placed right where examinations were being made with a view to sale of the timber, though the timber-sale contract had not been executed. In those cases we had the mineral examiner examine the claims carefully. He took very careful note of everything indicating the possibility of values and which threw any light upon the

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question of good or bad faith of the locators. In some cases we have had to go to the extent of asking for a hearing, with the result that the claims were canceled, and we proceeded with the sale of the timber.

D. W. BRUNTON, Denver, Colo.—In a case where two contiguous claims are held by the same owner, one of which has timber on it, and the other hasn't any timber, can a man operating a mine take the timber from the timber claim and use it on the other claim? I ask this because of a complaint that was made of a man who had two parallel claims in Idaho, both of which were being worked and producing, one claim being covered with timber, and the other absolutely bare. He claimed that the Forestry Department compelled him to purchase timber from them instead of taking it from the parallel contiguous claim which he owned, and which was covered with timber.

MR. HOYT.—The rule is that it is allowed without question; but it has happened in many instances that operators during the prospecting period of their mining have located surrounding ground with timber on it, and have made no effort whatever to develop that particular ground as a mine, but have simply used the timber off of it to develop a claim having actual mineral value. I can't understand why purchase was necessary in the case mentioned, because if the miner was simply prospecting he wouldn't have to purchase the timber, but if he has a paying mine then we would not allow him to locate timber land under mining claims without going ahead with *bona fide* development and showing mineral value on the timbered claim. In case the timber were required for a paying developed mine, they might require him to purchase at the rates prevailing in that immediate vicinity instead of allowing it to be taken from a located claim not showing mineral values.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OE MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

# An Improved Form of Cam for Stamp Mills

BY ARTHUR B. FOOTE, GRASS VALLEY, CAL.

(New York Meeting, February, 1915)

THE cams at present universally used in stamp mills lift the tappets with an involute form of curve, to which the surface of the tappet is always tangent; moreover, the line of contact between tappet and cam, if produced, would pass through the center of the stem. This is no doubt a desirable feature, but the writer has long believed that it would be much more desirable if the cam were to pick up the stamp without shock, and gradually increase the upward velocity of the stamp throughout its upward movement. The involute form of cam attempts to impart instantly a considerable upward velocity to the stamp, starting it from a state of rest, and the result is a destructive blow, a great deal of noise, and much wear and vibration.

The writer has developed a form of cam which will lift the stamp with a motion similar to that of the piston of an engine between the end and middle of its stroke; in other words, harmonic motion, from the point of zero to maximum velocity. The curve will give this ideal motion only when the stamp is set for the exact drop for which the cam was designed, but the improved cam will not be as bad as the involute cam until the drop is reduced one-half. In other words, if a cam is designed for a 6-in. drop, it will be some improvement over the usual form of cam until the drop is only 3 in.

Any cam, whatever its form, must of course lift the stamp in the same number of degrees of revolution; therefore, with this new form, since the stamp starts more slowly on its upward course, it must end up by going faster, the average speed being the same as with the involute cam. The stamp, after it is no longer in contact with the cam, keeps on going up a certain distance, which is easily calculated. It is therefore possible to hang up a stamp without a cam stick, if the fingers are not more than $\frac{1}{8}$ in. longer than necessary to hold the tappet above the cam.

With this design of cam the surface of the tappet is not tangent to the surface of the cam throughout the lift. The possible consequences of this were studied with a full-size model, and did not seem serious, as

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the engagement between the surfaces was an easy sliding motion, instead of a blow. If the drop is shortened, the blow becomes more and more pronounced, but the surfaces also become more and more nearly tangent.

Five cams of this new design have been running in one of the mills of the North Star Mines Co. now for over a month, fulfilling every expectation of the writer. Holding the hand on the tappet it is impossible to feel the cam strike the tappet, although the mill is running 107 drops per minute.

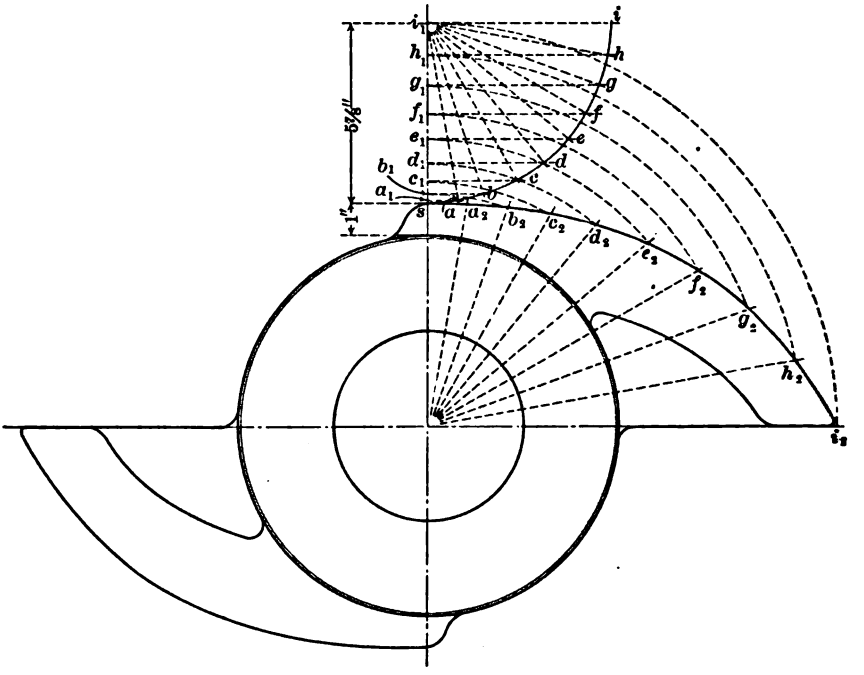


FIG. 1.—IMPROVED FORM OF CAM FOR STAMP MILLS.

Fig. 1 shows the method of laying off the curve of the cam which will give approximately harmonic motion to the stamp. Spaces $s\ a_1$, $a_1\ b_1$, $b_1\ c_1$, etc., represent the distances traversed by the stem in equal intervals of time.

The writer wishes to acknowledge valuable assistance rendered by Charles W. Taylor, President of the Taylor Foundry & Engineering Co., of Grass Valley.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

# A Study of the Chloridizing Roast and Its Application to the Separation of Copper from Nickel

BY BOYD DUDLEY, JR.,* STATE COLLEGE, PA.

(New York Meeting, February, 1915)

THE material presented in this paper is an abstract of a thesis submitted by the writer to the faculty of the Massachusetts Institute of Technology, as part requirement for the degree of Master of Science. The investigation was undertaken at the suggestion of Prof. H. O. Hofman, and it was conducted, under his direction, in the laboratories of the Mining Department of the Institute.

The purpose of the work was, first, to study some of the reactions that have been considered as a part of the mechanism of the chloridizing roast, and second, to study the conditions under which nickel oxide and copper oxide may be chloridized. While the production of chlorine gas by various reactions during the chloridizing roast may not be primarily useful in the chloridation of metal oxides and compounds, it nevertheless serves a useful purpose in preventing the dissociation and decomposition of chlorides once they have been formed; for example, the dissociation of cupric chloride into cuprous chloride and chlorine. Therefore it was thought that a study of some chlorine-producing reactions would prove of interest, and the first series of experiments dealt with this phase of the subject. The object of the latter part of the work was to show the possibility of treating heavy sulphide ores, such as those of Sudbury, Ontario, containing much iron and small amounts of copper and nickel, by chloridizing and removing the copper, leaving an iron-nickel oxide product suitable for smelting in the blast furnace for nickel-bearing pig iron. The chloridizing of the copper would, of course, be preceded by a roast for the removal of sulphur, the sulphur oxides being available for the manufacture of sulphuric acid, if under the local conditions this product would be of commercial value.

## REACTIONS OF THE CHLORIDIZING ROAST

*Reaction of Sodium Chloride, Sulphur Trioxide, and Oxygen.*—The first reaction studied was that involving the production of chlorine

* Non-member.

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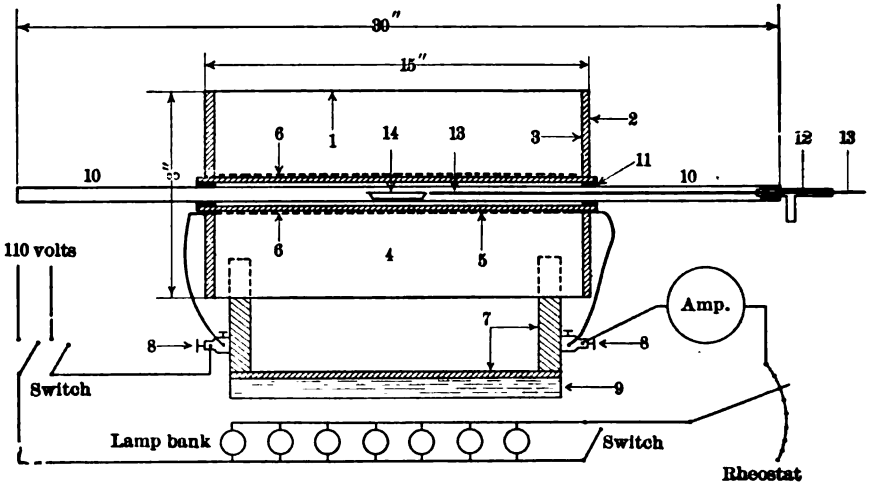
from sodium chloride by the action upon it of sulphur trioxide and atmospheric oxygen. This reaction may be expressed by the equation, $4\ NaCl + 2\ SO_3 + O_2 = 2\ Na_2SO_4 + 2\ Cl_2$. In the roasting furnace the sulphur trioxide is derived from the decomposition of sulphates that have been formed in the roasting ore, and the oxygen is supplied by the air in contact with the roast.

The action of sulphur trioxide alone upon sodium chloride has been studied by E. Kothny,[1] who concludes as follows: By treating sodium




1. Cylindrical shell of galvanized iron.
2. End of shell of galvanized iron.
3. Asbestos board 1/4 in. thick.
4. Magnesia.
5. Porcelain tube about which resistance tape is wound. Internal diameter 1 in.
6. Resistance winding of "Excello" alloy tape, 5.3 mm. wide by 0.4 mm. thick.
7. Asbestos board.
8. Binding posts.
9. Wood board.
10. Glazed porcelain combustion tube. Internal diameter 0.6 in.
11. Asbestos packing.
12. Glass T through which pyrometer tube and air enter 10.
13. Silica pyrometer tube.
14. Boat containing material to be heated.

FIG. 1.—CROSS-SECTION OF ELECTRIC RESISTANCE FURNACE.

chloride with sulphur trioxide there probably results $NaClSO_3$, which at ordinary temperatures is stable only in an atmosphere of sulphur trioxide, and decomposes at 150° C. with the evolution of sulphur dioxide and chlorine. When the sulphur trioxide is diluted with air or carbon monoxide the salt decomposes at ordinary temperatures. At a red heat sulphur trioxide and sodium choride react and give sodium sulphate, sulphur dioxide, chlorine, and probably sodium pyro-sulphate.

[1] *Berg und Hüttenmännisches Jahrbuch*, vol. lviii, p. 350 (1910).

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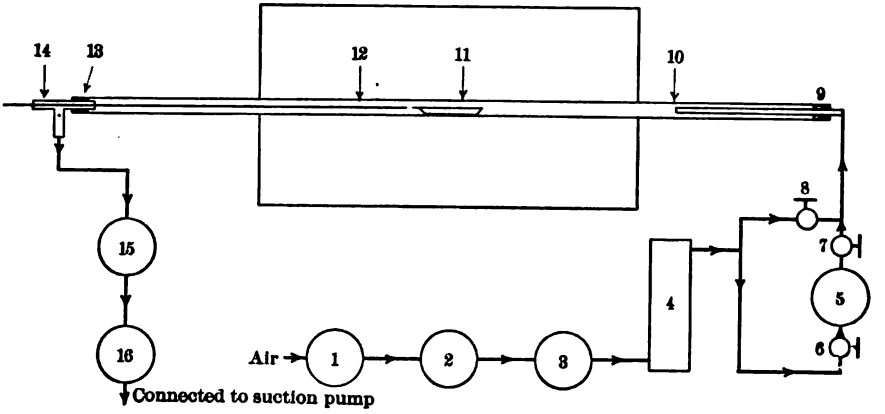
As was stated above, the experiments herein described involved the action of a mixture of sulphur trioxide and air upon sodium chloride; they therefore differ from those of Kothny, and more nearly approximate the conditions obtaining on the hearth of a roasting furnace. In these experiments it was not proved that the atmospheric oxygen actually enters into the reaction, nor was any attempt made to determine the equilibrium conditions of the reaction. It was considered sufficient to show that dry sulphur trioxide mixed with air reacts with




1. Wash bottle containing potassium permanganate solution.
2. Wash bottle containing potassium hydroxide solution.
3. Wash bottle containing sulphuric acid.
4. Drying tower containing phosphorus pentoxide.
5. U-tube containing sulphur trioxide and phosphorus pentoxide.
6, 7, 8. Glass stopcocks.
9. Plug of litharge-glycerine cement.
10. Porcelain combustion tube.
11. Porcelain boat containing sodium chloride.
12. Silica pyrometer tube.
13. Rubber stopper.
14. Glass T through which pyrometer tube and gases pass.
15 and 16. Glass railroad tubes for absorbing chlorine.

FIG. 2.—APPARATUS FOR SECURING DRY MIXTURE AND INTRODUCING IT INTO FURNACE.

sodium chloride to produce chlorine, and to show the effect of temperature upon the rate of reaction.

*Method of Conducting the Experiments.*—In the first experiments sodium chloride contained in a porcelain boat was heated in an electric tube furnace, through which was passed the mixture of sulphur trioxide and air. The rates at which the chlorine was evolved at different temperatures were determined by passing the gases through a solution of silver nitrate, and weighing the silver chloride precipitated in a known time.

Before placing the salt in the furnace it was heated to a temperature slightly below its melting point, after which the lumps formed during

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the ignition were broken in an agate mortar. In this way decrepitation in the furnace, with the consequent loss of material from the boat, was avoided. The salt was not pulverized, but was left coarse in order to expose as great a surface to the action of the gases as possible.

The details of the construction of the furnace and its electrical connections are shown in Fig. 1.

It was important to secure a mixture of sulphur trioxide and air free from water, because, as a result of the presence of water vapor, hydrochloric acid would be produced, and this would mask the tests for chlorine that could be readily made under the conditions of the experiments. The method of securing the dry mixture and of introducing it into the furnace is shown in Fig. 2. The train of apparatus was

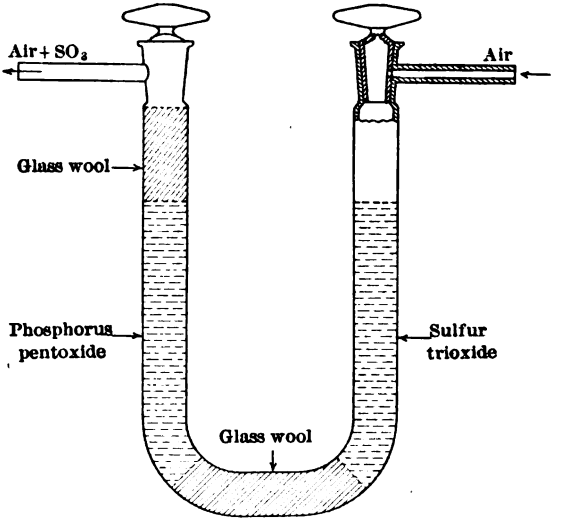


FIG. 3.—SULPHUR TRIOXIDE CONTAINER, 5 OF FIG. 2.

assembled as shown, the air being drawn through a series of wash bottles containing potassium permanganate solution, potassium hydroxide solution, and concentrated sulphuric acid, then through a 10-in. glass absorption tower containing phosphorus pentoxide, in the order mentioned. From the drying tower the air could be admitted directly into the furnace tube through the stopcock (8); in this manner the tube and its contents could be thoroughly dried. By closing (8) and opening (6) and (7) the air could be drawn through the sulphur trioxide container, which was a U-tube containing solid sulphur trioxide, phosphorus pentoxide, and glass wool arranged as shown in Fig. 3. All of the connections between the sulphur trioxide container and the furnace were made by blowing glass tubes together, thus avoiding the use of rubber or other

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connectors that might be attacked by the gas. The stopcocks were lubricated with syrupy phosphoric acid. No attempt was made to regulate closely the composition of the gas mixture entering the furnace. The air passing through the U-tube, mixed with the sulphur trioxide vapor and carried it away, the concentration of the mixture depending upon the vapor tension of the sulphur trioxide and the rate of flow of the air. Since the room temperature, which governs the vapor tension of the sulphur trioxide, and the rate of flow of the air were both practically constant in all cases, it is safe to assume that the composition of the gas mixture was practically the same in all of the experiments.

Upon leaving the furnace tube the gases passed through two railroad tubes containing a solution of silver nitrate, made by dissolving 3.5 g. of the salt in 250 cc. of nitric acid and diluting to 500 cc. with distilled water. It was found that this solution would absorb chlorine, forming a precipitate of silver chloride, without allowing the sulphur trioxide and dioxide present in the furnace gases to interfere by precipitating the silver as sulphate and sulphite, respectively.

Preliminary experiments with this apparatus, conducted by introducing the boat of salt into the cold furnace and then slowly raising the temperature while the gas mixture was being passed through the tube and into the absorption tubes, showed that the evolution of chlorine began with sufficient rapidity to be detected by the indicator at about 500° C. When the temperature was less than this the indicator showed no cloudiness.

In order to determine the effect of temperature on the rate of the reaction, a number of tests were made in which the temperature was the only variable. A porcelain boat containing 1 g. of sodium chloride that had been previously ignited was inserted into the hot furnace and allowed to come to furnace temperature, a current of dry air being meanwhile passed through the tube. At the end of about 10 min. the mixture of sulphur trioxide and air was admitted to the tube and was allowed to pass for 15 min., at the end of which time dry air was again passed for 10 min. The absorption tubes were then disconnected from the furnace tube and the boat was removed. The silver chloride that had been precipitated in the absorption tubes was removed, filtered from the solution, washed, ignited, and weighed. The salt was removed from the boat, and dissolved in water acidified with hydrochloric acid. After heating the solution to boiling an excess of barium chloride solution was added, and, after standing, the resulting barium sulphate was filtered from the solution and weighed The results of this series of experiments are shown in the following table:



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| 1 | 2 | 3 | 4 | 5 | 6 |
|---|---|---|---|---|---|
| Temperature, Degrees C. | AgCl, Gram | $BaSO_4$, Gram | $SO_3$, Gram | Cl, Gram | Cl (calc.), Gram |
| 533 | trace | trace | ..... | ..... | ..... |
| 597 | 0.0020 | 0.0048 | 0.0016 | 0.0005 | 0.0014 |
| 700 | 0.0136 | 0.0107 | 0.0037 | 0.0034 | 0.0032 |
| 710 | 0.0083 | 0.0087 | 0.0030 | 0.0020 | 0.0026 |
| 758 | 0.0255 | 0.0234 | 0.0080 | 0.0063 | 0.0070 |
| 769 | 0.0488 | 0.0554 | 0.0190 | 0.0137 | 0.0120 |

In the table above column No. 1 shows the temperature to which the salt was heated. No. 2 shows the amounts of silver chloride found in the absorption tube at the end of 15 min. No. 3 shows the amounts of barium sulphate obtained by the addition of barium chloride to the contents of the boat. No. 4 shows the amounts of sulphur trioxide contained in the barium sulphate precipitates. No. 5 shows the amounts of chlorine contained in the silver chloride precipitates. No. 6 shows the amounts of chlorine that correspond with the amounts of sulphur trioxide found combined with the salt, if one molecule of chlorine is liberated by the combination of one molecule of sulphur trioxide with the salt.

From the results of these experiments it appears that the rate of the reaction becomes appreciable at about 500° C. and increases rapidly with increasing temperature. The agreement between the amounts of chlorine found and those calculated from the amounts of sulphate found, while not close, is reasonably good considering the difficulties of manipulation and the small amounts of material worked with. The fact that there is no consistent variation between these quantities tends to indicate that each molecule of sulphur trioxide that combines with the salt liberates one molecule of chlorine. However, a more detailed study of the reaction would be necessary in order to fully establish this fact. From these results it also appears that the reaction is a comparatively slow one even at temperatures near the melting point of the salt; but since this apparent slowness could be attributed to a failure of the gases to come into intimate contact with the salt, it was decided to determine the effect of a more thorough contact upon the rate of chlorine evolution. In the series of experiments just described the highest temperature attained was 769° C.; the effect of higher temperature was not investigated, because it was not desired to melt the salt. At temperatures below 500° C. the indicators always remained clear, and no tests for sulphate could be obtained from the salt in the boat after the heating.

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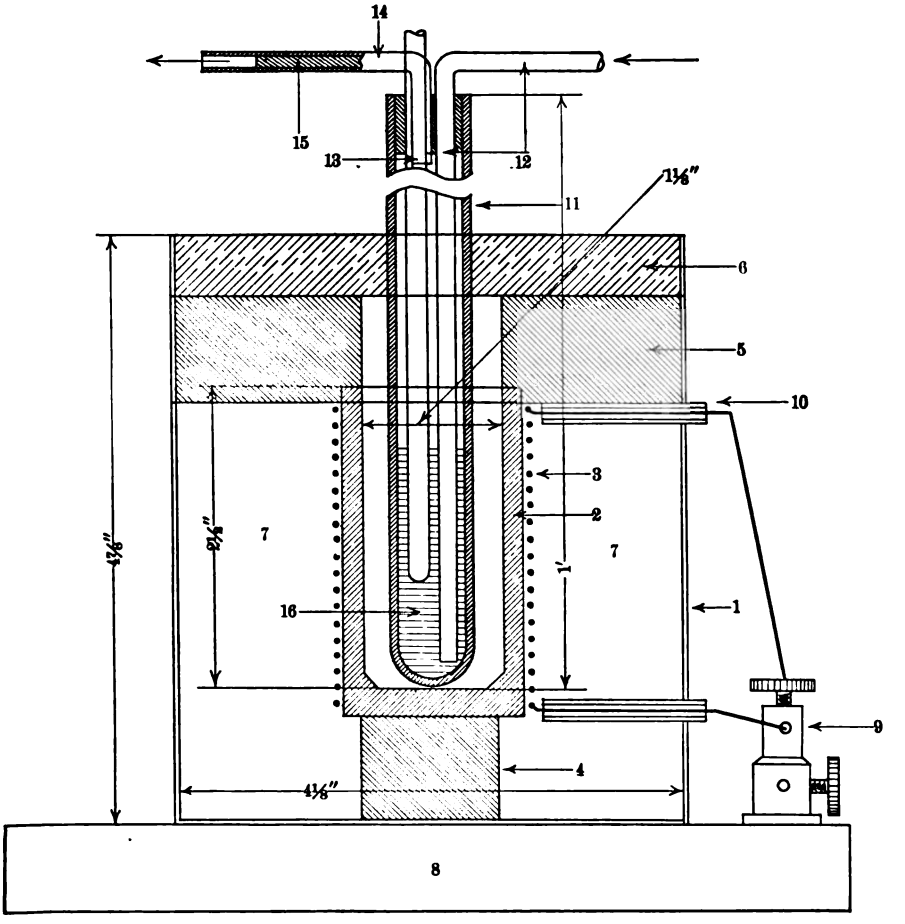


1. Shell of furnace consisting of quart tin can.
2. Fire clay cylinder with spiral thread cut on outside.
3. Resistance wire, German silver, wound in spiral on 2.
4. Block of asbestos board to support heating cylinder.
5, 6. Asbestos board.
7. Magnesia packing.
8. Asbestos board base.
9. Binding post for electrical connection.
10. Clay pipe-stem insulator for resistance wire.
11. Silica tube sealed at lower end.
12. Silica tube for delivering gases to bottom of 11.
13. Pyrometer tube of silica.
14. Glass tube for delivering gases from 11 to absorption tube.
15. Plug of glass wool in 14 to prevent salt from being carried mechanically into absorption tube.
16. Position of sodium chloride in silica tube 11.

FIG. 4.—SMALLER ELECTRIC FURNACE.

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The effect of a more intimate contact between the sulphur trioxide-air mixture and the salt was studied by constructing a small furnace and substituting it in the place of the larger tube furnace in the train of apparatus shown in Fig. 2. In the small furnace the salt was contained in the closed end of a silica tube, and through the column of salt passed the silica pryometer tube and a silica tube for the delivery of the gas mixture. The details of the arrangement are shown in Fig. 4. It is apparent that in this furnace the gas mixture was brought into much more thorough contact with the salt than in the one previously used. With 5 g. of finely pulverized sodium chloride in the tube the sulphur trioxide and air mixture was passed for 15 min., while the furnace temperature was held at 593° C. During this time 0.1248 g. of silver chloride was precipitated in the absorption tube, corresponding to 0.0308 g. of chlorine. A comparison of the amount of chlorine obtained in this experiment with the amount obtained from the tube furnace in the same time and at practically the same temperature (see the preceding table) shows, as might be expected, that better contact between the reacting substances greatly increases the rate of chlorine evolution.

Since the rate of chlorine production in the small furnace was considerably greater than in the tube furnace, further tests were made to determine the minimum temperature at which chlorine will be produced by the action of sulphur trioxide and air upon sodium chloride. About 5 g. of the pulverized salt was placed in the furnace tube, and, with the gas mixture passing, the temperature of the furnace was gradually raised. The indicator became turbid at a furnace temperature of 505° C. The furnace was then allowed to cool, fresh indicator was provided, and the heating was repeated. In three successive tests of this kind the first turbidity was noticed in the indicator when the furnace temperature was between 500° and 505°. It is therefore concluded that chlorine is first evolved at a sufficient rate to be detected by the methods employed at a temperature of 500° C. It seems quite possible that by mixing the salt with other materials, as it is mixed in the roasting furnace, catalytic effects might be produced, which would increase the velocity over that which was observed with the pure material. This point was not studied.

*Reaction of Sodium Chloride, Silica, and Oxygen.*—The second reaction studied was that involving the combination of sodium chloride, silica, and oxygen to form sodium silicate and chlorine according to the equation, $4\ NaCl + 2\ SiO_2 + O_2 = 2\ Na_2SiO_3 + 2\ Cl_2$. It has been supposed that this reaction takes place in the chloridizing roast, but experiments performed as described below failed to show any such reaction even up to the temperature at which the salt melts. The tests were made with a mixture of pure sodium chloride and pure silica in the proportion of 1.71 g. of the former to 1 g. of the latter. These

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proportions would yield a silicate of the formula $Na_2SiO_3$. The thoroughly ground mixture was placed in the silica tube of the small furnace, and the pyrometer tube and the tube for the delivery of air were arranged as shown in Fig. 4. Starting with the furnace cold and heating it slowly, air from the drying train was drawn through the salt and silica mixture and then through the absorption bulb. Up to the temperature at which the salt melted, 800° C., the indicator remained perfectly clear. Thinking that the presence of water vapor would aid in the decomposition of the salt with the formation of hydrochloric acid, a second similar test was made in which the air was passed through a wash bottle containing distilled water before it was admitted to the furnace. The result was the same as before; up to the temperature at which the salt melted no chlorine or hydrochloric acid was detected in the indicator. Therefore it is concluded that at temperatures below its melting point salt is not decomposed to an appreciable degree by the combined action of silica and atmospheric oxygen, either in the absence of water vapor or in its presence.

### The Chloridation of Copper Oxide and Nickel Oxide

The study of the chloridation of the oxides of nickel and copper was undertaken in order to determine the conditions most favorable for the chloridation of each, and thus to demonstrate whether or not a separation of these metals can be effected by the use of the chloridizing roast. In particular would such a process be useful in treating heavy pyritic ores containing copper and nickel. Such ores occur at Sudbury, Ontario, and are there treated by heap roasting followed by partial pyritic smelting for copper-nickel matte. The matte thus produced is concentrated in a basic converter to a matte containing 80 per cent. of combined nickel and copper with less than 1 per cent. of iron. The converter slag is resmelted in reverberatory furnaces. By this method of treatment all of the iron of the ore enters the waste slags of the furnaces and is therefore lost. If the copper can be removed from such ores and the iron and nickel left in a residue of such character as to be suited to blast-furnace smelting for nickel-bearing iron, the iron would be saved and the value of the ore would be correspondingly increased. In addition to saving the iron, there is of course the possibility of utilizing the sulphur for the manufacture of sulphuric acid, where such utilization is warranted by local conditions and markets.

The chemical reactions involved in the chloridation of copper in burnt pyrite have been investigated by E. Kothny,[2] who concludes that reactions represented by the following series of equations are largely responsible for the chloridation of the copper.

[2] *Berg und Hüttenmännisches Jahrbuch*, vol. lviii, p. 97 (1910). *Metallurgie*, vol. viii, No. 13, p. 389 (July 8, 1911).

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$$2\ FeS_2 + 7\ O_2 = Fe_2(SO_4)_3 + SO_2.$$
$$Fe_2(SO_4)_3 + 6\ NaCl = 3\ Na_2SO_4 + 2\ FeCl_3.$$
$$3\ CuO + 2\ FeCl_3 = 3\ CuCl_2 + Fe_2O_3.$$

Further points shown by Kothny are, that the reaction $4\ FeCl_3 + 3\ O_2 = 2\ Fe_2O_3 + 6\ Cl_2$ proceeds rapidly from left to right when a current of dry air is passed over the heated ferric chloride, while the similar reaction between cupric chloride and oxygen, $2\ CuCl_2 + O_2 = 2\ CuO + 2\ Cl_2$, is comparatively slow under the same conditions; also that the dissociation of cupric chloride into cuprous chloride and chlorine, $2\ CuCl_2 = Cu_2Cl_2 + Cl_2$, does not occur at temperatures of 340° to 550° C. when the cupric chloride is contained in a mixture in which an excess of sodium chloride is present. It is important that the two latter reactions be prevented from progressing in the left to right direction, because by so doing they tend to defeat the object of the roast. Since both are doubtless reversible, the presence of chlorine in the furnace gas and in the interstices of the roasting ore will tend to send these reactions in the right to left direction. Therefore it seems advisable to maintain as high a concentration of chlorine in the furnace gas as is possible without an undue consumption of salt.

In the experiments performed by the writer mixtures of copper oxide and nickel oxide with ferric oxide, silica, ferric sulphate, and sodium chloride were subjected to heatings at various temperatures and for different times in an electric tube furnace, a current of dry air being passed over the mixture during the heating. The compositions of the mixtures are shown in the following table:

| Constituents | Mixture No. 1 | | Mixture No. 2 | | Mixture No. 3 | |
|---|---|---|---|---|---|---|
| | Grams | Per Cent. | Grams | Per Cent. | Grams | Per Cent. |
| Cupric oxide | 0.2503 | 5.00 Cu | ..... | ..... | 0.2503 | 5.00 Cu |
| Nickel oxide | ..... | ..... | 0.2551 | 5.00 Ni | 0.2551 | 5.00 Ni |
| Ferric oxide | 1.778 | 44.5 | 1.639 | 41.0 | 1.523 | 38.1 |
| Silica | 0.400 | 10.0 | 0.400 | 10.0 | 0.400 | 10.0 |
| Ferric sulphate | 0.838 | 21.0 | 0.910 | 22.8 | 0.838 | 21.0 |
| Sodium chloride | 0.734 | 18.3 | 0.796 | 19.9 | 0.734 | 18.4 |
| Total weights | 4.0003 | | 4.0001 | | 4.0004 | |

In composition these mixtures are intended to represent dead-roasted sulphide ores containing copper or nickel or both, to which have been added ferric sulphate and sodium chloride. The percentages

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of copper and nickel in the mixtures are somewhat higher than in the ores that these mixtures are intended to represent. However, the quantity of mixture that could be used in an experiment was limited by the size of the boat and of the furnace to 4 g. Since it was not considered advisable to work with less than 0.2 g. of copper or nickel, the mixtures were made to contain 5 per cent., as is shown in the table. In order that no uncertainty should arise in regard to the amount of copper or nickel present in each experiment, the exact amounts of the ingredients of the different mixtures were weighed out and material for each experiment was mixed separately in an agate mortar. In practical applications of the chloridizing roast it is customary to produce the ferric sulphate, necessary for the decomposition of the salt, in the furnace by the oxidation of pyrite. In laboratory work with the small amounts of material used in these experiments it is more convenient to add prepared ferric sulphate, and by its use to avoid any irregularities that may arise from the roasting of the pyrite.

Mixture No. 1 contains double the amounts of ferric sulphate and sodium chloride necessary to convert all of the copper into cupric chloride. Mixture No. 2 contains double the amounts of these ingredients necessary to convert all of the nickel into nickel chloride. Mixture No. 3 contains enough ferric sulphate and sodium chloride to chloridize both the nickel and the copper.

*Preparation of the Materials.*—Of the materials used in the mixtures the ferric oxide, silica, and sodium chloride required no special preparation. The ferric oxide was that prepared by Kahlbaum, and showed no traces of copper or nickel when tested qualitatively. The silica was prepared by grinding clear white quartz in a porcelain jar mill. The sodium chloride was Merck's C. P., and was prepared by heating to about 600° C. and then pulverizing in an agate mortar.

The nickel and copper oxides were prepared by calcining their respective nitrates. Both nitrates were dehydrated over a gas flame and then heated in a muffle until decomposition was complete. The oxides thus produced were boiled with water to remove soluble material, and, after filtering and washing, the oxides were again heated in the muffle. The averages of three electrolytic determinations on each of the materials are given below and compared with the theoretical composition of the pure compounds.

| | Per Cent. |
|---|---|
| Copper oxide prepared | 79.88 Cu |
| CuO pure | 79.89 Cu |
| Nickel oxide prepared | 78.41 Ni |
| NiO pure | 78.58 Ni |

Pure anhydrous ferric sulphate was prepared from the C. P. salt of Baker & Adamson. The raw material contained, as undesirable impurities, sulphuric acid, nitric acid, and water. A part of the vol-

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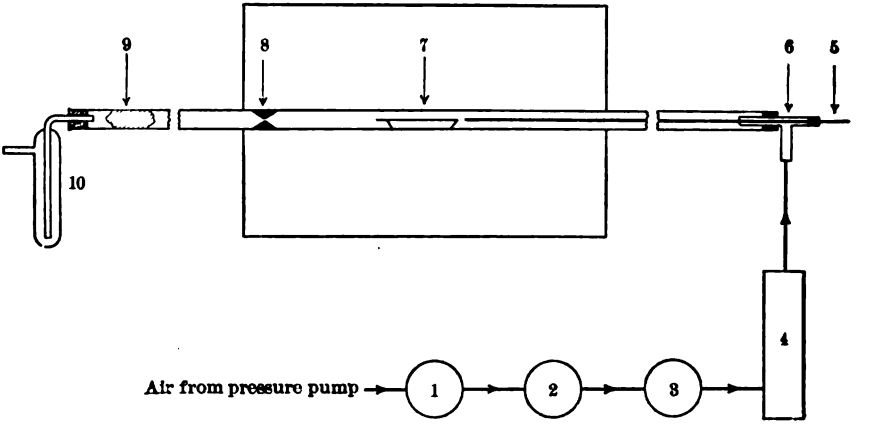
atile impurities was removed by heating over a gas flame. Then in order to free the salt from acid and water it was heated to 500° C. for a few minutes. The heating was done in a closed tube of hard glass, the closed end of the tube containing the salt being heated in the small furnace shown in Fig. 4, while the upper and open end of the tube was heated with a gas flame to prevent the condensation of the impurities. About 20 g. of the anhydrous ferric sulphate was prepared in this manner; it was in the form of a fine cream-colored powder. The analysis is shown




1. Wash bottle containing potassium permanganate.
2. Wash bottle containing potassium hydroxide.
3. Wash bottle containing sulphuric acid.
4. Drying tower containing phosphorous pentoxide.
5. Silica tube containing pyrometer wires.
6. Glass T through which pyrometer tube and air pass into the furnace tube.
7. Porcelain boat containing mixture to be roasted.
8. Point in the tube where most of the volatilized material condensed.
9. Plug of glass wool to provide condensation surface for volatilized material.
10. Railroad tube containing water.

The furnace and porcelain tube are the same as those shown in Fig. 1.

FIG. 5.—DIAGRAM OF APPARATUS USED IN ROASTING MIXTURES.

below compared with the theoretical composition of the pure compound; the average of two analyses is given.

Prepared ferric sulphate: $Fe_2O_3$ = 40.32, $SO_3$ = 59.68 per cent.
$Fe_2(SO_4)_3$ pure: $Fe_2O_3$ = 39.93, $SO_3$ = 60.07 per cent.

From the comparison it is evident that only an inconsiderable amount of the sulphate was decomposed during the process of drying.

*Method of Conducting the Experiments.*—The furnace shown in Fig. 1 was connected with other apparatus as shown in Fig. 5. The furnace was heated to the desired temperature with the current of dry air passing through it, then the boat with its contents was placed in the position shown. A plug of glass wool was inserted into the end of

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the tube for the purpose of providing a large surface upon which volatilized copper chloride might condense. The gases from the tube were passed through a railroad tube containing water in order to catch any copper salt that did not condense in the tube or on the glass wool.

During the first 10 min. of heating, chlorine and sulphur trioxide were evolved freely. After this length of time the rate of evolution of these gases usually diminished perceptibly, and continued to diminish until at the end of 4 hr. the air coming from the tube contained only small amounts of them. The time of heating was in most cases 4 hr.; in a few it was only 2 hr.

The products from an experiment performed as described were the volatilized material that had been carried by the air current to the cooler part of the furnace tube, where it condensed, and the residue remaining in the boat. The latter was in the form of a dark-brown porous cake, which could be removed from the boat in a single piece, but which was soft and could be easily pulverized. Most of the material volatilized from the boat condensed in a somewhat hard ring at the point (8) shown in Fig. 5; a small amount of it condensed on the glass-wool plug; in no case was a trace of copper or nickel found in the railroad tube. The removal of the ring of condensed material from the wall of the tube was at times quite difficult, and particularly so during the last series of experiments with mixture No. 3. In all cases it was necessary to place a rubber stopper in one end of the tube, then to introduce about 25 cc. of glass beads together with the solvent, and then, after inserting a stopper in the other end, to shake the tube and contents vigorously. The beads aided in removing the deposit from the wall of the tube and thus made the action of the solvent more rapid. One or two such treatments with ammonia followed by one with nitric acid were generally used. Even this treatment failed to remove all of the material after the glaze on the inner surface of the tube had been partly destroyed by the combined actions of the deposit and of the beads, so that in the later experiments all of the copper in the volatilized product was not recovered. An interesting point in regard to the volatilized material is that in no case was any appreciable amount of iron found in it. This fact indicates that if the volatile ferric chloride is formed during the roast, it is decomposed before it can pass from the roasting material into the gases.

The products from each experiment were treated as follows: In the volatilized material copper or nickel or both were determined electrolytically. The material in the boat was boiled with water, filtered, and washed, in order to remove water-soluble copper or nickel. Since this treatment would not remove cuprous chloride if present, the residue was further treated by boiling with saturated sodium chloride solution, after which it was again filtered and washed. The brine solution in all cases contained only small amounts of copper, and accordingly was

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not analyzed separately, but was added to the solution obtained from the water treatment. The copper in the combined solutions was determined electrolytically, after evaporating with sulphuric acid to white fumes, adding 3 cc. of nitric acid, and diluting to about 200 cc. with water. When nickel was present it was also determined electrolytically. In the case of the experiments with mixture No. 2, which contained no copper, the brine treatment was omitted. The residue from these operations, together with the incinerated filters that had been used, was boiled with three successive portions of concentrated nitric acid. This treatment served to remove the copper oxide without dissolving enough of the iron oxide to interfere with the subsequent electrolytic determination of the copper. No attempt was made to determine nickel in the residues. In treating the residues with water and with brine it was observed that not more than traces of iron were dissolved in any case.

*Results of the Experiments.*—The results of a number of experiments with the three mixtures are given in the following table:

| Mixture No. | | Temperature, Degrees C. | Time, Hours | Per Cent. Volatilized | | Per Cent. Soluble in $H_2O$ and Brine | | Per Cent. in Residue, Cu | Per Cent. Accounted for, Cu |
|---|---|---|---|---|---|---|---|---|---|
| | | | | Cu | Ni | Cu | Ni | | |
| 1 | Cu only.. | 500 | 4 | 24.9 | ..... | 36.2 | ..... | 29.0 | 90.1 |
| 1 | | 600 | 2 | 29.9 | ..... | 29.5 | ..... | 33.1 | 92.5 |
| 1 | | 600 | 4 | 68.1 | ..... | 7.5 | ..... | 17.2 | 92.8 |
| 1 | | 700 | 2 | 71.6 | ..... | 15.4 | ..... | 9.0 | 96.0 |
| 1 | | 700 | 4 | 92.5 | ..... | 0.9 | ..... | 1.9 | 95.3 |
| 2 | Ni only.. | 500 | 4 | ..... | trace[a] | ..... | 6.2 | ..... | ..... |
| 2 | | 550 | 4 | ..... | trace | ..... | 17.7 | ..... | ..... |
| 2 | | 600 | 4 | ..... | trace | ..... | 13.0 | ..... | ..... |
| 2 | | 650 | 4 | ..... | trace | ..... | 1.5 | ..... | ..... |
| 2 | | 700 | 4 | ..... | trace | ..... | 0.8 | ..... | ..... |
| 3 | Cu and Ni | 600 | 4 | 47.7 | trace | 21.5 | 2.3 | 12.9 | 82.1 |
| 3 | | 650 | 4 | 71.4 | 2.9 | 2.7 | 0.4 | 6.5 | 80.6 |
| 3 | | 700 | 4 | lost | 0.8 | 4.5 | 1.0 | 4.1 | ..... |
| 3 | | 700 | 4 | 67.6 | 1.6 | 8.5 | 1.2 | 3.4 | 79.5 |
| 3 | | 700 | 4 | 68.1 | 2.7 | 1.3 | trace | 5.1 | 74.5 |

[a] The amounts of nickel volatilized in the experiments with mixture No. 2 were too small to be estimated quantitatively. The volatilized portion was accordingly added to the solution containing the water-soluble nickel, and the combined solutions were analyzed for nickel. In the experiments with mixture No. 2 the brine treatment was omitted.

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In the first series of experiments the amounts of copper accounted for vary from 90.1 to 96.0 per cent., while in the third series still less of the copper was accounted for. This is due to the increasing difficulty that was encountered in removing without loss the volatile products from the tube; this point has been mentioned in a preceding paragraph.

In reviewing the results of the experiments it is necessary to bear in mind the facts that only small amounts of material were used, that the time of treatment in each case was comparatively short, and that the results cannot be directly compared with those secured when a chloridizing roast is performed on a large scale because of the differences in the conditions. It was the intention to study the behavior of copper oxide and nickel oxide when mixed with iron oxide and subjected to heating under the conditions that favor the chloridation of the metals. It is necessary to consider as chloridized that part of the copper and nickel which was volatilized, because neither the sulphates nor the oxides of these metals are volatile under the conditions existing in the experiments.

The experiments with mixture No. 1 show that the completeness of chloridation of the copper increases with increasing temperature and with increasing time. An interesting point is the fact that in the last experiment with this mixture 92.5 per cent. of the copper was recovered from the volatile product. Experiments with mixture No. 2 containing nickel oxide show the behavior of the nickel under conditions similar to those to which the copper oxide was subjected. Only traces of nickel were found in the volatile products, as is stated in the footnote to the table. The maximum production of water-soluble nickel resulted at 550° C., the nickel thus extracted amounting to 17.7 per cent. The experiments at higher temperatures showed decreasing amounts of water-soluble nickel. Roasting for 4 hr. at 700° C. produced only 0.8 per cent. of water-soluble nickel. Whether or not the nickel extracted in these experiments was in the form of chloride or sulphate was not determined; the conditions were favorable for the production of either compound.

Considering the results of these two series of experiments, it appears that nickel is not rendered water soluble nor is it volatilized to any considerable extent when the temperature of the roast is above 650° C., while such temperatures are quite favorable to the chloridation of copper. In confirmation of this conclusion we have the results of the third series of experiments, which show a maximum of 3.3 per cent. of the nickel converted into water-soluble and volatile compounds at 650° C. The increased tendency toward volatilization exhibited by the nickel in the presence of copper is probably due to the more or less mechanical influence exerted on the nickel chloride by the volatilization of the copper chloride.

These laboratory experiments of course do not prove that roasted pyritic ores containing copper and nickel can be treated commercially

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by chloridizing the copper and leaching it from the iron and nickel oxides, leaving a product suitable to be smelted to nickel-bearing pig iron. That could only be demonstrated by carefully conducted experiments on large quantities of the ore in question. However, the experiments do prove the possibility of effecting such a separation and indicate the temperatures at which the roast should be conducted in order to secure the desired results.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

# The Use of Mud-Ladened Water in Drilling Wells

BY I. N. KNAPP, ARDMORE, PA.

(New York Meeting, February, 1915)

*Introduction.*—The special object of these notes is to describe the mixing, testing, and use of mud-ladened water for rotary drilling in such a way as to make them helpful to the driller, the operator, or the engineer in solving his own special drilling problems.

The structures, apparatus and tools used are indicated in a general way. No attempt is made to describe the art of rotary drilling; only such descriptions as are necessary to make plain the use of mud are given.

The information is the result of actual experience in drilling in Coastal Plain formations. The materials encountered in the wells drilled were unconsolidated sands, gravels, and clays, in which thin layers of sandstones, shell conglomerates, and shales began to appear at about 1,200 ft. in depth, although one well was drilled to 3,018 ft. without encountering any cemented or indurated materials.

*Unusual Conditions.*—The general surface of the ground where the drilling was done being hardly a foot above mean tide, and the daily tidal variation being about 15 in., it was found necessary to plank over the sod surface with 3-in. plank to work from. A 30 by 30 ft. planked area was sufficient to carry the derrick with equipment and 2,500 ft. of 4-in. drill pipe stacked in it. The derrick was set up 3 ft. on cribbed blocking arranged to spread the load over the planking. The engine, boilers, and pipe yard were also carried on 3-in. planking.

*Drilling Outfit.*—This consisted of a derrick, 20 by 20 ft. and 84 ft. high, two 35 h.p., "oil country" boilers, one double 8 ¼ by 10 in. reversible engine, two 10 by 6 by 12 in. duplex mud pumps, one 15-in. rotary, "hoisting works" with chain drive, hoisting block with line, crown block with pulleys, rotary jetting swivels, hose, complement of tools, pipes and connections, and a mud mixer, with engine.

*Hydraulic Rotary Drilling Method.*—This method of drilling is an adaptation of Fauvelle's[1] invention made in France in 1833 for drilling

[1] See *Journal of the Franklin Institute*, 3d ser., vol. xii, pp. 369 to 372 (1846), for abridged translation of Fauvelle's own account On a New Method of Boring for Artesian Springs.

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artesian wells, and was the progenitor of the water-flush method now employed in Europe and the hydraulic rotary method of America. It is absolutely necessary to use this method to complete a well to depths of 200 ft. and over in the Coastal Plain formation. Its success depends on the proper use of mud-ladened water. Rotary drilling is an art that necessarily requires skillful work, and the best results in any district can only be had after numerous test wells have been drilled, the water sands and productive oil or gas horizons located, and proper drilling and casing setting methods worked out.

*Initial Ground Pressure.*—This in any oil or gas field is in general less than the hydraulic head of 43 lb. per 100 ft. of depth.

*Mud-Ladened Water.*—Water can easily be ladened with any clayey material that will remain in suspension for a considerable period of time. Such mixture may weigh 33 per cent. more than the same volume of clear water, or give a head of 57 lb. per 100 ft., or a margin of safety of 14 lb. on each 100 ft. of depth. Heavier or lighter mixtures can be used, but that mentioned was found to give good average results.

The time of settling of clayey material in still water varies greatly. One mixture will settle clear in a week, another in six months.

Laboratory experiments show that matter in suspension imparts a certain viscosity to fluids and the rate of settling is by no means dependent solely upon the difference in specific gravities, or upon the fineness of subdivision of the solid.[2]

*Requirements.*—It is fundamental in the hydraulic rotary method of drilling that a circulation of mud-ladened water, such as described, be maintained from the surface down through a vertical rotary drill pipe and bit and up between the drill pipe and the wall of the well back to the surface. Moreover, at all times, the mud pressure at any point in the well must exceed the pressure in the strata penetrated at that point, thus excluding all gas, oil, and water as they are encountered. The proper use of mud-ladened water gives a means of drilling wells and keeping them in a state of equilibrium until screens can be set in productive horizons and casing set and cemented in. Thus complete control is given in bringing them to producing. As a practical matter, in a drill hole 1,500 ft. deep a good mud will not settle in a week sufficiently to prevent a drill pipe from being run freely to bottom, and the mud pump from starting circulation, so rotating and drilling may be resumed. Also, such mud will not stick or freeze in a drill pipe if left in 24 hr.

It was necessary to use a power-driven mixer to supply, and a mud pit

[2] *American Chemical Journal*, vol. xlvi, p. 278 (1911) on The Viscosity and Fluidity of Suspensions of Finely Divided Solids in Liquids. Also, U. S. Geological Survey *Professional Paper No.* 86, Transportation of Débris by Running Water, p. 228 (1914).

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to store, sufficient mud properly to control a well drilled in the formations named. Such a storage pit is formed on the surface by building a dike about 3 ft. high around an area 20 by 30 ft. from which the sod has been taken. It must be so located that the thick mud maintained in it can be quickly drawn out into the pump pit (hereafter described) in quantity as desired to thicken the circulating mixture. Mud mixers of suitable design can be purchased of the well-supply houses.

*Mud Tests.*—It is very difficult to judge whether any particular clayey material will make a suitable mud. The only sure way to know is to mix a sample of available material and try it.

Mud can be best tested by weighing. I used a common market beam scale (without pan) having a capacity to weigh 50 lb. by ounces, and a common galvanized iron bucket in which to weigh the mud. Four equally spaced ¼-in. holes were punched und erthe top rim to furnish level marks to fill to.

The bucket is first hooked on under the beam, the poise brought to zero, and the whole balanced by counter-weighting. The bucket is then filled to the level holes with clear water and its weight taken and noted. It may be, for instance, 18 lb. The poise is again placed at zero and more counter-weight added to balance the clear water. This water is then thrown out and the bucket filled with mud, and the poise, when balanced on the beam, will indicate the additional weight of the clayey material in suspension over that of the same volume of clear water. The bucket may be filled with mud as many times as desired and weighed without further adjustment. If the additional weight found at any time should be 6 lb. it would indicate the mixture weighed about 33 per cent. more than the same volume of clear water. If a good working mixture is found to be 5 lb. of additional weight per bucket of mud, and in drilling it increases to show 6 lb., clear water may be slowly added until the weight drops to 5 lb. or a little less. This weighing is simple and practical and is no fad. It is just as necessary for the driller to have a scale and know the weight of the mud he is using as to have a gauge to show the mud pump pressure. A well can be drilled without a pump gauge, or scales to weigh the mud, but it can be drilled safer and in better time if both measurements are intelligently used.

*Other Tests.*—A water circulation carrying a considerable percentage of sand in suspension is not a proper mud for rotary drilling, although it may meet the weight test.

If for any reason a circulation of this kind is stopped, the drill pipe is likely to "freeze in" in a short time, depending on the rapidity with which the sand may settle. The amount of sand in suspension may be approximately estimated by taking one-third of a bucket of such mud mixture and adding one-half of a bucket of clear water, stirring thor-

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oughly, and letting stand half an hour. If less than an inch of clear sand then shows in the bottom of the bucket the circulation is reasonably safe to use. If it is much more it may not be safe. This is a matter that only experience and good judgment can determine.

The percentage of sand in any mixture can be accurately determined with a hand centrifuge with two-arm sedimentation attachment and graduated glass tubes. A proper mud shows no sign of separation—except perhaps 1 or 2 per cent. of sand—if put into such a machine and

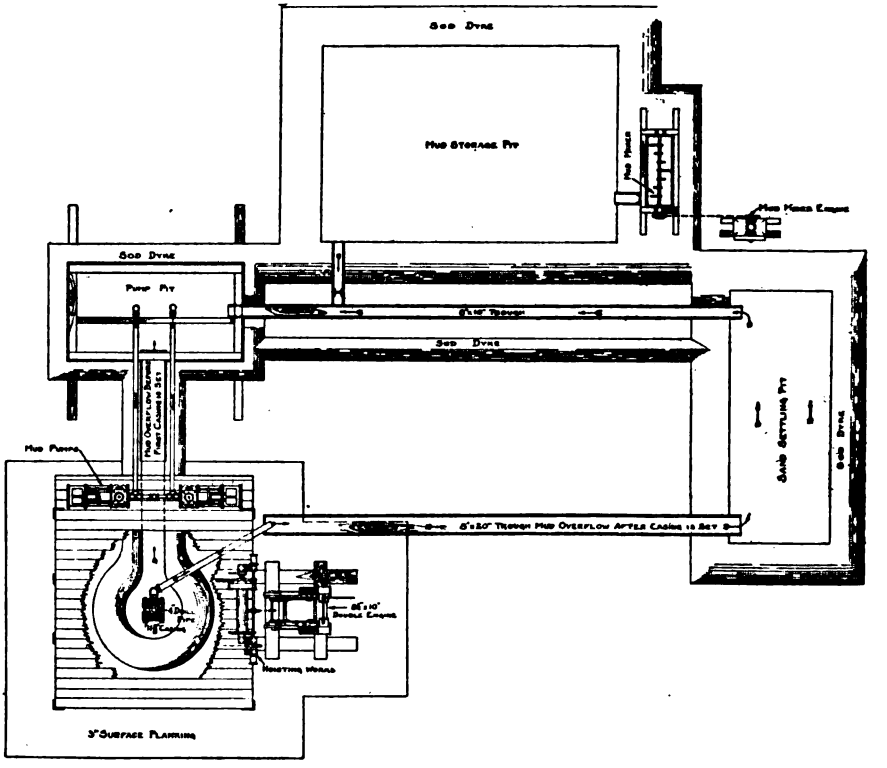


FIG. 1.—PLAN OF WELL-DRILLING PLANT SHOWING CIRCULATION OF MUD.

run at 1,200 rev. per minute for 5 min. This shows that the precipitation of suspended clayey matter in water is extremely slow and that sand settles very much quicker. By mixing mud half and half with saturated brine a quick separation takes place. Turning the machine for 2 or 3 min. at the speed mentioned produces a complete separation of all suspended matter. The precipitation shows itself in layers in the graduated glass tube, the sand coming down first.

*Pump Pit.*—This pit is located on the pump side of the derrick, with is longest center line parallel to and about 20 ft. from the derrick base.

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(See Figs. 1 and 2.) A convenient size is 8 by 16 ft. (inside timbers) by 5 ft. deep, and lagged with 2-in. plank 7 ft. long, placed vertically on the outside of a square set of 8 by 10 in. timbers.

The 2 ft. of lagging above the ground is backed with sod to prevent surface water from running into the pit. A sod dike about 6 ft. in diameter, with the well location as a center, and 20 in. high, is built on the 3-in. plank. This is connected directly to the pump pit by a ditch

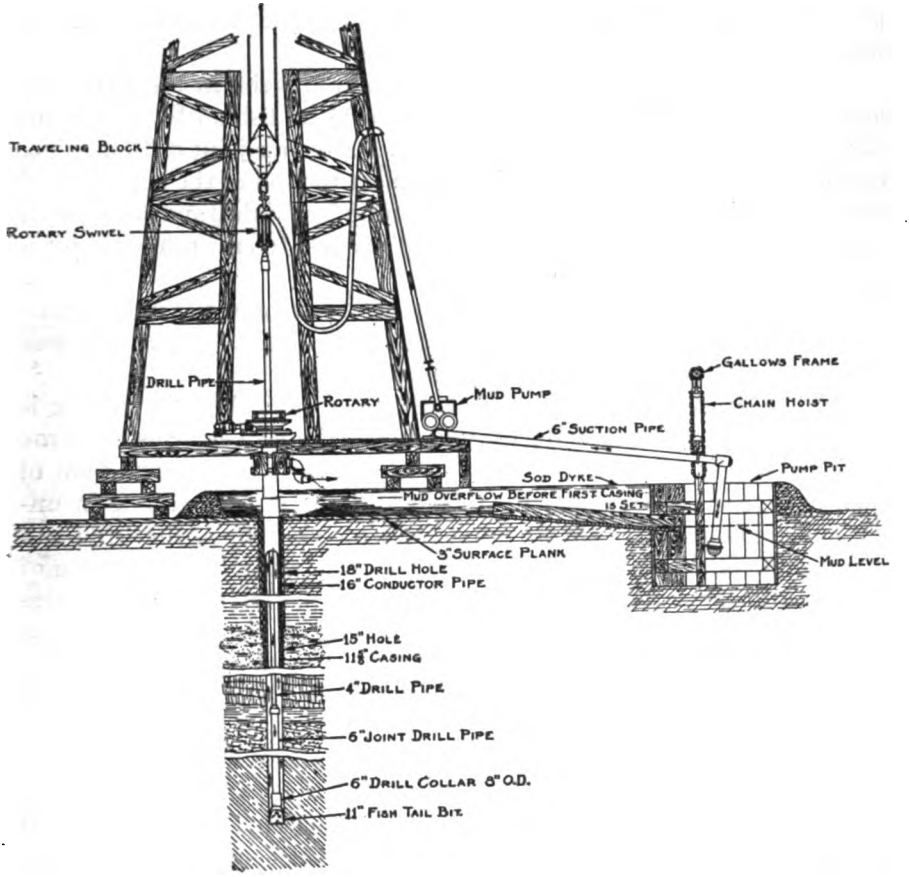


FIG. 2.—ELEVATION OF WELL-DRILLING PLANT SHOWING CIRCULATION OF MUD.

banked up 20 in. by 2 ft. wide. The dike protection was absolutely necessary, as the tide rose at times as much as a foot over the surface.

*Pump Strainer.*—The large amount of roots and trash in the clay available for mud mixing, and also the fact that wood and swamp materials were drilled through at various depths, made it necessary to arrange for frequent cleaning of the pump strainer. The best arrangement found for conveniently raising and lowering the pump suction pipe for this purpose



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was a 1-ton chain hoist hung over the pit from a gallows frame, as indicated in Fig. 2. With it one man could handle a pump suction with ease while removing floating trash from the strainer with his hands.

*Preparations for Drilling.*—The drilling apparatus being all ready, a mud pump is started and the thin water pumped out of the pump pit. The latter is then filled with mud to above the ground or tidewater level, so there can be no chance of the inflow of clear water thinning the mud. A circulation is then established, by operating a mud pump, from the mud pit through the drill pipe and bit, back to the pit, and continued until the whole content of the pit is properly tempered.

*Drilling.*—The rotary is started and the drill begins at the surface to make hole. To a depth of 15 ft. I used an 18-in. fishtail bit with 6-in. shank and 6 by 10 in. drill collar (12 in. O. D.) on one joint of 10-in. pipe, followed by a string of 6-in. drill pipe. Near the top of the hole slabs of mud as large as a man's two hands were cut up by the drill and floated out, and had to be shoveled away. Also mud was lost to crawfish holes for a time before they became mudded off.

A 16-ft. length of galvanized, corrugated, culvert pipe was used for the conductor pipe and driven into place. (See Fig. 2.) The top was belled out and spiked to the 3-in. surface planking.

The 18-in. fishtail bit being replaced by a 15-in. bit, the drilling is continued through the conductor pipe and the mud kept close to some standard weight. The large drill collar (12 in. O. D.) on a 10-in. joint of pipe in a 15-in. hole retards to some extent the rate of drilling in unconsolidated material. On the other hand, its use compels the drilling of a round straight hole that requires no reaming before setting a string of casing; also, its use seems more effectively to plaster the drill cuttings and mud into the walls of the well in puddling off the porous strata as encountered, and so preventing loss of mud. In pulling out, there is no dragging or turning of the drill pipe, and in running in, the bit goes plumb to bottom.

As a matter of experience, it had been found that a well drilling in soft sandy material with a 15-in. fishtail bit, followed by a 6-in. (about 7 ½ in. O. D.) drill collar and 6-in. pipe, was easily deflected by concretions or pieces of wood, resulting in crooks or humps in the hole. In extreme instances, after pulling out and starting in with a sharp bit such crooks or humps would form enough obstruction so that the bit would not spud past them, and upon rotating a new hole would form, side-tracking a considerable depth already drilled, which was consequently lost. Furthermore, it was often necessary in running back into crooked holes to start pumping and rotating before getting to bottom, because materials which were scraped off the walls of the well and carried down made a bridge or stoppage. In any case a crooked, humped or deflected hole requires the

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use of a rotary shoe or reamer to size the hole down before a string of casing can be run in and set.

Just before pulling out the drill pipe, the mud in the pump pit and the surface circulation should be stirred up, so as to weight the descending mud in order that the drill string will uncouple dry. In pulling out, mud must be frequently pumped into the well so that the "head" of mud, or pressure from within the well, will meet the fundamental requirement before stated. After running a drill pipe to bottom or to a bridge in the hole, no attempt to start a circulation for drilling should be made until the surface mud is first circulated and brought to a proper temper.

In case of a heavy rain it is best to stop drilling and pumping, and pull up off bottom before the circulation thins out. With a good mud there is no danger of "freezing in" even if the drill pipe is left in over night. A circulation should be maintained only when rotating. Pump pits and mud circulation must always be protected against surface drainage and rise of tide.

*Action of Water and Mud.*—Clear or turbid water flowing on the surface will erode unconsolidated materials, the rate depending upon the velocity or grade and upon the volume. It has also a solvent or slacking action on earthy matter. If such flow was, for instance, through a shallow ditch it might perhaps cause the banks to cave sufficient to make a meander outside of the dug channel. If, however, clayey matter is added to the flowing water the erosive action weakens until a point is reached where erosion practically ceases and the suspended matter will begin to build up and protect the loose material. The solvent action also will be very much weakened and in such a case a shallow ditch in loose material will tend to keep its shape as dug. Clear water will continue to sink in sand or porous material in quantity for an indefinite time. Turbid water will gradually render sand or porous material impervious, but the suspended matter will be carried in and deposited in the sand to a considerable depth. A good mud (or mud-ladened water) will render sand or porous materials impervious almost instantly and the penetration of the mud is small. These are all matters of common observation.

If clear or turbid water is circulated in rotary drilling such water will erode the walls of the well, particularly in loose sands and sandy materials, and cause them to cave and to lose their cylindrical form.

Sticky clays and such unconsolidated strata as can resist the erosion of circulating clear water will drill much faster with it than with mud, on account of its solvent action. It is a matter of necessity, however, to start with and keep a mud circulation of a standard that will resist erosion of the weakest sand strata, if the well is to be kept in proper cylindrical condition for successful cementation and completion. If the walls of a well in sand or other material cave, a large amount of such

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material will come out with the overflow. This caving increases enormously the area to be mudded off.

*Gain or Loss in Mud.*—In rotary drilling there is always a slow increase in the weight of the circulating mud. When the mud gets too heavy a part of it may be pumped back into the storage pit and clear water slowly added to the circulation until the original volume and standard weight are restored.

In drilling a 15-in. hole through sand beds the amount of mud lost should be observed. For instance, in my drilling through 60 ft. of coarse reddish water sand the pump pit would have mud from storage drawn into it several times, equal in all to about 4 ft. in depth by 9½ by 17½ ft. area, or 673 cu. ft. If this mud is one-third to one-half material in suspension, there would be 226 to 337 cu. ft. of material to lodge in the pore space in the sand. The porosity of sand ranges in general from 26 to 47 per cent., so it is fair to assume 38 per cent. as an average. If we assume that the mud may penetrate the walls of the 15-in. well 2 ft., we have a column of sand 5 ft. 3 in. in diameter, less the 15-in. core, and 60 ft. long, or 1,225 cu. ft., having a porosity of 456 cu. ft. to hold suspended material. When the mud was properly mixed and skillfully used, the loss to the porous strata was found to be surprisingly small, and, on the other hand, the amount of cuttings brought out was also small and did not appear to exceed the cubical contents of the hole which had been made.

In my first attempts at drilling in this formation too much clear water was sometimes used, resulting in caving from several sandy horizons at the same time. At such times as many as four men were busy shoveling away the sand that was being brought out in the overflow, and in such cases it required the use of much mud to stop the caving. At these times there was also danger of "freezing in" the drill pipe.

*The Sampling of Drill Cuttings.*—It is extremely difficult in rotary drilling to get reliable samples of the material passed through. If clear or turbid water is used, the cuttings tend to dissolve or separate by the jigging action of the ascending circulation, and the heavy particles lodge in the caves and erosions in the walls of the well and mix with the caved materials. It is very difficult in such cases to find any material in the overflow from which to judge the strata through which the drill has just passed. With a heavy mud there is much less dissolving, mixing and separating of the drill cuttings, and, on washing out a sample of mud, particles may be found which fairly indicate the material just drilled through. I found that a No. 18 mesh bronze wire mosquito screen set in a wood frame about 16 by 20 in. made a good screen on which to separate the shells and coarse particles from the mud by washing. Gas sands can be recognized by their porosity, as revealed by examination under a magnifying glass of 20 diameters. Oil sands have the same kind of porosity and in addition can be recognized by their taste.

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After setting and cementing[3] in a casing and using a regular mud, in drilling out a cementing plug it was observed that wood cuttings appeared in about 25 min. and brass, from a back-pressure valve, in about 40 min., when the calculated flow would have taken 30 min., showing the acceleration by floating of the wood and retardation by sinking of the brass.

*Open Hole.*—With a good mud and skillful working there was no difficulty in drilling 700 ft. of 15-in. open hole before setting the first string of casing (11⅝ in.). The drill passed through several layers of fine water sand, having a total thickness of perhaps 200 ft.; through 100 ft. of sandy materials and shells, and 400 ft. of clays and sticky mud. The short ditch, about 25 ft. long and the pump pit, previously described, were of sufficient settling capacity, because the proper use of mud did not permit the sands to run. Starting with the storage pit full of thick mud, it was not found necessary to mix any additional mud to reach a depth of 700 ft., under the conditions named. As a matter of experience, I drilled 1,880 ft. in depth of 8½-in. hole with the rotary, using mud, going through three well-known water-bearing loose sands with strata of clays, shales, shell beds and some sheets of rock, and then set a string of 6⅝-in. casing, 60 ft. of 14-in. O. D. drive pipe being all the other pipe in the hole. There was no caving and but little loss of mud. The work was done in day time only and the drill pipe was many times left in over night with no sign of sticking or freezing.

*First String of Casing.*—This should set so that the top coupling will come just above the 3-in. surface plank, and so that a drilling nipple can be screwed into it. This nipple should have a 6-in. hole cut in its side, so as to bring the overflow from the well as high up as possible under the derrick floor. A wood block is fitted to match over the side hole and a 6-in. pipe flange bolted to the latter and the whole clamped to the nipple. A 6-in. pipe is connected (with nipples and two elbows to make a swing) to the flange to carry the mud circulation of the derrick to an overflow just back of the place where the driller stands at the hoisting-drum brake. The advantages in bringing the overflow up to this point are that it can easily be seen by the driller and that there is a better chance for getting samples of cuttings than from a ditch. Flash boards may be placed across the trough to act as riffles to catch the sand. A convenient size for the trough is 8 by 20 in. inside and 48 ft. long. This is blocked up as high as possible and still permit it to carry the mud from the pipe overflow into the end of a sand settling pit about 45 ft. from the derrick. This pit is made by banking sod 30 in. high around a plot about 10 by 30 ft. From its opposite end an 8 by 10 in. trough (made of 2 by 10 by 16 plank) carries

[3] I. N. Knapp: Cementing Oil and Gas Wells, *Bulletin* No. 87, pp. 471 to 488 (Mar., 1914).

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the mud back to the pump pit. This 8 by 10 in. trough is sunk into the ground toward its discharge end and so cuts through the dike around the pump pit and therefore requires a dike protection. This arrangement prevents surface, tide or ground water from getting into and thinning the mud and was absolutely necessary under the conditions in which the work was done. The wear and tear on the mud pumps is much reduced by thoroughly separating the sand from the mud.

An 11-in. fishtail bit with 6-in. shank and drill collar (about 8 in. O. D.) on one joint of 6-in. pipe and a string of 4-in. drill pipe were used to drill through the 11 ⅝-in. casing. The same condition of tempering the mud in the surface circulation and in the pump pit before starting to drill should be observed.

*Experiences in Drilling.*—After the casing was set in one hole at about 700 ft. in clay, drilling was resumed with clear water. The casing shoe did not hold against the difference of pressure between the clear water inside and the thick mud outside of the casing. The heavy mud outside slid down, followed by heaving sand. This made it necessary to pull out the drill pipe to prevent freezing and thereafter to use mud. On running in again, sand was found bridging the hole and it was necessary to rotate and wash out the lower 200 ft. of casing, using a proper mud. The lower end of the casing was thrown out of line so much by caving that the drill-pipe couplings would sometimes catch under the casing shoe. Such experience shows the need of cementing in each string of casing set in unconsolidated formations and the necessity of keeping up at all times a mud of proper temper which is suitable for the work in hand. I found no difficulty in keeping 1,200 ft. of hole open below the first string of casing, with mud, and controlling all water and gas pressure where there was a proven gas pressure of 650 lb. at about 1,600 ft. in depth. I do not know the limit in depth of open hole that can safely be maintained in unconsolidated material when mud is skillfully used. On the other hand, as I have shown, one can get in serious trouble by caving, in drilling only a short distance (less than 100 ft.) below a string of casing when clear water is circulated. It has also been my experience that the circulation of clear water in a well drilled with cable tools in hard ground may bring about serious trouble through erosion and caving. In unconsolidated material the circulation of clear water would mean the probable loss of the well.

I must, therefore, take issue with geologists and petroleum engineers, who recommend washing out wells with clear water before cementing—not only on account of erosion and caving, but on account of the danger of gas blow-outs, and the entrance of oil and ground waters having a deleterious action on the setting of the cement.

I have drilled to 2,007 ft. with the mud circulation showing a temperature of 72° F. The well on being bailed established a flow of 450 gal. of water per minute with a head of 80 ft. and a temperature of 99 ½° F.

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This shows that by circulating mud, hot water under 34 lb. surface pressure can be kept back, giving ample time to cement in a casing at ordinary temperature.

*Invention of the Use of Mud.*—The mixing of clay with circulating water for sealing porous strata in drilling wells is mentioned on p. 167 of *Bulletin No.* 212, *U. S. Geological Survey* (issued in 1903). This is the earliest written mention I find of the use of mud in any kind of drilling.

It was no doubt noted early in drilling by the hydraulic rotary method that after going through a sand or porous stratum and then entering into clay, such porous material was rendered impervious by the clay cuttings; also that caves in sand were built up and plastered over by such cuttings.

The use of mixed mud appears to have had a gradual introduction during the past 20 years and is not the invention of any one in particular. The testing of mud by weighing as set forth in this paper is, I believe, new.

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# Steep Pitch Mining of Thick Coal Veins

BY W. G. WHILDIN, LANSFORD, PA.

(Pittsburg Meeting, October, 1914)

THIS paper will be confined to a discussion of the methods in use in the property of the Lehigh Coal & Navigation Co. in the Panther Creek valley. Only the methods used in mining the Mammoth vein will be considered, as the methods in use in the smaller veins are adaptations of the same.

Typical cross-sections of the coal basin are shown in Figs. 1, 2, and 3.

The Mammoth vein varies in thickness. It is 21 ft. at one colliery, 50 to 125 ft. at another, and over 200 ft. at still another. Its normal thickness is 35 to 40 ft.

At some points in the Panther Creek basin the Mammoth vein is made up of three splits—the Bottom, Middle, and Top, while in other portions the dividing strata between the benches amount to only 1½ to 2½ ft., and the vein is practically in one. The section of the Mammoth vein shown in Fig. 4 was taken at the Greenwood colliery, and shows a thickness of 60 ft. The miners know it by benches, as follows: The Three-Foot, the Four-Foot, the Eighteen-Inch Slate, the Bony, the Grey Slate, the Grain Clear, the Five-Foot, the Seven-Foot, the Slaty or Dirty benches, the Blue Slate, and the Top bench.

The present-day methods of mining are an outgrowth of the experiences of the last 100 years in this territory. The first attempt at mining, according to the company's maps, was made at Summit Hill in 1792, when open cuts and pits were made. Later the pillar-and-breast method was introduced. This and other methods will be discussed hereinafter.

Drifts, slopes, and shafts, according to the requirements and physical conditions, are the main openings, from which the gangways are driven. The gangways (haulageways) and airways are driven along the strike of the vein; the gangways in the various veins on the same level being connected by tunnels through the intervening rock. The gangways and airways in thick veins are heavily timbered throughout. The sets of timber, consisting of a collar and two legs, are spaced 5 ft. apart, and later, after the ground has settled, additional sets, called "liners" or "relief sets," are placed between, to relieve the strain on the original timbers, so that the result is a set of timber every 2½ ft.

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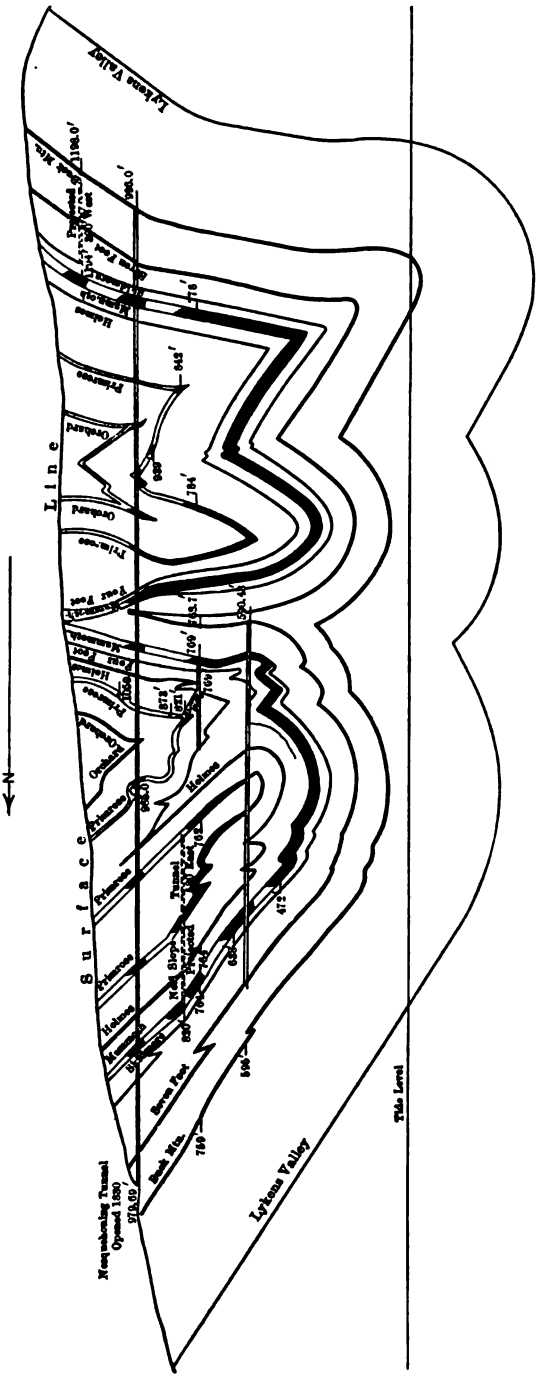


FIG. 1.—CROSS-SECTION OF THE NESQUEHONING TUNNEL.

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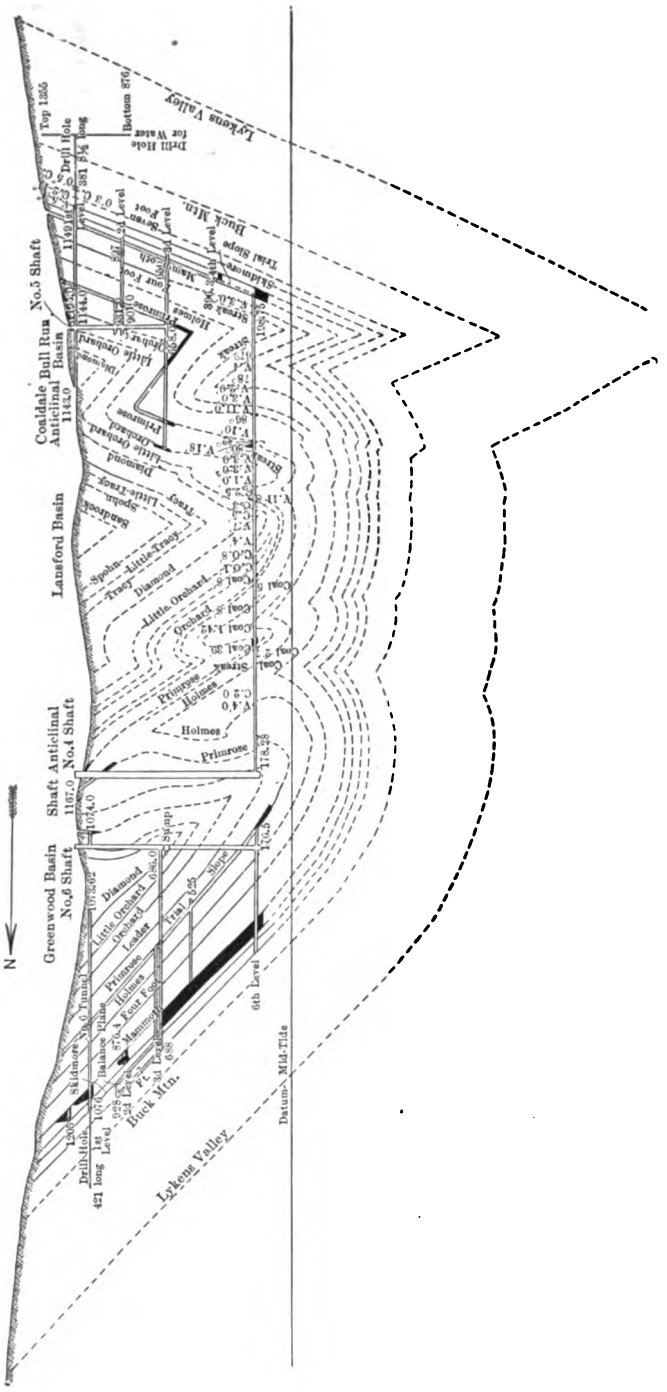


FIG. 2.—CROSS-SECTION THROUGH PANTHER CREEK VALLEY AT LANSFORD COLLIERY ON LINE OF NOS. 4, 5, AND 6 SHAFTS.

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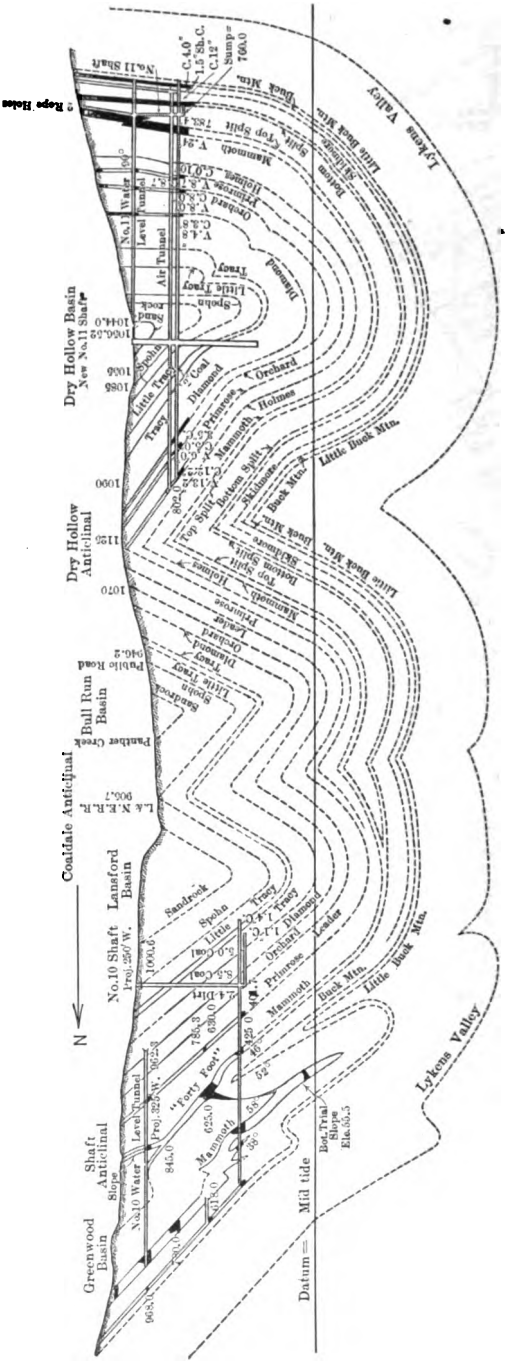


FIG. 3.—CROSS-SECTION THROUGH PANTHER CREEK VALLEY AT GREENWOOD AND RAHN COLLIERIES ON LINE OF NOS. 10 AND 11 SHAFTS.

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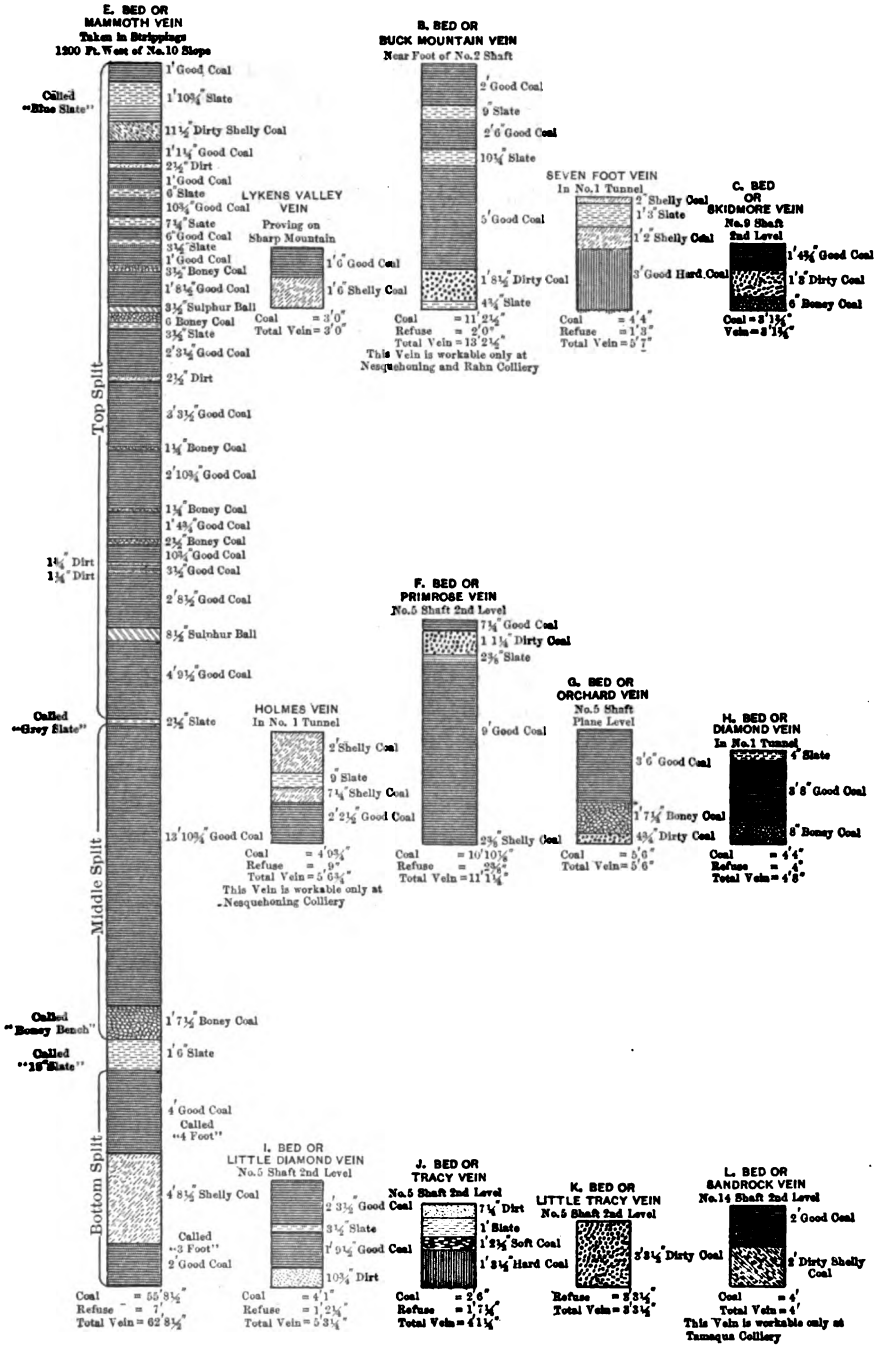


FIG. 4.—SECTIONS OF THE COAL BEDS IN THE PANTHER CREEK VALLEY.

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The sizes of the ordinary timbers, when of wood, are: Collar, 8 ft. 6 in. long and 15 in. diameter; legs, each 9 ft. long and 14 in. diameter, set on a batter of 3 in. to the foot. A section of the gangways in the clear, therefore, is 7 ft. 6 in. in width at the top and 11 ft. 9 in. in width at the bottom at the height of the track, and 7 ft. 6 in. high above the track.

The size of the timber in the airways is 10 to 12 in. diameter when of wood, and the size of the opening in the clear is 5 ft. in width at the top and 8 ft. in width at the bottom, and 6 ft. in height. A considerable amount of steel timber is used, the sizes being for gangways a 10-in. I beam, 25 lb. per foot, for collar, and 8-in. H columns, 34 lb. per foot, for legs; for airways, a 9-in. I beam, 21 lb. per foot, for collar, and 6-in. H columns, 23.8 lb. per foot, for legs. Some 6-in. H beams, 23.8 lb. per foot, are being tried out for collars instead of the I beams, and so far are giving better results. Where wood is subject to dry rot the use of steel timber has been very successful, but where there is a constant squeezing or heaving it has been found inadvisable to substitute it for wood. A very considerable saving in timbering maintenance has been made up to this time by the use of steel, and its use is being extended.

All turnouts and permanent openings, where timbering is necessary, are being timbered with steel, and sections of various sizes up to 70 lb. per foot are used, according to the requirements. I cannot state at this time what the average life of a set of steel timber in a gangway will be, but it will probably be at least 10 years. The average life of wooden timber in a gangway is three years. In many instances wooden timber has lasted 15 years, and in many others it has lasted less than one year, depending entirely upon the physical conditions. There are also many instances where it was almost physically impossible to hold open certain stretches of gangways and airways until after steel timber had been installed.

Figuring the average life of wooden timber as three years, and of steel timber as 10 years, the saving made by using steel will be fully 100 per cent.

A notable instance of the saving of steel timber as against wood was in the case of a turnout which had to be retimbered every nine months at a cost of $20 per set for labor and material. The steel timber cost $40 per set for labor and material. Steel timbering was installed four years ago, and will last many years more. It will be seen that the saving amounts to more than $60 per set to the present time.

We use practically all peeled timber, although at times we are unable to procure it. We have not used treated timber in the mines, although some of the other anthracite companies have experimented rather extensively with it for a number of years at one or two of their collieries. We are experimenting with steel timbering instead of wood in a few of our

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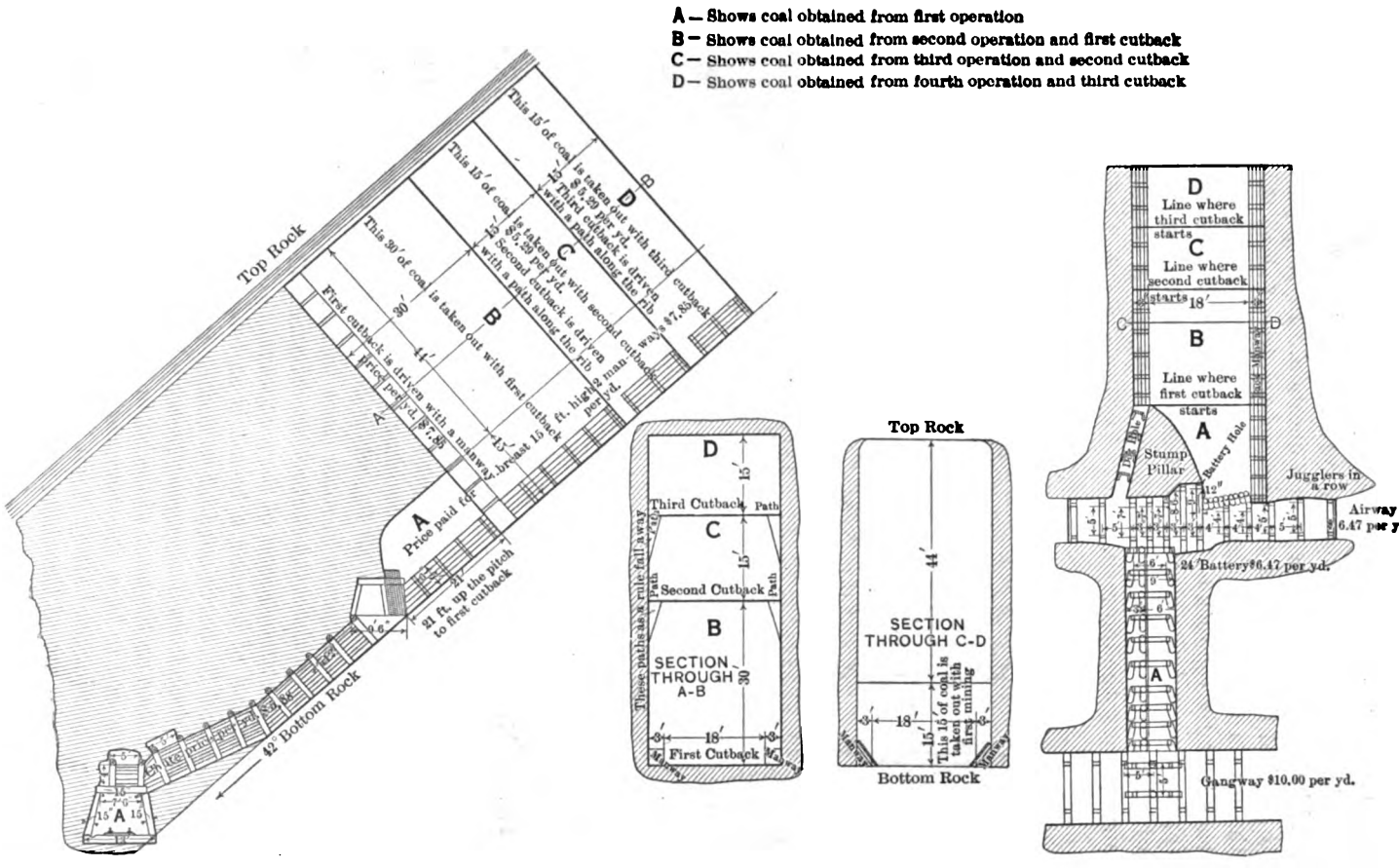
A — Shows coal obtained from first operation
B — Shows coal obtained from second operation and first cutback
C — Shows coal obtained from third operation and second cutback
D — Shows coal obtained from fourth operation and third cutback
Top Rock
This 15′ of coal is taken out with third cutback
15′ Third cutback is driven with a path along the rib
$5.39 per yd.
This 15′ of coal is taken out with second cutback
15′ Second cutback is driven with a path along the rib
$5.39 per yd.
This 30′ of coal is taken out with first cutback
First cutback is driven with a manway breast 15 ft. high
$7.83 per yd.
Price paid for
21 ft. up the pitch to first cutback
42′ Bottom Rock
Third Cutback
Second Cutback
First Cutback
Path
Manway
These paths are to rule fall away
SECTION THROUGH A-B
SECTION THROUGH C-D
This 15′ of coal is taken out with first mining
Bottom Rock
Line where third cutback starts
Line where second cutback starts
Line where first cutback starts
Stump Pillar
Battery Hole
Juggler in a row
Airway 6.47 per yd.
24′ Battery $6.47 per yd.
Gangway $10.00 per yd.

FIG. 5.—CUT-BACK IN NO. 62 BREAST, EAST MAMMOTH VEIN, NO. 10 SHAFT.

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main chutes, those which will have to be held open for five or six years, and believe it will result in a saving.

The proper position of gangways and airways with reference to the top and bottom rocks is sometimes a question. In veins pitching from 60° to vertical the gangways and airways are driven along the top rock, for the reason that the loading chutes, on about 30° pitch, can be driven back from the top rock to the bottom rock, providing a safe working place for loaders, and also so that the loose coal can be controlled. In veins with lesser pitches the gangways are driven on the bottom rock and the airway along the top rock.

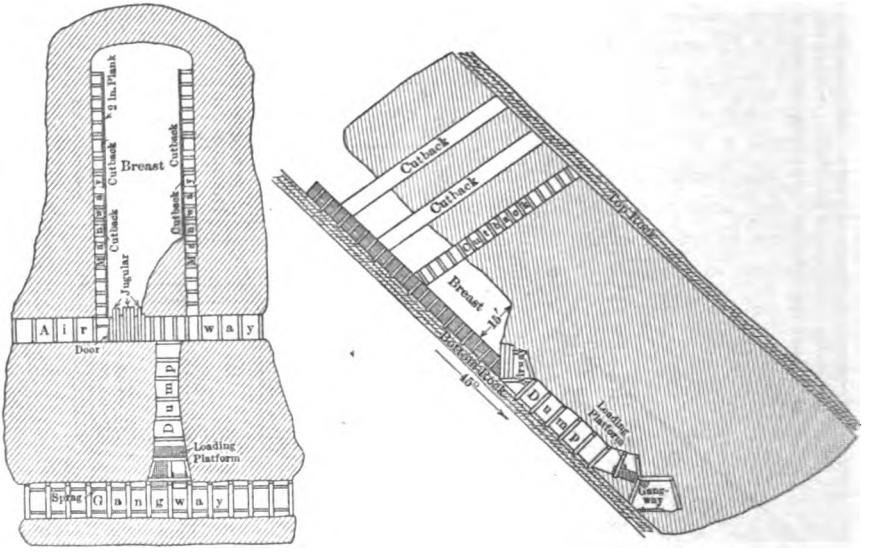


FIG. 6.—METHOD OF WORKING THE MAMMOTH VEIN, LOWER LEVEL, NO. 8 COLLIERY.

The distance between a gangway and its airway, when both are driven along the same rock, varies according to conditions. Years ago the distance varied from 20 to 30 ft., while during the last few years it has been found advantageous in some cases to increase this distance to 50 ft. in the sections where the chutes were driven on 60-ft. centers, resulting in a lower maintenance cost of the gangways.

The method of opening a breast is as follows (see Figs. 5 and 6): Chutes, 6 by 6 ft., partitioned off to make a traveling way and a loading chute, are driven from the gangway on a pitch of 22° until the bottom rock is met, and then up the pitch of the bottom rock, to the height at which it has been decided to have the stump heading or bottom breast cross-cut. At this point a juggler battery is put in and a 4 by 6 ft. cross-cut is driven connecting the top of the chute with the top of the next chute. This

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cross-cut is used for ventilating and travel, and is locally known as the "bottom-breast cross-cut."

In forming a breast, the "front" manway or dog hole is driven on one side 20 ft. up the pitch and a back manway carried on the other side; both these are opened off the bottom-breast cross-cut. The opening between the back manway and the battery collar is 10 ft. wide and 8 ft. high, and this opening is carried up the pitch and gradually widened until it reaches the top of the dog hole, where a connection is made, forming what is called a "stump" above the battery. The breast is now 18 ft. wide between the manways. A length of manway is put in on top of the dog hole and the first cut-back is made in the following manner: The top of the dog hole forms the bottom of the cut-back; manways are carried on both sides and the coal is blasted 18 ft. wide between the manways and 10 ft. high all the way back to the top rock of the vein. After the first cut-back has been completed the breast is driven 30 ft. farther up the pitch, where the second cut-back is made. The second and additional cut-backs are similar to the first. Where the pitch of the vein is 45°, for instance, the loose coal does not fill the whole space made. In this case the miners carry a "path" along each rib of the cut-back. Two miners blast the coal, each traveling his own path, until the top rock is reached and all the coal is taken out between the first and second cut-backs. The breast is again driven up the pitch 15 ft., where the third cut-back is made in the same way as the second. If the breast does not run away, or the top coal does not fall, a cut back is made every 15 ft. up to the old level. In steep pitching veins no paths are necessary, the miners standing on the loose coal to make the cut-backs.

It frequently happens that the breast runs away after the second or third cut-back is made, in which case a chute is driven along the back manway of the breast to a point about 30 or 40 ft. above the face, a battery is put in, and another breast is driven similar to the one just described. If this breast runs away, the chute is carried farther up the pitch past the face of the runaway breast and another breast opened, probably reaching the old level.

At one colliery where the vein is practically vertical and the breasts are driven along the bottom rock, the manways are driven along the top rock, and this has been found of great assistance, because the manways are not so liable to become blocked by falls, thus interfering with the ventilation of the breast.

The driving of breasts as I have just described is the common practice.

Where the vein is divided into its various splits, a different method is used. At the Tamaqua colliery there are a bottom and a middle split, the parting slate being from 12 to 18 ft. thick. The bottom split is 7 ft. thick and the middle split 4½ ft. thick. The gangway is driven in the bottom split, as is also the airway, 40 ft. above. Rock



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holes, 50-ft. centers, are driven from the bottom-split airway to the middle split. The middle split is mined by breasts 8 yd. wide, and robbed by skipping the pillars, beginning at the top. After the completion of the robbing, the bottom split is mined in the same way directly underneath the middle-split breasts. The cost of coal mined by this method is high because two breasts have to be driven for 12 ft. thickness of coal.

It was originally planned to use the breast method in the middle split and the chute method in the bottom split; the chutes being driven while the breast was being worked. Care was taken, however, that the robbing at the top of the chutes in the bottom split was not commenced until the robbing had been completed at the top of the breast in the middle split. It was found that as soon as the pillars were cut in the middle split there was so much squeezing and heaving in the chutes in the bottom split that it was almost impossible to hold them open, and the plan had to be abandoned. At the present time, breasts are being driven in both splits satisfactorily, except as to high cost.

Many years ago breasts were driven 10, 12, and 20 yd. wide, and in the old Rhume Run tunnel of Nesquehoning colliery some were driven 30 yd. wide. At the present time breasts are driven 8 yd. wide, and where the vein is free not more than 6 yd. wide. I should say about 40 per cent. of them are 6 yd. wide.

As the width of breasts has decreased, the width of the pillars between breasts has increased. For instance, where 12-yd. breasts were driven on 50-ft. centers, the pillars were 14 ft. wide. At the present time 6-yd. breasts driven on 50-ft. centers give a width of pillar of 32 ft.

In thick veins breast work is preferable to chute work, especially where there is a good supply of air. The amount of blasting necessary is very much less, and consequently the yield of prepared sizes is larger. This is due to the fact that after the first cut-back has been made there is a loose end. Very often a breast will run away after the second or third cut-back has been made, or sooner if the vein is soft, and 2,000 or more cars will be loaded out before the breast has run through to the old lift above. However, when a breast runs it always fills with gas, and remains so until it has run through to the old workings, and this condition requires great care to avoid explosions.

In chute mining in new ground the chutes are driven almost to the old lift above, and as soon as the upper section of the vein has been worked and breaks through to the old workings there is not much further trouble with gas.

The coal is mined by driving chutes back toward the top rock, where breasts are driven first along the top rock, and later up the pitch. Retreat is made down the main chute for 30 to 50 ft. and another chute driven to the top rock, and the operation repeated. This method is carried on until the chute has been robbed down.

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All the coal has to be blasted, as heavy falls cannot be expected on account of the shortness of the sections, and also because there are no loose ends. The maintenance cost of the chutes is very high. The result of a test made at one colliery from 1907 to 1909 showed that the breast method cost only 60 per cent. as much as the chute method. However, the recovery by the breast method was only 92 per cent. as much as by the chute method.

The most important question in mining, especially in so far as the ultimate yield of the vein is concerned, is the robbing of the pillars.

In the early mining, the practice was to cut the pillars at the bottom of the breast by stripping a slice off the pillar at the top of the dog hole and bearing into and cutting it through if possible. It was followed up the pitch until it became dangerous to follow any farther, or until the gob from the old breasts rushed in. By this method a large percentage of the pillar was left standing, as the heavy rock from the breast would soon rush in and the place would be abandoned.

A later method was, to drive a chute in the center of the pillar some distance up the pitch, where the pillar was cut through from the rib of one breast to the rib of the other. A larger percentage of coal was obtained by this method, but it was soon found that the maintenance cost was very high, due to the squeezing of the pillars, although it was more successful on the slighter pitches than on the heavy pitches. On the heavy pitch, the coal and rock coming down the chute knocked the timber out, causing heavy maintenance charges.

Another former method was to couple up the old manways of the breasts on each side of the pillar and put in a battery (the old breasts being left full of coal). The bottom benches were taken out between the manways and were cut back in the middle of the pillar toward the top rock, care being taken to leave a small pillar on each side above the old manways which had been coupled, in order not to break them. After these pillar breasts had been driven up to the same height as the face of the old breasts, loading was commenced from the three batteries, trusting that the small pillars over the manways would crush, and all of the coal would be won. This method was found objectionable, as the pillars squeezed so much that it became too dangerous for men to continue to work, and they were seldom able to drive the pillar breast as far as the face of the old breast.

The chute method of robbing pillars was then tried out. These chutes were driven about 25 ft. apart and to within 30 ft. of the old level above, and robbing commenced at the top. It was first thought that by this method it would be easier to reach the coal between the chutes; however, it was soon noticed that so many openings caused such a general squeezing in the chutes that it was almost impossible to keep them open, and in fact many of them were lost. This distance between centers was then

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increased to 50 ft., and later to 60 ft. The main chutes were connected up by slant chutes, which were used for ventilation and later for robbing the various blocks of coal in the pillar. This method is the present practice.

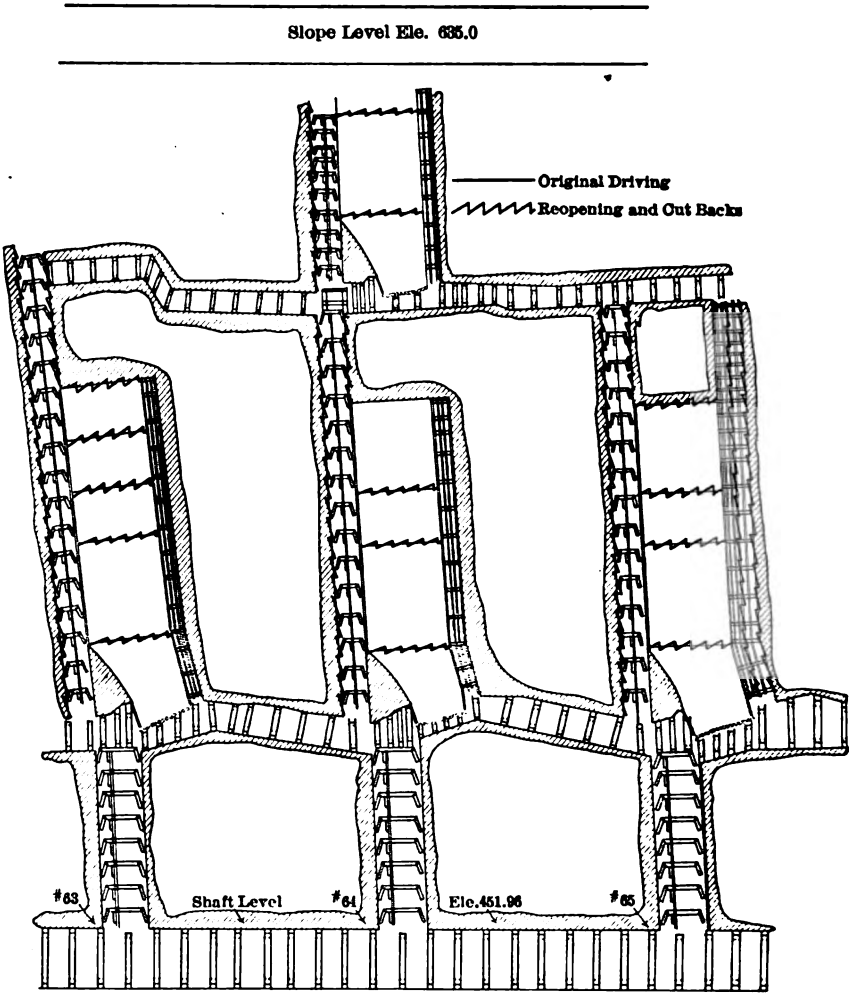


FIG. 7.—METHOD OF WORKING BREASTS AND PILLARS, EAST MAMMOTH SHAFT LEVEL, COLLIERY NO. 10.

At the Greenwood colliery a record was kept of every step in the driving of two adjoining breasts and the robbing of the pillar between (see Figs. 7 to 9), in order to determine what recovery of the vein was being made by the method in general use. At this point the vein was 59 ft. thick. The section was 190 ft. long. The total contents of this

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section were 590,000 cu. ft. of solid coal, or 8,260 mine cars of 125 cu. ft. each. There were loaded out 5 442 mine cars, a 65.9 per cent. recovery. This work was done in 1909, 1910, 1911, and 1912.

The method of robbing the pillar was as follows: Chutes were

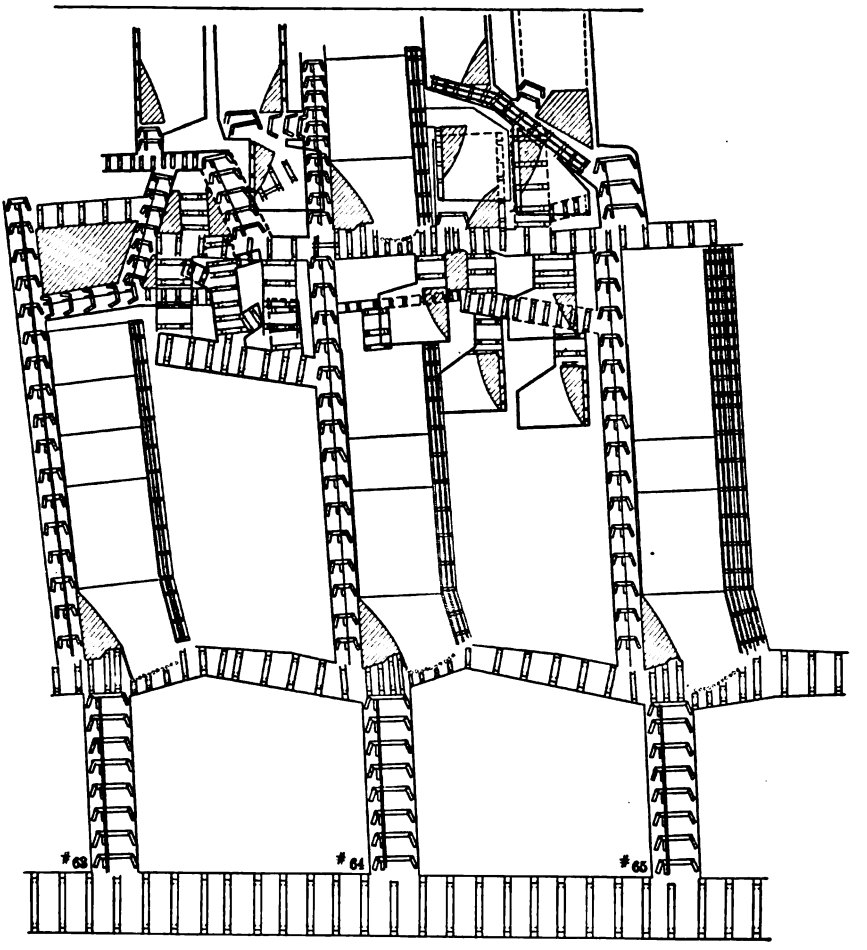


FIG. 8.—METHOD OF WORKING BREASTS AND PILLARS, EAST MAMMOTH SHAFT LEVEL, COLLIERY NO. 10.

first driven along the manways; slant chutes were driven across the pillars, and a chute driven back almost to the top rock. Then a breast was driven up the pitch and the coal mined. Chutes along the small pillars between this breast and the old breasts recovered that part of the vein. After all of the coal had been mined near the top rock, another

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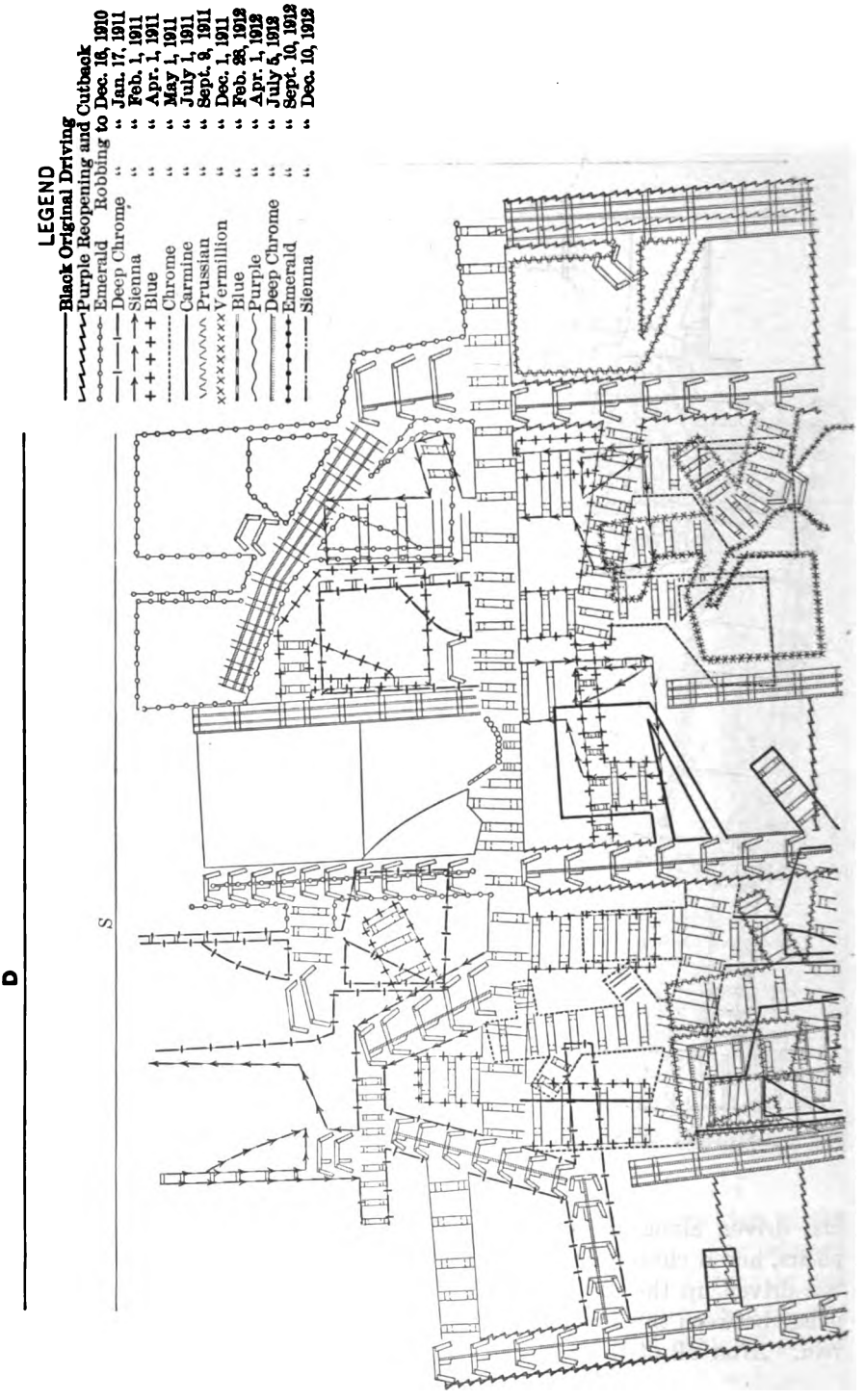
LEGEND
Black Original Driving
Purple Reopening and Cutback
Emerald Robbing to Dec. 16, 1910
Deep Chrome " Jan. 17, 1911
Sienna " Feb. 1, 1911
Blue " Apr. 1, 1911
Chrome " May 1, 1911
Carmine " July 1, 1911
Prussian " Sept. 9, 1911
Vermillion " Dec. 1, 1911
Blue " Feb. 28, 1912
Purple " Apr. 1, 1912
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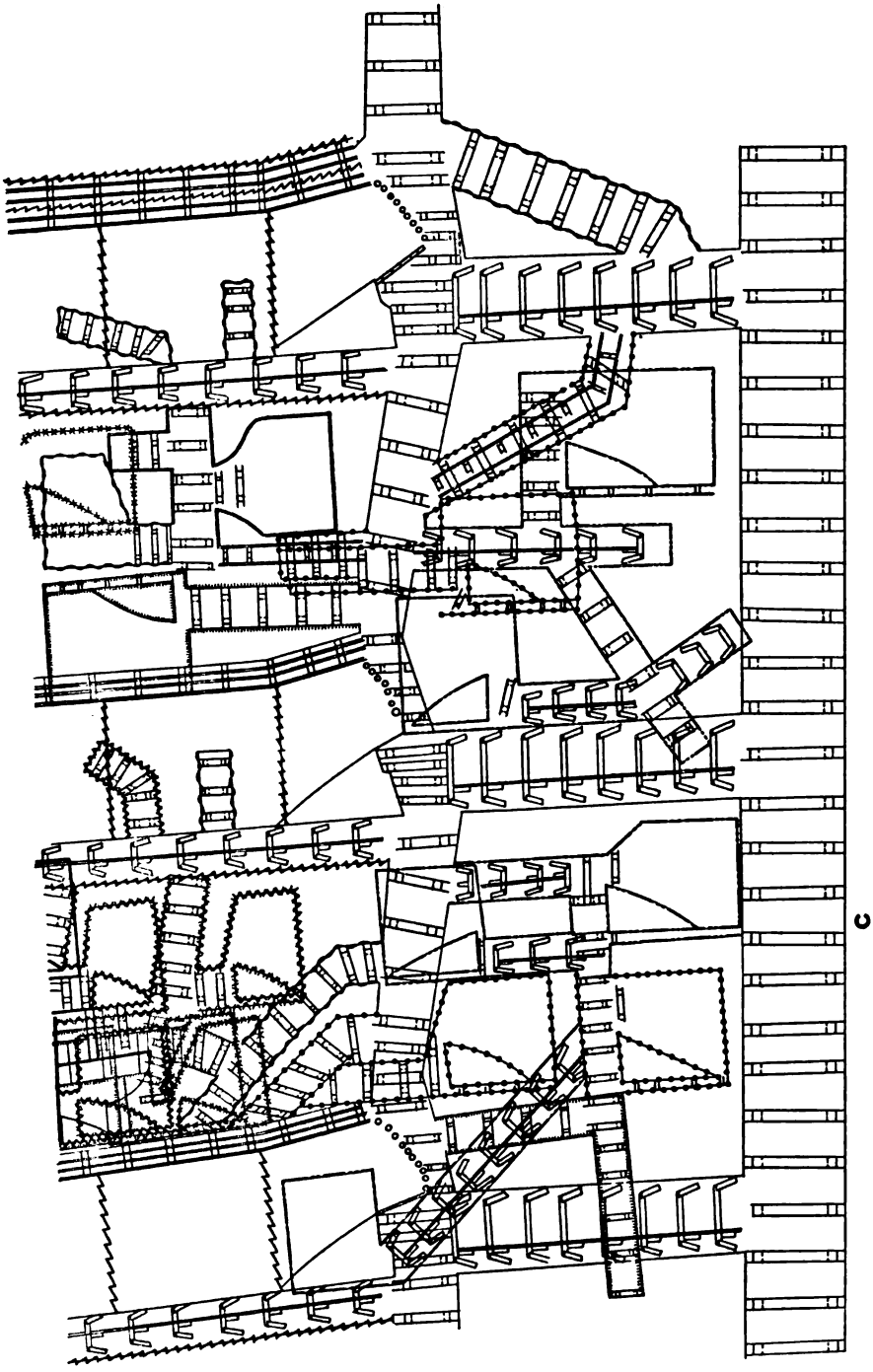


FIG. 9.—METHOD OF WORKING BREASTS AND PILLARS, EAST MAMMOTH GANGWAY, SHAFT LEVEL, No. 10 SHAFT, GREENWOOD COLLIERY.

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breast was opened off the main chute (driven back through the pillar), and later chutes were driven to mine the small pillars between the breast and the old breasts. When all of the coal possible had been mined in this way toward the top rock, a breast was opened up the pitch along the bottom rock. Again chutes were driven to mine the small pillars between this breast and the old breasts. After all of this had been done the upper part of the pillar was supposed to have been robbed completely; then retreat was made down the chute about 30 ft. and a main chute driven back through the pillar, and the whole operation repeated.

The drawings show a network of chutes and breasts. It has been

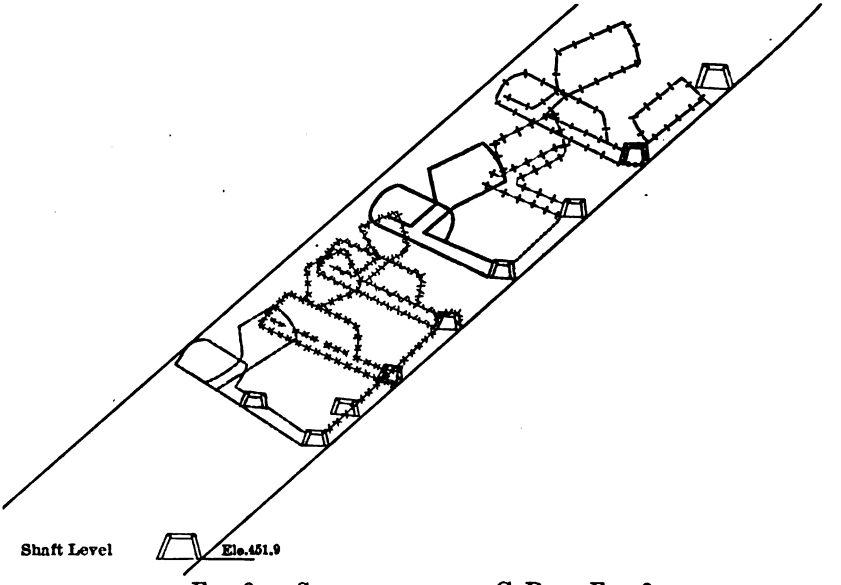


FIG. 9A.—SECTION THROUGH C–D OF FIG. 9.

found by experience that, in order to recover a good percentage of the coal, where the vein is so thick, chutes must be driven in any and all small pillars that have been left between any of the breasts or chutes.

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*Notes to Fig.* 10. (See opposite page.)

*A*. Rock Chutes, Skidmore to Bottom Split of Mammoth. About 30 ft. long on 35°, 6 by 8 ft. Chute, 120-ft. centers.

*B*. Slant Chutes on Bottom Rock of Bottom Split of Mammoth. About 70 or 80 ft. long on 35°, 6 by 6 ft. Chutes.

*C*. Straight Chutes up the Bottom Rock of Bottom Split of Mammoth. About 20 ft. long on 70°, 5 by 6 ft. Chute.

*D*. Breasts on Bottom Rock of Bottom Split of Mammoth. About 160 ft. long to surface on 70°, 8-yd. breast, 50-ft. centers.

*E*. Tapping Chutes from Bottom Split to Middle Split of Mammoth. Used as batteries for running coal.

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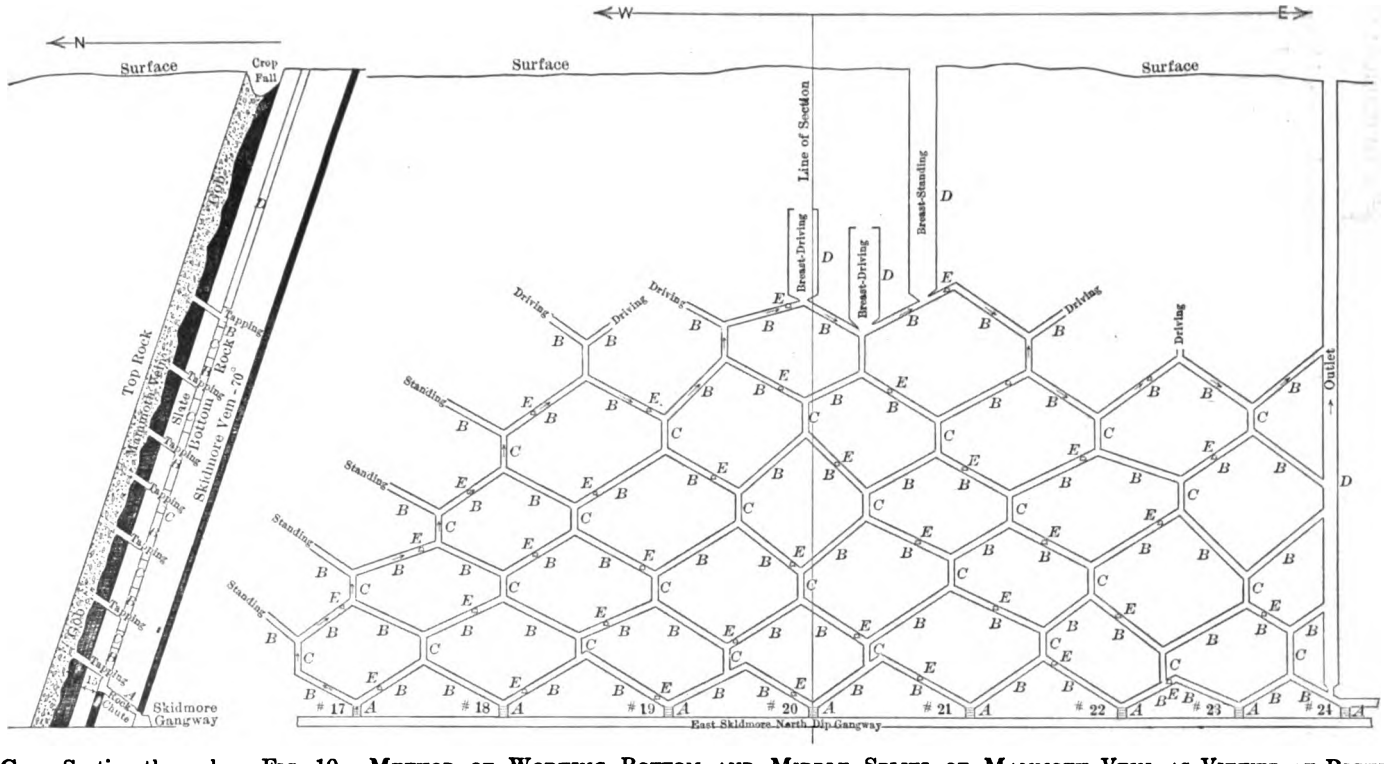


Cross-Section through Chute No. 20.

FIG. 10.—METHOD OF WORKING BOTTOM AND MIDDLE SPLITS OF MAMMOTH VEIN, AS VIEWED AT RIGHT ANGLES TO THE PITCH. NO. 5 SHAFT, SECOND LEVEL, EAST SKIDMORE, NORTH DIP GANGWAY.

(See notes on opposite page.)

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In every case, however, the pillar being robbed must be controlled; it cannot be allowed to run away.

The reworking of old territory is always an interesting problem Years ago it was the practice to reopen an old gangway in the vein and as a rule the cost was heavy, especially if the ground had not been standing long enough to settle thoroughly. In some instances ground has settled in six or seven years, while other ground has not settled in 15 years. The most satisfactory way to rework the old Mammoth territory has been by driving the gangways in the Skidmore vein and driving rock chutes to the Mammoth vein. At first these rock chutes were driven about 100 ft. apart. This distance has now been extended to 120 ft., 140 ft., and even 150 ft. at one colliery. The rock chutes are connected at the top with slant chutes; then, about 10 ft. off the rib of the rock chute, main chutes are driven straight up the pitch, and these main chutes connected by slant chutes across the pitch. These slant chutes provide ventilation and also determine the location of any pillars which had not been worked. At several collieries where the pitch is almost vertical, a straight or box chute about 40 ft. in length is driven up the pitch from the apex or meeting point of the two slant chutes (Fig. 10). At the top of the box chute two other slant chutes are driven, and again at their meeting points a box chute is put in. This is continued and we are able to reach up the pitch any distance desired. In many cases we have driven up the pitch over 400 ft. This method has proved its value and at all times the miners are held safe.

As a rule this system of chutes is made in the bottom bench of the vein, which in most cases has been found solid; the reason being that the coal was extremely hard and could not be mined at a profit many years ago. At intervals, holes are driven through the 18-in. slate into the middle and top benches of the vein, and the gob from the old breasts is drawn out. Where the gob is of such a nature that it can be driven through, chutes are driven on a pitch of 30° to 35° through the vein into the top rock. A pitch of 35° is not too heavy through gob, as more rock has to be handled, and the gob itself is sticky or gummy. Where the bottom bench has been worked, it is necessary to do all of the opening work for robbing purposes in the gob. On the steep pitches we prefer to make the chutes on a pitch of from 30° to 35° and drive across the vein (along the strike) or else directly through it (at right angles to the vein), for the reason that in the gob it is necessary to protect the roof of the chute by driving poles, 6 to 10 ft. in length and 4 to 6 in. in thickness, ahead of the actual working face, to give full protection for the placing of the next set of timber.

It is readily seen that in loose ground, if the chutes were driven directly up the pitch on say 70°, the poling process would be of no benefit, and miners would be in danger from falls of material.

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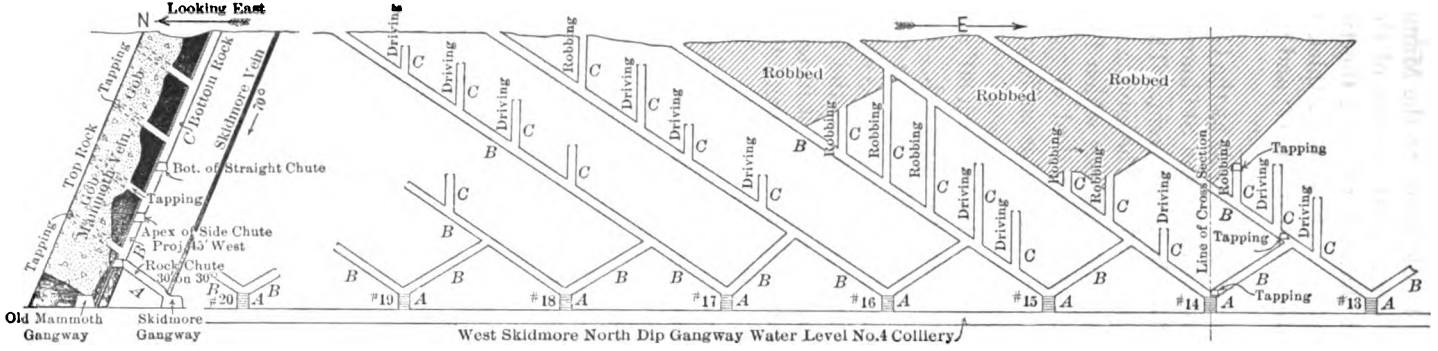


Cross-Section through Chute No. 14.

*A*. Rock Chutes from Skidmore to Mammoth. 6 by 8 ft. Chute, 35 ft. on 35°, 100-ft. centers.

*B*. Slant Chutes on Bottom Rock of Mammoth. 7 by 7 ft. Chute on 35°, 300 ft. to surface. Side slants 55 ft. long.

*C*. Straight Chutes up on Bottom Rock of Mammoth. 5 by 5 ft. Chute, 60 to 100 ft. long on 70°. These Chutes on about 35-ft. centers, depending on condition of gob.

FIG. 11.—METHOD OF WORKING THE MAMMOTH VEIN, AS VIEWED AT RIGHT ANGLES TO THE PITCH. NO. 4 WATER LEVEL, WEST SKIDMORE, NORTH DIP GANGWAY.

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In driving rock chutes from the Skidmore to the Mammoth vein, the question often arises as to the economical spacing of the rock chutes, due to the varying thickness of the rock between the two veins. Our experience has been that where the rock is 30 ft. thick the chutes can be driven 100 ft. apart; 40 ft. thick, 120 ft. apart; and 90 ft. thick, 150 ft. apart.

About 1891, when the fourth and fifth levels at the Coal Dale colliery were opened, the gangways were made in the Skidmore vein and rock chutes driven every 50 ft. to the Mammoth vein. These chutes were about 30 ft. in length and on a pitch of 35°. The mining in the Mammoth vein was done according to the usual methods which I have described. These Skidmore gangways required a great deal of repair, due to the very poor top rock of the Skidmore vein at this colliery, and also to the small distance between the chutes.

At one colliery, where the pitch varies between 80° and 90°, a somewhat different system of slant chutes was adopted. The slant chutes were driven across the pitch at about 30° and along the bottom rock. At the junction point of the slant chutes a chute was driven through the vein to the top rock, then turned back again to the bottom rock and slant chutes again driven. There are a number of instances where more than 1,000 ft. of such chutes have been driven from the top of one of these rock chutes.

In other cases (depending upon the conditions), instead of driving the rock chute from the Skidmore to the Mammoth, a chute has been driven up the pitch in the Skidmore to the predetermined height, and then a rock chute driven into the vein, the object being to make the opening above the point where the breast has run away and filled with rock. Where we have been able to work this method it has proved successful.

A variation of this method is being tried out at the Lansford No. 4 colliery (Fig. 11), where the vein is on a pitch of 72°. The rock chutes are driven from the Skidmore gangway to the Mammoth, and from the top of these chutes long slant chutes—7-ft. collar and 7-ft. leg—on a pitch of 35° are driven along the bottom slate of the Mammoth vein to the bottom of the old breaches on the surface. This is being done for the purpose of re-robbing the old workings between the old Water Level gangway and the surface, the lift being about 300 ft. From these slant chutes, straight chutes—5-ft. collar and 5-ft. leg—are driven up the pitch at about 35-ft. centers, depending upon the condition of the gob. The robbing work is started at the top and is continued down the pitch. Great care is taken to keep the robbing faces far enough ahead of the chutes driven later, in order to not bring on a squeeze.

Still another variation is being tried out at Lansford No. 4 colliery, fourth or Slope Level. This plan (Fig. 12) differs only from the scheme

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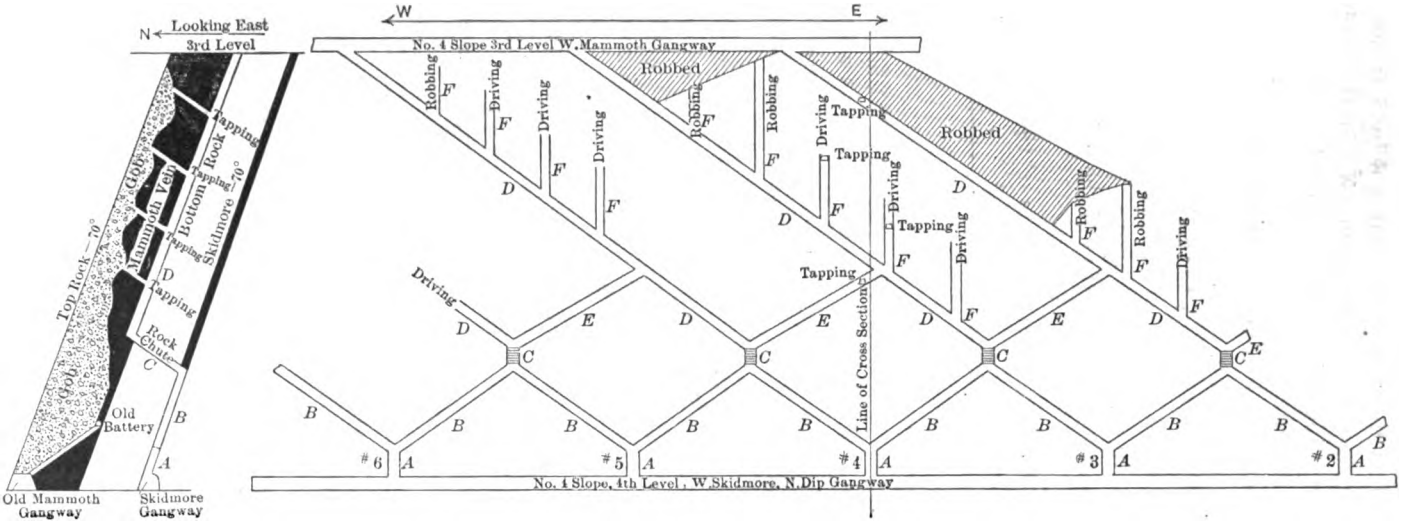


Cross-Section through Chute No. 4.

*A*. Straight Chutes on Bottom Rock of Skidmore Vein. Chute 15 ft. on 70°, 6 by 8 ft. Chute, 140-ft. centers.
*B*. Slant Chutes on Bottom Rock of Skidmore Vein. Chute 80 ft. on 35°, 6 by 8 ft. Chute.
*C*. Rock Chutes from Skidmore to Mammoth. Chutes 30 ft. on 35°, 6 by 8 ft. Chute.
*D*. Slant Chutes on Bottom Rock, Mammoth Vein. Chute 300 ft. to third level on 33°, 7 by 7 ft. Chute.
*E*. Slant Chutes on Bottom Rock, Mammoth Vein. Chute 80 ft. on 35°, 7 by 7 ft. Chute.
*F*. Straight Chutes on Bottom Rock of Mammoth Vein. Chutes 60 to 100 ft. on 70°, 5 by 5 ft. Chute, 35-ft. centers.

FIG. 12.—METHOD OF WORKING THE MAMMOTH VEIN, AS VIEWED AT RIGHT ANGLES TO THE PITCH. NO. 4 SLOPE, FOURTH LEVEL, WEST SKIDMORE, NORTH DIP GANGWAY, LANSFORD COLLIERY.



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used in the Water Level, just mentioned, in that a 6 by 8 ft. chute is driven from the Skidmore gangway straight up the pitch in the Skidmore vein for 15 ft., these chutes being spaced 140 ft. apart on the gangway. From the top of the straight chutes, slant chutes on a pitch of 35°, 6 by 8 ft. in size, are driven along the bottom rock of the Skidmore vein for about 80 ft. and connected at the top. From that point a rock chute—6 by 8 ft.— on a pitch of 35° is driven back to the Mammoth vein, about 30 ft.; then long slant chutes are driven on 33° pitch along the bottom rock of the Mammoth vein to the old level above. The intention was to strike the Mammoth vein above the batteries of the old breasts which were filled with rock.

We have lately arranged to make further tests of these schemes in

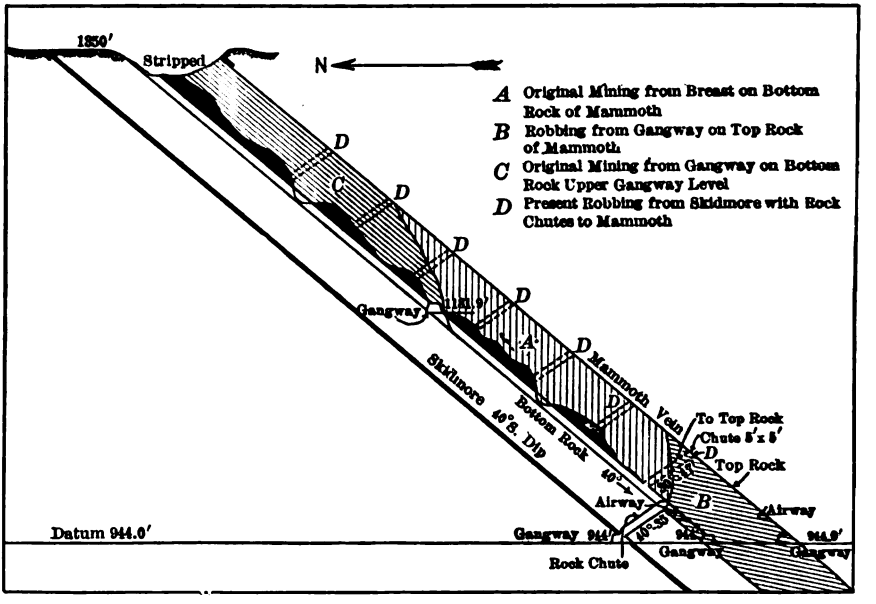


FIG. 13.—CROSS-SECTION SHOWING ORIGINAL MINING AND PRESENT ROBBING OF MAMMOTH VEIN, WATER LEVEL, COAL DALE COLLIERY.

working new territory, in order to make comparisons with the results obtained from the old established pillar-and-breast method.

The question naturally arises, which is the better of these chute plans. The chute yardage driven is practically the same in both methods, with a little saving made in the long slant chute method. The great advantage of the long-chute method over the box-chute method lies in the flow of the coal throughout practically all of its course on the light pitch, causing very much less breakage; the only real drop of the coal being in the straight chute immediately at the robbing point. In the

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box-chute method there is bound to be considerable breakage in the straight box chutes unless they are kept full at all times, and this, we know, is frequently neglected by the workmen. At best there is bound to be a certain amount of grinding of the coal in the box chutes.

In many cases the present working is the fourth mining of the same piece of territory. The results of the old mining showed that on the first mining 30 per cent. had been recovered; in the first robbing, 16 per cent.; in the second robbing, 8 per cent.; and in the present working, 10 per cent.

At Coal Dale No. 8 colliery workings, about 60 years ago the first gangway was driven in the Mammoth vein along the bottom rock, and mining was done by the old pillar-and-breast method. After the gangway had been driven to its limit it was robbed out. About 15 years later another gangway was driven through the same territory along the top rock of the vein, and this in turn was robbed out. About 30 years ago another gangway was driven at a level about midway between the two gangways just mentioned and the crop; later the gangway was robbed out. Five years ago, believing there was a large quantity of coal to be won in this territory from the Mammoth vein, we began to drive a gangway in the Skidmore vein 30 ft. underneath the Mammoth and drove rock chutes, 100-ft. centers, to the old Mammoth workings. We figure the present recovery from this section at 10 per cent.

If the present method of mining—that is, mining the Mammoth vein from the Skidmore vein by means of rock chutes—had been adopted in the original mining, there could have been recovered not only the 10 per cent. now being obtained, but also the 54 per cent. recovered by means of the three gangways and their airways above mentioned, with their various breasts and chutes.

The writer had a record kept of the recovery and the cost of working 900 ft. of this territory during the past five years, and estimates that the $45,000 or $50,000 which these old openings cost could have been saved. The cross-section (Fig. 13) of the vein shows the position of the old gangways and the present Skidmore gangway.

In mining very thick veins on steep pitches, it will always be possible, even though the most improved methods are used in the original mining, to win some coal by re-mining, but the cost of obta ning what remains will depend upon market and wage conditions, and may be prohibitive.

Many years ago no coal except that of the very best fracture and best appearance was mined, because it was the only kind found salable. Also, only sizes above chestnut (made over a $\frac{7}{8}$-in. round screen) could be sold. In recent years, all sizes including No. 4 buckwheat (made over a $\frac{9}{16}$-in. round screen) are easily salable, and as a consequence, territory that years ago could not possibly be mined at a profit can now be made to pay.

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It has been found of great advantage to make panels of a territory where coal is virgin or where the distance between the Skidmore and the Mammoth veins is too great for practical use of rock holes. The length of these panels depends upon the distance between the veins; the panels in some cases being 600 ft. long, and varying up to 1,200 ft. The paneling system has been very much extended during the last five or six years in the Panther Creek field, and the results have been so satisfactory that the new lifts are being laid out and worked on this plan. At practically all points, none of the veins underlying the Mammoth are workable. The haulageway is being made in the Skidmore vein, which varies from a thin leader to 3 ft. in thickness, and tunnels are being driven at 1,000-ft. intervals from the Skidmore to the Mammoth, Primrose, and Orchard veins, and gangways opened in both directions off the tunnels, thus making a section of about 500 ft. of each gangway which can be mined and robbed. This will allow an increased number of working faces, and also a greatly reduced time through which the gangways must be maintained. In years to come this will result in a great saving in the mining of the Mammoth vein.

Where a panel tunnel is driven a pillar is left over it, this pillar being recovered when the gangway is being robbed out. Every 10 breasts a solid block of coal is left; that is, the coal which would be mined by the eleventh breast is left solid with the exception that the gangway and airway are driven through it. This block is mined on the retreat. The reason is that in case of a mine fire a solid piece is available from which to begin the fight to extinguish the fire. These pillars also have a checking effect on a "creep."

Experience has shown that when the Mammoth gangway, where the vein is 50 to 60 ft. thick or thicker, is being robbed out the best results are obtained by bringing back the face 80 to 120 ft. per year. To many people this has seemed unnecessarily slow, and many attempts have been made to hasten the work, but in every case there has been a very decided loss in the amount of coal recovered. This is due to the disturbance of the thick vein matter causing a general squeeze a considerable distance back from the point of actual robbing and making the maintenance of the gangway not only costly but almost a physical impossibility. In some instances a couple of hundred feet of gangway has been lost, owing to quicker robbing than that mentioned.

When opening a new colliery or a new lift at an old colliery, the question often arises whether it is not better to drive gangways and airways to the boundary limits before opening breasts; also whether it is advisable to open two lifts instead of one. I believe that with the proper system of paneling better results can be attained, because it is possible to have several working faces instead of one, enabling a quicker

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development. These panels can then be robbed out in a reasonable time, thus preventing heavy timber maintenance cost.

With the system of driving the gangways to the limit there is a period when nothing but development work can be done, and later nothing but robbing work, whereas in the paneling system some sections can be robbing while others are developing, thus keeping the product of more uniform quality.

The driving of the gangways and airways in the Skidmore vein has not only reduced the timbering expense, but it has improved the ventilation, for the reason that, the vein being so thin, there are no falls to block the airways.



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## Dip Chart

Discussion of the paper of HOWLAND BANCROFT, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 91, July, 1914, pp. 1767 to 1769.

THEODORE SIMONS, Butte, Mont. (communication to the Secretary*). —While preparing maps and models for use in mining litigations the writer was frequently confronted with the problem referred to in Mr. Bancroft's paper. His chart gives the apparent angle of dip with which a vein should be plotted on a vertical section not at right angles to the strike of the vein. For a quick determination of this apparent angle the writer has used an exceedingly simple graphical method, which he believes may prove helpful to engineers engaged in practical work. It is submitted, not as a criticism of Mr. Bancroft's paper, but as a means of solving the same problem when no chart is available and when calculations would consume too much time.

The writer's method requires merely a few lines marked off on a piece of tracing cloth, as shown in Fig. 1: Draw line *bc* on any convenient scale. At *b* erect a perpendicular, *ba*. By means of a protractor cut off on *ba* a number of angles within the probable range of variation of dip, so that a line connecting *a* and *c*, for instance, forms an angle of 60° with *bc*, etc. With *b* as a center and *bc* as a radius, draw a circle through *c*. This completes the tool for solving the problem.

Fig. 2 represents the line of strike of the vein and *db* the trace of the vertical plane on which the vein is to be plotted with the apparent angle of dip. $A°$ is the angle which this plane makes with the strike.

Fig. 3 shows the application of the method: Place line *cb* of tracing over line *db* of Fig. 2 and move fore and back until line of strike becomes tangent to the circle. Connect *d* with the point (*a*) that marks the true angle of dip as found by measurement in the mine (60° in the case illustrated). The angle *adb* is the apparent angle of dip ($C°$) to be used in the vertical section.

*Proof.*—As stated in Mr. Bancroft's paper, the apparent angle of dip is:

$$\tan C° = \sin A° \tan B° \qquad (1)$$

From Fig. 4, which is the same as Fig. 3 with a few lines added for sake of demonstration, we have:

$$\frac{ab}{bd} = \tan C° \text{ whence } ab = bd \tan C°;$$

also

$$\frac{ab}{bc} = \tan B° \text{ whence } ab = bc \tan B°;$$

* Received July 28, 1914.

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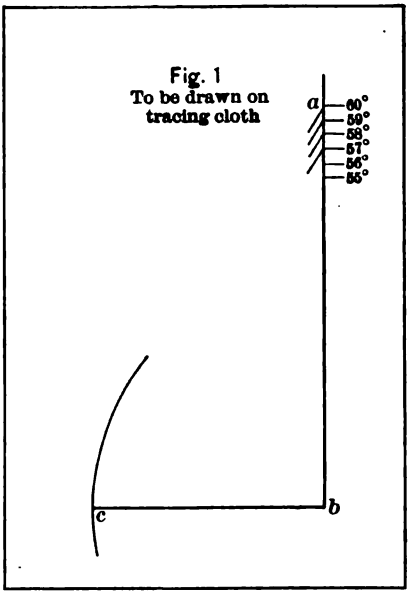


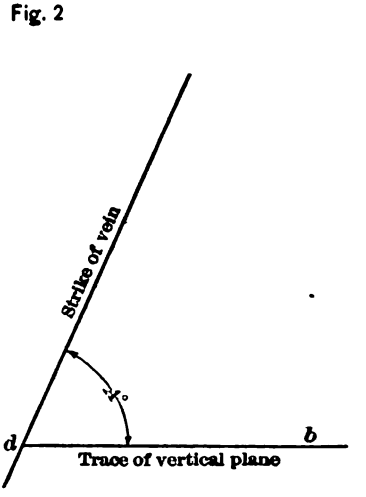


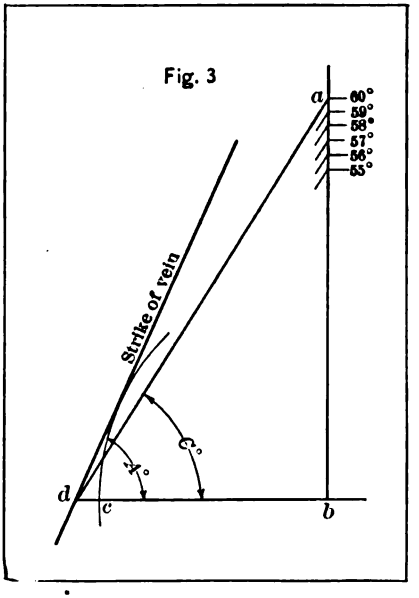


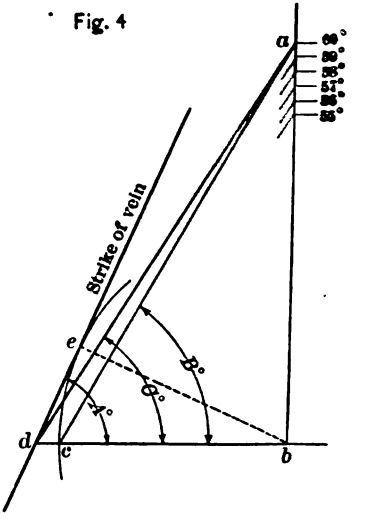


FIGS. 1 TO 4.—METHOD OF USING DIP CHART.

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whence $bd \tan C^\circ = bc \tan B^\circ;$

or $$\tan C = \frac{bc}{bd} \tan B^\circ \qquad (2)$$

From Fig. 4, $$\frac{bc}{bd} = \frac{be}{bd} = \sin A^\circ$$

Introducing in equation (2) we get:

$$\tan C^\circ = \sin A^\circ \tan B^\circ \qquad (3)$$

This is the same as equation (1) and shows that angle $C^\circ$ in Fig. 4 is the required angle.

FRANK A. LINFORTH, Butte, Mont. (communication to the Secretary*).—Referring to Howland Bancroft's paper, entitled Dip Chart, and also to the various other charts and formulæ for the correction of dip in making geologic cross-sections, I would like to emphasize the importance of making these corrections and to call attention to the method used by the geological department of the Anaconda Copper Mining Co. for this work.

Cross-sections are used extensively in Butte for laying out development work in the mines, for checking the interpretation of geologic facts on the separate level maps, and to some extent in legal work. They are not generalized drawings, but accurate records based on observed facts, and great care is taken to make them geometrically as well as geologically correct. The complex structure of the Butte district with its east-west, northwest, and northeast systems of veins and faults renders necessary the corrections for dip on almost every cross-section. If a section is laid out at right angles to any one system of veins, the observed dips for that system can be platted without alteration, but, obviously, the observed dips for the other two systems must be corrected before platting if they are to appear properly on that section. There are many examples in the geologic work at Butte where it has been necessary to make accurate vein correlations between new or partly developed mine levels. In some of these cases there have been a number of veins to choose from and the correct correlations have come from the application of corrected dips on the sections. That these correlations are correct is invariably proved by later developments in the mines.

Formerly these corrections were roughly calculated. More recently D. F. Hewett's excellent chart referred to in Mr. Bancroft's paper was enlarged and used in the routine work of the office. As a wall chart, however, it was found to be less convenient than a tabulated arrangement of values in which the angles of intersection with the section appeared across the top of the sheet and the angles of observed

* Received Sept. 21, 1914.

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dip were shown in the left-hand margin. The required value is found at the intersection of the vertical and horizontal columns under the proper headings. This is believed to be a better scheme than using the curves, but both are now supplanted by a small instrument devised and patented by the writer. This instrument, Fig. 5, is a symmetrically designed box 1¼ in. square, and about 8 in. long. A small square opening is cut in the top of the box near each end. One opening bears the designation, "Angle with section"; the other, "Observed dip." The observed values are made to appear in their respective openings by turning small knobs, one at each end of the box. Increments of 5° are used and this interval is found to be satisfactory for practical purposes, although in a slightly larger instrument this interval could be diminished. As soon as the observed values are set, the required value appears as a definite number of degrees in an opening cut in the top of the box. No

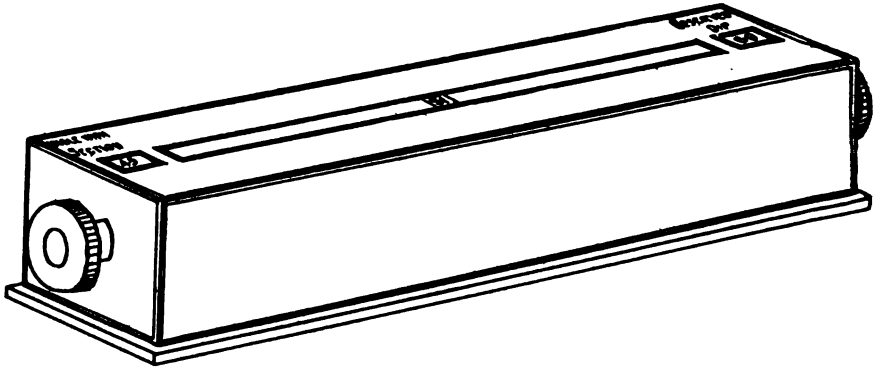

FIG. 5.—THE DIPOMETER.

other numbers appear and no chance for error is introduced, provided that the observed data are properly set up. All confusion is removed, and the determination is made much more rapidly than by tracing out a curve on a chart or figuring from a formula.

The construction of the instrument is very simple. The values of the required angle were figured from the various dips and angles with the section and were tabulated. The sheet of figures thus obtained was mounted on a cylinder, and set in the box so that it could be easily rotated by one of the knobs. A cylindrical shell in which certain openings had been cut was placed over the first one, and its rotation about the common axis is controlled by the other knob. The indices were then adjusted so that the proper value appears for any given set of angles.

The small dimensions of the device, the ease of making the determinations, and the absence of chance for error make this method ideal.

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## Melting of Cathode Copper in the Electric Furnace

Discussion of the paper of DORSEY A. LYON and ROBERT M. KEENEY, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 92, August, 1914, pp. 1791 to 1800.

LAWRENCE ADDICKS, Chrome, N. J.—I read this paper for the first time this afternoon, and it is really worthy of a much more careful reply than I can make with such a short time to consider it, but I just want to say that I differ with Dr. Lyon in his premises and conception of what happens in a refining furnace. He cites steel as an instance where the production of special grades justifies the high cost of power in an electric furnace, and goes on to say the same will be true in the case of copper, for the reason that you will get a better quality of product. I don't think you will get a higher grade of copper, or a better conductivity, as I will explain in a moment. The cost of fuel in the Eastern plants is somewhere between 30c. and 40c. per ton of copper refined, and, if my memory serves me correctly, in an open-hearth steel furnace it is something like $1.25, so it is evident that steel could be handled to better advantage than copper on the basis of fuel. Now, taking the Great Falls plant to compare with is not quite fair, because that plant operates under peculiar conditions. The furnaces are relatively small; the coal is very bad and is very high in sulphur. No one would think of using a coal with 3.6 per cent. sulphur were anything better available. Due to these conditions, the amount of slag is inordinate, it being quoted as 3.5 per cent., and it is necessary to use lime, as there is a tendency to have troubles from arsenic owing to the very high current density used. I think he has taken about as hard a case as he could pick out. Now as to the conductivity of copper, my own idea about the reason that melted copper does not give as high a conductivity as pieces drawn directly from a cathode, or native copper, is that in the latter case the impurities are present as a mechanical mixture, while after melting they are chemically combined with the copper, which is a very different condition. Further, it is necessary to take the sample from a relatively smooth and consequently pure part of the cathode, as otherwise it would not draw. The same with the mass of copper; you pick out a good solid part of the copper, which is not representative. In fact, the refiners have all found they could not take fair samples of cathodes for a chemical analysis. It is always customary to use wire-bar copper for the purpose of sampling, and the impurities are several times as great as shown by samples taken from the cathodes. I think if we examine all of the examples he has given of cases where copper is higher in conductivity, it will fit with that explanation. For example, the Elmore process is a direct electrolytic deposit. Incidentally, at the present time there are a number of ex-

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periments going on on the question of electrolytic wire which look quite promising.

About the oxygen content of copper, copper dissolves gases the same as other metals when it is molten. Any of you who have worked with silver know how it spits as it cools, every one knows how iron "pipes," and copper does exactly the same thing. I am not sure it is oxygen it absorbs; it may just as well be CO or $CO_2$; but there is no difficulty in getting rid of the oxygen in the present furnace. The only difficulty is than when the oxygen is poled out the copper "spews." I have taken copper wire bars with practically no oxygen, which were in horrible shape physically, and they rolled perfectly as far as conductivity goes, and there was no trouble at all with the copper, so that if we melted copper in a neutral atmosphere it does not follow that we would be able to do without the cuprous oxide and get a perfect casting.

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## Basic-Lined Converter Practice at the Old Dominion Plant

Discussion of the paper of L. O. HOWARD, presented at the Salt Lake meeting, August, 1914, and printed in *Bulletin* No. 91, July, 1914, pp. 1785 to 1790.

E. P. MATHEWSON, Anaconda, Mont.—I would like to add a few words of praise on Mr. Howard's work at the Old Dominion. Mr. Howard has given this particular converter shell his personal attention, and the results are shown in the figures given. Similar converter shells, a little different in shape and of about the same size, have been tried in many plants throughout the United States. The results have been highly satisfactory, but there is no shell of the size of the one described by Mr. Howard that has given anything like the results that Mr. Howard's shell has given. I know of larger shells that have given very good service, and are still in operation, and require very little repair. The main thing about the whole process of using the basic lining in copper converting is to keep the temperature within reasonable bounds, particularly not to get the furnace too hot. If it is overheated a little bit, for say half an hour, an entire lining may be ruined. It is very easy to keep the temperature regulated if the mass is large. If you have a large mass in your converter you can easily prevent it from getting overheated, and that is the big argument in favor of the extra large converters. Of course, there is a limit to what a plant can turn out in the way of matte. Very few plants can supply matte sufficient to keep a converter of 20 ft. in diameter in operation constantly, but the 20-ft. converter is very much superior to the 12-ft. converter in easy handling and absolute control of the temperature. I think Mr. Howard is to be congratulated on the excellent showing he has made.

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# WOODBURY JIGS

## The Science of Ore Dressing

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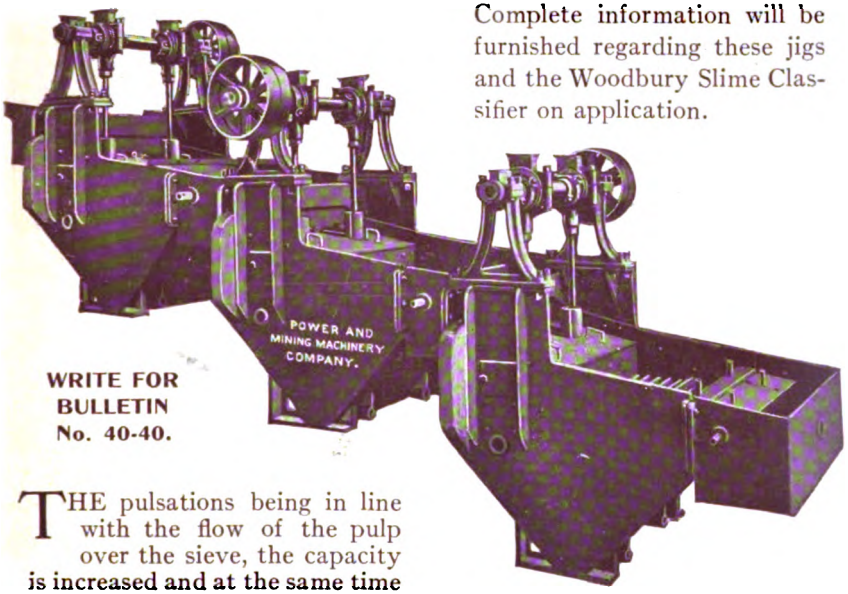


WRITE FOR BULLETIN No. 40-40.

THE pulsations being in line with the flow of the pulp over the sieve, the capacity is increased and at the same time a complete stratification takes place near the head of the screen. This allows the discharge points to operate on layers which contain their proper grains only. Hence the products.

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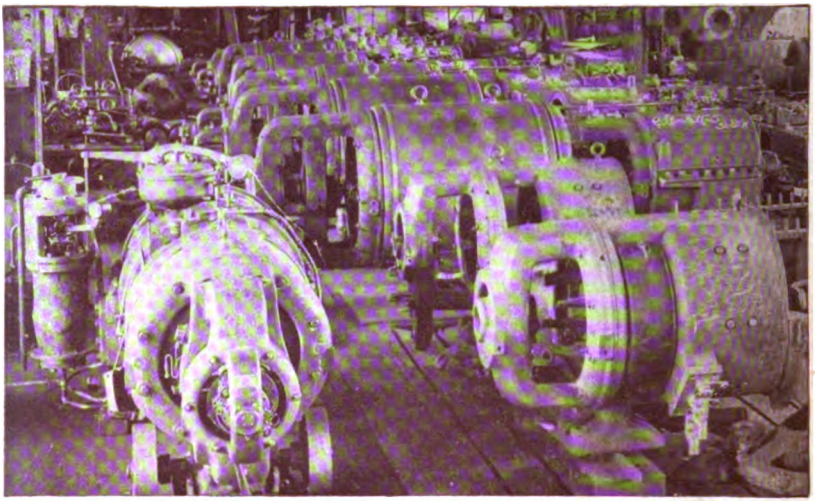

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SEPTEMBER, 1914

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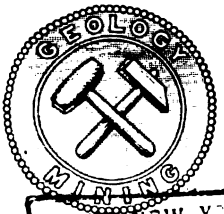


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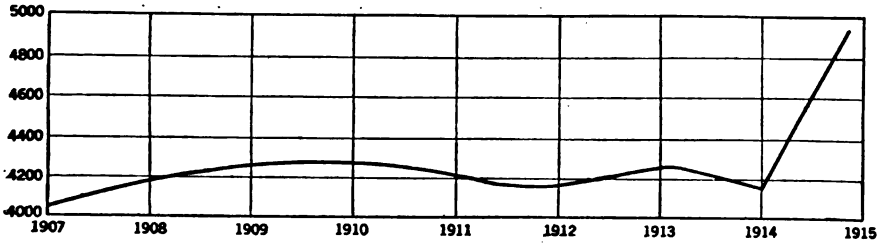
# *NEARING THE 5,000 MARK*

The diagram below shows the rate of growth of the American Institute of Mining Engineers during the past seven years. The Institute now has a greater number of members than ever before in its history, and we are now close to the 5,000 mark. Before the annual meeting in




February we want to cross it. The officers and committee members have been laboring (without pay, except the satisfaction of doing a worthy task) to make the Institute stronger, better, and more valuable to each member. Why not show your appreciation of this by inducing some amply qualified engineer to join?

*Committee on Increase of Membership,*
ADOLPHE E. BORIE, *Chairman,*
THOMAS T. READ, *Secretary.*

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AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

# PROPOSAL FOR MEMBERSHIP

*Mr.* _____
(Name in Full)

*Occupation* _____

*Address* _____

*is hereby proposed by the undersigned, as a* _____

_____

*of the American Institute of Mining Engineers.*

_____
_____
_____
} Signatures of three Members or Associates.

*Place of birth* _____ *Year of birth* _____

*Education, general and technical, when, where and how acquired, with degrees, if any.*

| *Dates* | |
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*Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)*

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*Present position*_____

_____

_____ *Signature.*

*Dated*_____ *191*

## EXTRACTS FROM THE CONSTITUTION.

### ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

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# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 96 DECEMBER 1914

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. Wisotzkey, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., Bradley Stoughton, Editor, F. O. Pierce, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), $10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, $5 per annum. Single copies (including postage), $1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

## DUES OF THE YEAR 1915

In accordance with the Constitution of the Institute, notice is hereby given that membership dues for the year 1915 will be payable on Jan. 1, 1915. The Constitution is mandatory upon officers and members of the Institute alike, requiring that, if members do not pay their dues within four months after the time when they are payable, the *Bulletin* can no longer be sent to them.

## BACK BULLETINS WANTED

The stock of *Bulletins* listed below is almost exhausted in the Institute's reserve supply and the office of the Institute will, therefore, pay the sum of 50 cents each for any of these *Bulletins* in good condition:

| No. | | | No. | | |
|---|---|---|---|---|---|
| 7 | 1906 | January | 13 | 1907 | January |
| 8 | 1906 | March | 51 | 1911 | March |
| 9 | 1906 | May | 86 | 1914 | February |
| 10 | 1906 | July | 87 | 1914 | March |



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## BOARD OF DIRECTORS

*Meeting of Oct.* 23, 1914.—It was voted to take no action concerning the proposed amendments to Article VIII, Sections 2 and 3, but that they be sent to the membership, as provided by the Constitution, together with a statement of the arguments for and against the proposed changes.

It was voted that the following proposed amendment, which had been unanimously approved at the annual meeting of the Institute on Feb. 18, 1914, be sent to the membership as provided by the Constitution, together with the approval of the Board of Directors of this change:

Article III, Section 1, paragraph 3, sentences 2 and 3: substitute the following:

"Any Member, Junior Member or Associate in arrears for two years shall be dropped from the rolls unless the time for payment is extended by the Directors. A Member so dropped may be reinstated by the Directors on paying the arrears of dues and other indebtedness with which he was charged on the books of the Institute at the time he was dropped. A Member or Associate thus reinstated shall receive the *Transactions* for the back year for which he pays."

It was unanimously voted to extend the sincere thanks of the Institute to Samuel A. Taylor, Chairman, and to each member of the several Committees of the Pittsburgh meeting, for the enjoyable and successful meeting which they brought about.

It was also resolved that the Board of Directors extend the sincere thanks of the Institute for courtesies extended to its members and guests on Oct. 8, 1914, to the following: The Carnegie Institute, University of Pittsburgh, Carnegie Institute of Technology, United States Bureau of Mines, Carnegie Steel Co., National Tube Co., and the Harbison-Walker Refractories Co.

The Nominating Committee presented its report of nominations for officers of the Institute to be voted on at the Annual Meeting in 1915.

The Executive Committee reported that it had unanimously decided that the Institute should continue to give its support to the International Engineering Congress.

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## TRANSACTIONS—VOLUMES XLVI, XLVII, AND XLVIII

Since the amount of publications by the Institute during the year 1914 is so large as to demand, for the first time, the issuance of three volumes in any one year, it proved impossible to publish all of these three volumes, namely, Volumes XLV, XLVI, and XLVII, previous to the autumn. Volume XLV was issued in May. Volumes XLVI and XLVII are entirely completed except for the printing of a few signatures and the binding. It is hoped that they will be delivered before the close of the year, or, in any event, early next year. The first volume of the year 1915, Volume XLVIII, is also nearly completed, and it is hoped that it will be in the hands of members before the February meeting.

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## PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Oct. 10 to Nov. 10, 1914:

**Jintaro Kojima**, Ashio, Japan.
**J. Homer Stover**, Trenton, N. J.
**Kirby Thomas**, New York, N. Y.
**Keijiro Nishio**, Tokyo, Japan.
**Alexander K. Hamilton**, Chicago, Ill.
**Harold N. Lawrie**, Portland, Ore.
**C. Spence Thomas**, Cardiff, Wales.
**Henry Tackmann**, New York, N. Y.
**E. R. Hinckley**, Seoul, Korea.
**S. A. Taylor**, Pittsburgh, Pa.
**Lawrence Addicks**, Douglas, Ariz.
**Noel Cunningham**, Bethlehem, Pa.
**E. C. Vigeon**, London, England.
**S. E. Fairchild, Jr.**, Philadelphia, Pa.
**W. C. Schmidt, Jr.**, New York, N. Y.
**Kenneth C. Browne**, New York, N. Y.

**Dr. David T. Day** has resigned his position with the U. S. Geological Survey and opened an office at 1331 F Street, N. W., Washington, D. C., for practice as a consulting engineer.

**C. H. Strand**, formerly with the Pennsylvania Railroad, at Altoona, Pa., has accepted a position as metallurgist for the Crown Cork & Seal Co., Baltimore, Md.

**L. Selmi** has accepted the position of chief chemist with the Otis Steel Co., Cleveland, Ohio.

**Lloyd T. Buell and Eleanor Latham Newcomb** were married at New York on Oct. 14, 1914.

**Richard Blackstone**, formerly chief engineer of the Homestake Mining Co., has succeeded the late Thomas J. Grier in the management of this company.

**A. C. Bailey** has been appointed manager of the Porcupine Pet Mine, in Dolora township, Ontario.

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## AFFILIATED STUDENT SOCIETY NOTES

At a recent meeting of the Mining Society of Sheffield Scientific School, Yale University, the following officers were elected for the year 1914–15: President, William Emery, Jr.; Secretary-Treasurer, Walter C. Schmidt.

Sanford L. Willis has been elected President, and Kenneth M. Sully Secretary, of the Mining Engineering Society of the Massachusetts Institute of Technology.

The Kentucky Mining Society of the University of Kentucky held the first monthly meeting of the new college year on Oct. 6. Officers were elected as follows: President, William H. Noel; Vice-President, S. J. Caudill; Secretary and Treasurer, W. C. Eyl. Thomas Robinson, class of '14, now mine foreman for the Darby Coal Co., gave an interesting account of his recent experiences in mining. S. J. Caudill described experiences in surveying at Hazard, Ky., and H. C. Thompson spoke on experiences in mining at Sternes, Ky.

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## ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Technical graduate, aged 31, with experience as surveyor, chemist with a sulphur company, practical miner, and as chief engineer of an iron mining company, desires a position as mining engineer or superintendent. Speaks French and German. No. 165.

Member, aged 31, married, graduate Yale and Massachusetts Institute of Technology, six years' experience in coal and iron mining, from engineer to general superintendent, desires position with large coal or iron company. Will go anywhere. References. No. 167.

Member, technical graduate, with experience as machinist, chemist, construction engineer, and manager of mines in the United States and Mexico, desires position as mine superintendent or manager. Speaks Spanish. No. 168.

Member, technical graduate, aged 26, with experience as engineer, mine foreman, assistant superintendent for a pyrite mining company. Experience includes diamond drilling, shaft sinking, underground work, milling of pyrites, and mine accounting. No. 169.

Member, experienced as mine superintendent, desires position. Twenty years' experience in gold mines and mills. Competent to construct mine and mill structures. No. 170.

Member, technical graduate, aged 44, with extensive experience in mining and milling of gold, silver, copper, and talc, and in construction work, desires position as superintendent of mine or mill, or as field engineer. Speaks Spanish, French, and German. No. 171.

---

## BOSTON LOCAL SECTION

*Executive Committee*

HENRY L. SMYTH, *Chairman*

ALFRED C. LANE, *Vice-Chairman*

AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.

ROBERT H. RICHARDS ALBERT SAUVEUR

Meets on the first Monday evening of each winter month

### *Twenty-first Meeting*

The Twenty-first meeting of the Boston Local Section of the Institute was held at the Engineers' Club, 2 Commonwealth Avenue, Boston, Monday evening, Oct. 5, 1914. Twenty members and two guests were present.

The Chairman, Henry L. Smyth, presided at the dinner, at the close of which he introduced, as the speaker of the evening, John M. Longyear, who gave a most interesting talk about his work in exploring for coal on the islands of Spitzbergen, 400 miles north of the North Cape of Norway. The talk was illustrated with many photographs, which showed clearly the nature of the work and the extraordinary difficulties of the location.

AUGUSTUS H. EUSTIS, *Secretary.*

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## NEW YORK LOCAL SECTION

*Executive Committee*

LEWIS W. FRANCIS, *Chairman*

WILLARD S. MORSE, *Vice-Chairman*

THOMAS T. READ, *Secretary*, Woolworth Bldg., New York, N. Y.

PHILIP A. MOSMAN, *Treasurer*

LOUIS D. HUNTOON WILLIAM A. POMEROY

The November meeting of the New York Local Section was held on Nov. 4, at the Kaiserhof, following a dinner at which 60 members were present.

Dr. James Douglas related interesting incidents in the history of American mining which have been of the greatest influence in shaping progress. E. Gybbon Spilsbury spoke of the Phœnixville furnace; Arthur L. Walker spoke of the early smelting practice at Globe; E. Cappellen Smith discussed operating conditions at Mansfeld; W. H. Aldridge referred to the probable far-reaching effects of flotation processes; and H. W. Hardinge described how copper smelting in Colorado took its rise from a casual suggestion made by Dr. Douglas—to all of which Dr. Douglas added interesting additional material. D. H. Browne told an amusing story of early days at Sudbury and Denis M. Riordan described his experiences in Europe in connection with the relief ship *Tennessee*.

The next meeting, to be devoted to the discussion of leaching problems, will be held during the first week in December.

THOMAS T. READ, *Secretary*.

---

## SAN FRANCISCO LOCAL SECTION

*Executive Committee*

S. B. CHRISTY, *Chairman*

H. C. HOOVER, *Vice-Chairman*

ABBOT A. HANKS, *Secretary-Treasurer*, 630 Sacramento St., San Francisco, Cal.

F. W. BRADLEY C. W. MERRILL

The fifth meeting of the San Francisco Local Section of the A.I.M.E. was held Oct. 12, in the rooms of the Engineers' Club, Hotel Sutter, and was preceded by the usual informal dinner. Prof. S. B. Christy presided and 28 members were present.

E. Stedler, Junior Mining Engineer with the Bureau of Mines and connected with Mine Rescue Car No. 5, gave an interesting account of the Draeger oxygen apparatus, illustrating with charts and diagrams the method employed for purifying and cooling the air.

James A. Hyde read a paper on The History of Flotation Concentration Processes. Mr. Hyde illustrated his most interesting paper with a number of experiments. At the close of the paper there was a discussion in which a number of members participated.

J. C. Ray, of Palo Alto, will address the November meeting on The Secondary Enrichment of Ores.

ABBOT A. HANKS, *Secretary*.

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## COLUMBIA LOCAL SECTION

*Executive Committee*

F. A. THOMSON, *Chairman*

GEORGE W. RODDEWIG, *Vice-Chairman*

L. K. ARMSTRONG, *Sec.-Treasurer*, P. O. Drawer 2154, Spokane, Wash.

R. S. McCAFFERY S. H. RICHARDSON

The Annual meeting of the Columbia Local Section, A.I.M.E., was held in the Moorish Room, Spokane Hotel, Spokane, Nov. 20, following a dinner.

After the business session, for the presentation of annual reports and the election of officers and new members, Prof. F. M. Handy presented a paper entitled Notes on the Economic Geology of Eastern Washington, which was followed by a discussion and short talks for the good of the Section.

L. K. ARMSTRONG, *Secretary.*

---

## MONTANA LOCAL SECTION

*Executive Committee*

E. P. MATHEWSON, *Chairman*

FRANK M. SMITH, *Vice-Chairman*

D. C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.

JAMES L. BRUCE OSCAR ROHN

The Semi-annual meeting of the Montana Local Section, American Institute of Mining Engineers, was held at the Silver Bow Club, Butte, Mont., Oct. 16, and was preceded by an informal subscription dinner. The meeting was attended by 57 members and guests. Chairman Mathewson called the technical session to order.

L. V. Bender presented a paper on Coal-Dust Fired Reverberatories at Anaconda, illustrated by large wall diagrams. Discussion was participated in by Messrs. E. P. Mathewson, W. D. Leonard, Frederick Laist, C. D. Demond, and C. R. Kuzell.

E. D. Gardner read his paper, Mining Claims within the National Forests, which had already been printed by the Institute. It was read at the Section meeting because of its interest to the local mining men. Discussion of Mr. Gardner's paper was participated in by Messrs. D. C. Bard, Samuel Barker, E. P. Mathewson, William Scallon, and District Forester F. A. Silcox, of Missoula.

D. C. BARD, *Secretary.*

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## LIBRARY

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Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

### LIBRARY ACCESSIONS

### PARTIAL LIST CLASSIFIED BY SUBJECTS

### Mining and Metallurgy

AUFBEREITUNG VON ERZEN UND KOHLE. By Frd. Freise. Leipzig, n.d.

COPPER SMELTING INDUSTRIES OF CANADA. Ottawa, 1913.

CLEANING OF BLAST-FURNACE GASES. By F. H. Wagner. New York, 1914.

DARSTELLUNG DES EISENS UND DER EISENFABRIKATE. Ed. 2. By Ed. Japing, Wein, 1913.

DIE KUPPLUNGEN DER WALZWERKE. By F. Peter. Halle a. S., 1914.

DAS KALIBERIEREN DER WALZEN. Pts. 1–4. By Alb. Brovot. Leipzig, n.d.

DAS ROHEISEN UND SEINE DARSTELLUNG DURCH DEN HOCHOFENBETRIEB. By F. Lichte. Leipzig, n.d.

EISENHÜTTE. By Oscar Stillich und H. Steudel. Leipzig, n.d

HINTS ON COAL-MINE VENTILATION. Miners' Circular 16, U. S. Bureau of Mines. Washington, 1914.

LODE MINING IN YUKON; AN INVESTIGATION OF QUARTZ DEPOSITS IN THE KLONDIKE DIVISION. Ottawa, 1914.

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## Geology and Mineral Production

AN ESSAY TOWARD A SYSTEM OF MINERALOGY. By A. F. Cronstedt; translated from the original Swedish by Gustav von Engestrom. Vols. 1–2. London, 1788.

BIBLIOGRAPHY OF NORTH AMERICAN GEOLOGY FOR 1913, WITH SUBJECT INDEX. Bull. 584, U. S. Geological Survey. Washington, 1914.

COLORADO FERBERITE AND THE WOLFRAMITE SERIES. Bull. 583, U. S. Geological Survey. Washington, 1914.

CONGRÈS GÉOLOGIQUE INTERNATIONAL. Compte-Rendu de la XII^e Session, Canada, 1913. Ottawa, 1914. (Gift of Congrès Géologique International.)

GOLD FIELDS OF NOVA SCOTIA. Memoir 20–E. Canada Department of Mines. Ottawa, 1912.

INDIA, DEPARTMENT OF MINES. Report of the Chief Inspector of Mines in India, 1913. Calcutta, 1914.

MAGNETITE OCCURRENCES NEAR CALABOGIE, RENFREW COUNTY, ONTARIO. Ottawa, 1914.

MINERAL INDUSTRY. Vol. xxii, 1913. New York-London, 1914.

[NOTE.—Volume xxii of this useful publication has been compiled under the supervision of a new editor, A. Roush, Assistant Professor of Metallurgy, Lehigh University, and Assistant Secretary of the American Electro-chemical society. Among the 95 contributors to this issue we find a few that had signed articles in the first volume, some who have been regular contributors to the more recent volumes, and many who have contributed for the first time. The volume as a whole, therefore, has the advantages to be gained from experienced writers on the one hand and from the infusion of new blood on the other. The general plan for this volume, with the exception that certain of the less important minerals are grouped under class heads, is the same as for all previous volumes. A cursory examination of the present volume impresses one that each branch of the mineral industry has been adequately treated, with a minimum of needless repetition. In 1893, when the first volume was published, developments in the gold and silver industry were of such commanding importance that the various phases of this branch of the industry were comprehensively covered, including an article on the cyanide process, which, at that time, was struggling for more extensive adoption. Now, the hydro-metallurgy of copper is engaging the thought of many metallurgists and operators, so that the careful review by T. T. Read in the current volume is timely and appropriate.

Professor Roush is to be congratulated upon maintaining the high standard set by his notable predecessors, thereby continuing this work, which has now become an institution, as a first aid to searchers for timely and accurate information on the statistics, technology and trade of the mineral industries of the world.—B.A.R.]

MINERAL RESOURCES OF ALASKA, REPORT ON PROGRESS OF INVESTIGATIONS IN 1913. Bull. 592, U. S. Geological Survey. Washington, 1914.

## Non-Metallic Minerals

OIL PROSPECTING, DRILLING AND EXTRACTION. By F. J. S. Sur. Calgary, 1914. (Gift of Author.)

POTASH SALTS AND OTHER SALINES IN THE GREAT BASIN REGION. By G. J. Young. Bull. 61, U. S. Dept. of Agriculture. Washington, 1914. (Gift of Author.)

SLATE IN THE UNITED STATES. Bull. 586, U. S. Geological Survey. Washington, 1914.

STUDY OF THE OXIDATION OF COAL. Technical Paper 65, U. S. Bureau of Mines. Washington, 1914.

## Chemistry and Physics

ANALYTICAL ESSAYS TOWARD PROMOTING THE CHEMICAL KNOWLEDGE OF MINERAL SUBSTANCES. By M. H. Klaproth. London, 1801.

ELEMENTS OF THE ART OF ASSAYING METALS. By J. A. Cramer. London, 1764.

ENZYKLOPADIE DER TECHNISCHEN CHEMIE. By Fritz Ullmann. Band 1. Berlin, 1914.

HANDBUCH DER MINERALCHEMIE. Vol. 3, pt. 4. By C. Doelter. Dresden, 1914.

HANDBUCH DER PRÄPARATIVEN CHEMIE. II Band; Organischer teil. By Ludwig Vanino. Stuttgart, 1914.

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## General

COLLEGE PHYSIOGRAPHY. By Ralph S. Tarr. Published under the editorial direction of Lawrence Martin. New York. The Macmillan Co., 1914. (Gift of Publishers.) Price $3.50.

[NOTE.—The book was written by the late Professor Tarr, and was based on his lectures on physical geography at Cornell. Professor Martin has edited Professor Tarr's manuscript, written seven chapters to complete the book, selected the illustrations and added copious bibliographical references. The work is extremely well done.—W. P. C.]

DAS VERZINEN, VERZINKEN, VERNICKELN, ETC. By F. Hartmann. Ed. 6. Wien, 1913.

DER ERZ- UND METALLMARKT. By A. Haenig. Stuttgart, 1910.

GRAPHIC METHODS FOR PRESENTING FACTS. By W. C. Brinton. New York, 1914.

HANDBOOK OF TABLES AND FORMULAS FOR ENGINEERS. By C. A. Peirce and W. B. Carver. McGraw-Hill Book Co., New York, 1914.

LACKAWANNA STEEL CO. HANDBOOK, EDITION OF 1915. Lackawanna, N. Y., 1914. (Gift of Company.)

NEW INTERNATIONAL ENCYCLOPÆDIA. Vols. 3–6. New York, 1914.

OUR MINERAL RESERVES. How to make America Industrially Independent. Bull. 599, U. S. Geological Survey. Washington, 1914.

QUARRY ACCIDENTS IN THE UNITED STATES DURING THE CALENDAR YEAR 1913. Technical Paper 92, U. S. Bureau of Mines. Washington, 1914.

## Company Reports

ORIENTAL CONSOLIDATED MINING CO. Report, 1914. New York, 1914. (Gift of Company.)

ELECTROSTATIC SEPARATION. N.p., n.d. (Gift of American Zinc Ore Separating Co.)

PLUMB PNEUMATIC JIG. N.p., n.d. (Gift of American Zinc Ore Separating Co.)

## Trade Catalogues

AMERICAN STEAM PUMP CO., Battle Creek, Mich. Pump test made by Robt. W. Hunt & Co. of Chicago, Ill.

AMERICAN STEEL & WIRE CO., Chicago, Ill. American Wire Rope News. June, 1914.

DETROIT FUSE & MFG. CO., Detroit, Mich. "Detroit" ironclad fused switches and motor starters. "Arkless" indicating enclosed fuses. 28 pp.

GENERAL ELECTRIC CO., Schenectady, N. Y. Bull. No. 48014. Mine Hoist equipment, Sept., 1914.

GENERAL FILTRATION CO., INC., Rochester, N. Y. Filtros, the perfected porous mineral ware. 28 pp.

KEASBEY & MATTISON CO., Ambler, Pa. Comments on the evil wood shingle roof. 15 pp.

LESCHEN, A. & SONS ROPE CO., St. Louis, Mo. Leschen's Hercules. Oct., 1914.

LONGYEAR, E. J. CO., Minneapolis, Minn. Bull. No. 5, "UG" long-stroke screw-feed diamond drill.

PHILADELPHIA STEEL & FORGE CO., New York, N. Y. High Grade Steels. 46 pp.

SPARTA IRON WORKS CO., Sparta, Wis. Sludge Bucket. Nov., 1914.

SUPPLEE-BIDDLE HARDWARE CO., Philadelphia, Pa. Monel Metal. Sept.–Oct., 1914.

WEBSTER MFG. CO., Tiffin, Ohio. Webster Method. Sept., 1914.

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## MEMBERSHIP

### NEW MEMBERS

The following list comprises the names of those persons who became members during the period Oct. 10 to Nov. 10, 1914:

*Members*

ADDAMS, CHARLES EDWARD, Asst. Supt...Ray Consolidated Copper Co., Ray, Ariz.
ALEXANDER, HARRY H., Asst. Mgr. and Mgr., American Smelting & Refining Co., Maurer, N. J.
ALLING, MARK N., Min. Engr.....Downieville, Sierra Co., Cal.
ANDERSON, LESLIE DOUGLASS, Supt.....U. S. Smelting Co., Midvale, Utah.
ATKINS, WILLIAM, JR., Sec'y.....Kingwood Coal Co., Howesville, W. Va.
BLAKE, TRUE WALTER, Min. Engr.....Rombauer Coal Co., Novinger, Mo.
BRACKETT, GEORGE SYLVESTER, Supt...Pittsvein Coal Co., Flemington, W. Va.
BRAINERD, HOWARD STANFORD, Met.....Ingersoll-Rand Co., Phillipsburg, N. J.
BROWN, THOMAS J., Genl. Supt., Nova Scotia Steel & Coal Co., Sydney Mines, N. S., Canada.
BROWNE, KENNETH CAULFIELD, Min. Engr.....1235 Tinton Ave., New York, N. Y.
BRYAN, JOHN K., Genl. Foreman....Raritan Copper Wks., Perth Amboy, N. J.
CADMUS, DANIEL H., Resident Engr.....Central Coal & Coke Co., Huntington, Ark.
CAPLES, JAMES WATTS, Min. and Civ. Engr.....Salmon, Idaho.
CARPENTER, EDWIN L., Pres.....Consolidated Fuel Co., Salt Lake City, Utah.
CARPENTER, HENRY CORT HAROLD, Prof. of Met., Imperial College of Science and Tech. (Royal School of Mines), So. Kensington, London, S. W., England.
CHARBONNIER, JULES, Genl. Mgr., West Canadian Collieries, Ltd., Blairmore, Alta., Canada.
COMSTOCK, ELTING H., Prof.....School of Mines, Univ. of Minn., Minneapolis, Minn.
CORNELL, IRWIN HEWLETT, Sec'y., St. Joseph Lead Co., 61 Broadway, New York, N. Y.
DELANO, LEWIS A., Chemist and Foreman, Flotation Plant, St. Joseph Lead Co., Bonne Terre, Mo.
DUFUR, WILLIAM HARRISON.....Oil City, Pa.
DYE, ROBERT EMMITT, Met...Care Buffalo Mines, Ltd., Cobalt, Ont., Canada.
EAMES, LUTHER B., Supt.....Goldfield Cons. Mill, Goldfield, Nev.
FERGUSON, WILLIAM CURTIS, Genl. Supt., The Calumet Fuel Co., Utah Fuel Co., Denver, Colo.
GARRETT, ARTHUR JOHN, Min. Engr., Lehigh Coal & Navigation Co., Lansford, Pa.
GRAUPNER, MARCELLUS F.....753 Laguna St., San Francisco, Cal.
GREENAN, JAMES OWEN, Min. Engr., Care Nevada Hills Min. Co., Fairview, Nev.
GRETHER, WALTER SCOTT, Min. Engr.....R. F. D. 1, Asbury, Mo.
GRIFFITHS, FRANK I., Asst. Met., Mt. Morgan Gold Mining Co., Mt. Morgan, Queensland, Australia.
HAMILTON, ROBERT, Cons. Engr., Tennessee Coal, Iron & R. R. Co., Birmingham, Ala.
HARDISON, S. J., Supt.....Nevada Petroleum Co., Coalinga, Cal.
HARPER, JOHN BRADSTREET, Supt., Copper Queen Gold Min. Co., Stoddard, Yavapai Co., Ariz.
HASE, HERMAN CARL, Millman, Care Inspiration Cons. Copper Min. Co., Miami, Ariz.
HASTINGS, VERNON P., JR., Smelter Supt.,.....P. O. Box 477, Morenci, Ariz.
HASTINGS, WARREN, Mine Foreman.....New Jersey Zinc Co., Ogdensburg, N. J.
HAYES, JAMES EDWARD, JR., Vice-Pres. and Genl. Mgr., New Jersey Zinc Co., 55 Wall St., New York, N. Y.
HENRY, JOHN THOMPSON, Min. Engr.....Martha Furnace, Center Co., Pa.
HESKETT, JOHN A., Met.....44 Mayfield St., Coburg, Vic., Australia.
HILLEKE, AUGUST F., Dist. Supt.....Semet Solvay Co., Ensley, Ala.
HOLT, JACOB M., Supt.....Colonial Collieries Co., Natalie, Pa.
HUTCHINSON, B. E., Vice-Pres.....Blair Engineering Co., Chicago, Ill.
HUTCHISON, WILLIAM, Chief Engr., Hillcrest Collieries, Ltd., Hillcrest Mines, Alba., Canada.
HUTTELMAIER, GUSTAVE E., Engr.....H. C. Frick Coke Co., Scottdale, Pa.
JULIAN, ESTEY A., Genl. Mgr.....Nevada Hills Mining Co., Fairview, Nev.
KENDALL, MESSMORE, Lawyer.....233 Broadway, New York, N. Y.
KERR, WALTER, Supt., St. Louis, Rocky Mountain & Pacific Co., Koehler, N. M.

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KINGDON, GEORGE, Supt.....Cananea Cons. Copper Co., Cananea, Son., Mexico.
LIST, ELMER, Engr.....Granby Mining & Smelting Co., Neodesha, Kan.
LUTHER, ROLAND Y., Genl. Mgr.....Peerless Coal & Coke Co., Vivian, W. Va.
MIZEN, BEN ALFRED, Min. Engr.....Crosby, Minn.
MOODY, NELSON KINGSLAND, Vice-Pres., Prairie Oil & Gas Co., Independence, Kan.
NICOL, JAMES, JR., Genl. Mgr.....Galloway Coal Co., Carbon Hill, Ala.
NORTON, WADSWORTH W., Supt., Murray Plant, Amer. Smelt. & Ref. Co., Murray, Utah.
O'BRIEN, THOMAS S., Supt...Original Amador Cons. Mines Co., Amador City, Cal.
OHNSORG, NORMAN L., Asst. Supt.....Phosphate Mining Co., Nichols, Fla.
PARKER, J. HEBER, Met.....Carpenter Steel Co., Reading, Pa.
PAYNE, CALVIN N.....Titusville, Pa.
PIEZ, CHARLES, Pres.....Link Belt Co., 39th St. and Stewart Ave., Chicago, Ill.
POWELL, FREDERICK J., Lawyer, Mining .....52 William St., New York, N. Y.
RADCLIFFE, DONALD HEWSON, Instructor in Mineralogy, Missouri School of Mines, Rolla, Mo.
REID, JOHN CALLUM, Genl. Mgr., Chinook Coal Co., Ltd., Lethbridge, Alta., Canada.
RHOADS, EUGENE A., Contractor.....Rhoads Contracting Co., Ashland, Pa.
RINGLUND, SOREN, Min. Engr.....Empire Zinc Co., Silver City, N. M.
ROGERS, BENJAMIN C., Engr.....Ray Cons. Copper Co., Ray, Ariz.
ROTH, ROBERT F., Asst. Chief Engr., E. E. White Coal Co., Glen White, Raleigh Co., W. Va.
ROUSH, GAR A., Asst. Prof. Met.....Lehigh Univ., So. Bethlehem, Pa.
RUILOBA, JOSE A., Chairman, Zinc Com., U. S. Steel Corpn., 71 Broadway, New York, N. Y.
SALISBURY, GEORGE W., Mine Inspector, United Verde Copper Co., Jerome, Ariz.
SCARFE, GEORGE O., Supt., Middle Yuba Power Co., Alleghany, Sierra Co., Cal.
SCHULTZ, ROY WILSON, Mill Supt., Mond Nickel Co., Ltd., Coniston, Ont., Canada.
SEDERHOLM, CHARLES AUGUSTUS, Supt., Anaconda Copper Mining Co., Coal Dept., Sand Coulee, Mont.
SHILLING, JACOB DAVID, JR., Asst. Supt. of Mines, Utah Copper Co., Bingham Canyon, Utah.
SMITH, CECIL WELDEN, Min. Engr.....Nokomis Coal Co., Nokomis, Ill.
SMITH, JAY MARK, Sales Engr.....Ingersoll-Rand Co., Hartford, Conn.
SMITH, JESSE M., Consulting Engr.....32 West 40th St., New York, N. Y.
SOPER, RALPH H., Geol.....Care E. A. Soper, 1401 Cowper St., Palo Alto, Cal.
SPERR, J. DANA, Min. Engr.....Tom Reed Gold Mines Co., Oatman, Ariz.
STALDER, WALTER, Oil Geol.....1022 Crocker Bldg., San Francisco, Cal.
STRAND, KARL AKSEL, Supt.....Empire Zinc Co., Hanover, N. M.
THORNHILL, E. BRYANT, Supt.....Buffalo Mines, Cobalt, Ont., Canada.
TINGLEY, TRACY W., Asst. Supt., Coal Washery Plant, Cambria Steel Co., Johnstown, Pa.
TSCHUDY, FREDERICK, Cons. Engr...Tenn. Coal, Iron & R. R. Co., Ensley, Ala.
UMPLEBY, JOSEPH B., Geol.....U. S. Geological Survey, Washington, D. C.
WALTER, RAYMOND A., Chief Engr., Md. Div., Consolidation Coal Co., Frostburg, Md.
WARNER, CHARLES M., Field Engr., Dwight & Lloyd Smelting Co., 29 Broadway, New York, N. Y.
WATSON, GEORGE THOMAS, Vice-Pres., Consolidation Coal Co., Fairmont, W. Va.
WEHR, ARTHUR J., Engr.....Ramsey Co., 1052 Fairmont Ave., St. Paul, Minn.
WEISENBURG, ANDREW.....2030 Chestnut St., Philadelphia, Pa.
WOODSON, EDWARD FRANCIS, Min. Engr., Central Coal & Coke Co., Rock Springs, Wyo.
WRAMPELMEIER, ERNEST LEON SWIFT, Min. Engr., 565 Monadnock Bldg., San Francisco, Cal.
WRIGHT, CLARK W., Min. Engr., Care Little River Drainage District, Cape Girardeau, Mo.
WRIGHT, IRA L., Min. Engr.....Pinos Altos, N. M.
YOUNG, ALFALES BURGESS, Chief Testing Engr., Tooele Plant, International Smelting Co., Tooele, Utah.
YOUNG, FRED WILLARD, Min. Engr., General Delivery, Cobalt, Ont., Canada.

## *Associate Members*

COLQUHOUN, WILLIAM WALLACE, Met. Engr., 31 Ferndale, Tunbridge Wells, England.
JONES, ARTHUR LUCAS, Dist. Engr.....General Electric Co., Denver, Colo.

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LEWISOHN, ADOLPH, Pres., Miami Copper Co., 61 Broadway, New York, N. Y.
NOTMAN, GEORGE, Secy. and Treas., Phelps, Dodge & Co., 99 John St., New York, N. Y.
WRIGHT, SPENCER DISSTON, Mfr.....1017 N. Front St., Philadelphia, Pa.

### *Junior Members*

CRAMPTON, THEODORE H. M.....Care "Mt. Chief," Placerville, Idaho.
FOX, WALTER V., Min. Engr.....Copper Giant Mine, Hackberry, Ariz.
PRYER, WILLIAM CRISTY, Student.....Trojan, S. D.
SEN, JANSHI, Student.....433 Furnald Hall, Columbia Univ., New York, N. Y.

## CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Oct. 10 to Nov. 10, 1914, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

### *Members*

**Rodolphe Louis Agassiz**, Boston, Mass.
Proposed by Robert H. Richards, W. E. C. Eustis, H. L. Smyth.
Born 1871, Cambridge, Mass. 1892, Harvard; A. B. Chairman of Committee on Geology, Mineralogy, and Petrography at Harvard.
Present position: Vice-Pres., Calumet & Hecla Mining Co.

**Charles Philip Boone**, Berkeley, Cal.
Proposed by S. H. Loram, Ross E. Douglass, B. T. Colley.
Born 1883, Berkeley, Cal. 1905, Grad. Univ. of Cal., B. S. 1906–08, Butters Salvador Mines. 1909–10, Empire Mines, Grass Valley, Cal. 1911, John Taylor & Sons, India. 1912–13, Charles Butters & Co., Ltd.
Present position: Charles Butters & Co., Ltd., Valparaiso, Chile.

**Eric Edwin Booth**, Cloncurry, N. Queensland, Australia.
Proposed by W. H. Corbould, T. T. Read, Charles F. Rand.
Born 1881, Moruya, N. S. Wales. 1895–98, Boys' High School, Sydney, N. S. W. 1903–04, Sydney Univ. 1905–07, School of Mines, Ballarat, Vict.; A. S. M. B. 1898–1904, N. S. Wales Government as Surveyor and Engineer, on harbor construction works. 1907–09, Surveyor, Lloyd Copper Co., Burraga, N. S. W. 1910, Constructing Engr., smelting plant, Mt. Elliott Copper Mines, N. Queensland. 1911, Mine Mgr., Cobar Gold Mines, Cobar. 1912–13, Mine Mgr., Lloyd Copper Mine, Burraga, N. S. W.
Present position: Engineering Draftsman, Mt. Elliott Mine, N. Queensland.

**S. H. Brady**, Tonopah, Nev.
Proposed by Horace V. Winchell, Warren V. Richardson, W. H. Wiley.
Born 1874, Detroit, Mich. No technical education; practical experience. 1894–95, Assayer, Last Chance Mine, Bingham, Utah. 1896, Shift Boss, Yellow Jacket Mine, Idaho. 1897, Assayer, Foley Gold Mines, Ontario, Canada. 1898–1907, Underground Supt. and Asst. Supt., Michigan Copper Mining Co., and Adventure Min. Co., Ontanagon county, Mich. 1907–09, Supt., Tonopah Belmont Dev. Co., Tonopah, Nev.
Present position: 1909 to date; Genl. Mgr., West End Cons. Min. Co.; Supt., Halifax Tonopah Mining Co., Tonopah, Nev.

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**George Henry Caperton**, Charleston, W. Va.
Proposed by James S. Cunningham, J. M. Clark, Charles E. Krebs.
Born 1860, Lynchburg, Va. 1874–75, High School, Staunton, Va. 1875–78, Virginia Agricultural and Mech. College. 1884–1906, Genl. Mgr., Fire Creek Coal & Coke Co. 1895–1906, Genl. Mgr., Thrumond Coal Co. 1895–1897, Genl. Mgr., New River Coke Co. 1900–1906, Pres., Wright Coal & Coke Co. 1903–06, Pres. and Mgr., Tinkey Knob Coal Co. 1903–05, Pres. and Mgr., Piney Colliery. 1906–14, Pres., Slab Fork Coal Co. 1906–14, Pres., South Side Co.
Present position: Pres., Scotia Coal & Coke Co., Fayette and Raleigh counties, W. Va.

**Thomas William Dawson**, Scottdale, Pa.
Proposed by Howard N. Eavenson, Edward O'Toole, Samuel A. Taylor.
Born 1876, Helfenstein, Pa. 1892, Mt. Carmel, Pa., High School. 1901, Grad. Ohio Normal Univ., Ada, Ohio, in Min. Engrg.; B. S. 1892–97, Phila. & Reading Coal & Iron Co., Locust Gap, Pa., clerk and later chainman on Engrg. Corps. 1901–02, Draftsman on general work, Westinghouse Electric Co., East Pittsburgh, Pa. 1902–03, Mine Transitman, H. C. Frick Coke Co., Scottdale, Pa. 1903–04, Mine Transitman and Property Draftsman, H. C. Frick Coke Co., Scottdale, Pa. 1904–07, Div. Engr., H. C. Frick Coke Co., Scottdale, Pa.
Present position: 1907 to date, Asst. Chief Engr., H. C. Frick Coke Co.

**James Wilbert Deetrick**, Youngstown, Ohio.
Proposed by John E. Perry, George H. Morse, C. S. Robinson.
Born 1870, Sharpsburg, Pa. Common and high schools, Youngstown, Ohio. 1890, In laboratory of Youngstown Steel Co. 1893, In laboratory of Ohio Steel Co. 1895, Supt. Blast Furnace, Mahoning Valley Iron Co. 1899, Supt. Furnaces, Republic Iron & Steel Co. Have held various positions, Supt., and Manager of Blast Furnaces, Steel Plants, and Rolling Mills, with this company.
Present position: Genl. Mgr., Operating Departments, Republic Iron & Steel Co.

**Nelson Dickerman**, Barranquilla, Colombia.
Proposed by Newton Cleaveland, E. B. Kimball, W. P. Hammon.
Born 1881, Denver, Colo. 1900–01, Surveying and assaying, Tomboy Gold Mines Co. 1901–02, University of Cal. 1902, Erection and running Rawhide Cyanide Plant. 1902, University of Cal. 1903, Mine sampling and testing, Spokane, Wash. 1903, Bunker Hill and Sullivan Mine. 1903–04, Met., Lead Metals Co. 1904–05, Grad., Univ. of Cal.; B. S. 1905–10, Dredging, Yuba Cons. Goldfields. 1910–11, Asst. to Genl. Mgr., Cleaveland-Natomas Cons. 1911–13, Kutley Creek Gold Dredging Co., Salmon, Ida.
Present position: Genl. Mgr., Pato Mines, Ltd., Colombia.

**Ralph Dougall**, Bankhead, Alta., Canada.
Proposed by Lewis Stockett, Arthur W. Jenks, Frank D. Adams.
Born 1875, Montreal, Can. 1882, Tutors and private school, Montreal. 1883, Preparatory school, Montreal. 1885, Brooklyn Academy, Brooklyn, N. Y. 1887, Whitestone, L. I., School, N. Y. 1892, Flushing, L. I., School, N. Y. 1898, McGill Univ., Montreal; B. A. Sc. 1898, Transitman, running township lines in Rainy River district, Ontario. 1899, Asst. Chemist, Guggenheim Smelter, Aguascalientes, Mexico. 1900–03, Chief Chemist, La Compania Minera de Penoles, Mapimi, Dgo., Mexico.
Present position: 1911 to date, Chief Engr., Bankhead Mines.

**Lee D. Dougan**, Terry, S. D.
Proposed by J. V. N. Dorr, Noel Cunningham, Charles A. Chase.
Born 1884, Plankinton, S. D. 1903, Plankinton High School. 1904–05, Univ. of S. D. 1906–07, Univ. of Washington; B. S. 1907, Homestake Slime Plant, Deadwood, S. D. 1906, summer, Granby Mining & Smelting Co., Phœnix, B. C.
Present position: 1907 to date, Asst. Supt., Mogul Mining Co., Plume, S. D., and Terry, S. D.

**George Albert Easley**, La Paz, Bolivia.
Proposed by P. A. Seibert, G. H. Cox, Francis Church Lincoln.
Born 1885, Harwood, Mo. 1899–1901, Nevada, Mo., High School. 1901–03, Warrensburg, Mo., State Normal. 1903–09, Missouri School of Mines, Rolla, Mo.; B. S. 1912, M. E. 1906–07, Transitman, on location in Sonora, Mexico, E. P. & S. W. Ry., El Paso, Texas. 1909–10, Supt., Pan American Tin Co., La Paz, Bolivia. 1910–11, Supt. 1911–12, Mgr., Olla de Ore Gold Mining Co., La Paz, Bolivia. 1912–13, Reported on mines in Spanish Honduras for Olla de Ore.
Present position: Mgr., Olla de Ore Gold Mines, Ltd., La Paz, Bolivia.



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**Charles Lewis Taylor Edwards,** Bethlehem, Pa.
Proposed by W. R. Shimer, E. O. C. Acker, W. L. Cumings.
Born 1889, Duquesne, Pa. 1907–08, Carnegie Tech. Night School. 1909–13, Lehigh Univ., Met. 1905–06, Bessemer Steel Dept., National Tube, McKeesport. 1906–07, National Rolling Mills, McKeesport. 1907–08, Eng. Dept. and Rail Inspector, Tenn. Coal, Iron & R. R., Ensley, Ala. 1908–09, Construction, order clerk, National Tube, McKeesport. Porter Locomotive Wks., Pittsburgh, Pa. 1914, Open Hearth helper, So. Bethlehem, Pa. Special Steel Inspector.
Present position: Asst. to Supt. Blast Furnaces, Bethlehem Steel Works, S. Bethlehem, Pa.

**Roy Feldenheimer,** Portland, Ore.
Proposed by O. N. Friendly, James Humes, George S. Krueger.
Born 1889, Portland, Ore. 1906, Portland Academy. 1906–08, Univ. of Cal. 1909–10, Europe. 1910–12, Univ. of Cal.; B. S. in Mineralogy. 1907, summer, Survey with Spring Valley Water Co., S. F., Cal. 1908, summer, underground North Pole Min. Co. 1909, Assaying, sampling ore, Idaho Invest. Co. 1909, Topman, Humboldt Gold Min. Co. 1912, six months, Gold Bullion Assayer, Selby Smelter. 1912–13, four months, Keystone Drill & Oiler, Natomas Cons. 1913, six months, Roaster Engineering Dept., Steptoe Valley Smelter, clerk and draftsman. 1913, Member Canadian Inst. Min. Engrs.
Present position: Unemployed.

**James Hyde Forbes,** Stanford Univ., Cal.
Proposed by James C. Ray, C. F. Tolman, Jr., G. H. Clevenger.
Born 1889, Chicago, Ill. 1908, Los Angeles High School. 1913, Leland Stanford, Jr., Univ. 1908–09, Yellow Aster Mine, Randsburg, Cal. 1909, Gold Midas Min. Co., Knob, Cal.
Present position: 1913 to date, Geol., and Hydrographical Engr., Spring Valley Water Co.

**Alfred Karl Friedrich,** Cleveland, Ohio.
Proposed by G. A. Reinhardt, Charles H. Fulton, A. W. Smith.
Born 1883, Meadville, Pa. 1904, Case School of Applied Science; B. S. 1908, Harvard Univ.; M. E. 1904, Rodman, C. & N. W. Railroad, Boulder, Colo. 1905, Transitman, on Coal Mine. Surveyor for C. W. Courtney, Cleveland, Ohio. 1905–06, Engineer's Office, City of Cleveland. 1906–07, Chemist, Union Sulphur Co., Sulphur, La.
Present position: 1908 to date, Engr., Rogers-Brown Ore Co., Crosby, Minn.

**Ernest Gheur,** Red Deer, Alta., Canada.
Proposed by Lewis Stockett, W. F. McNeill, R. H. Morris.
Born 1878, Liége, Belgium. 1901, Grad. as Civil and Ming. Engr., Univ. of Liége, Belgium. 1901–09, Supt., Charbonnayes des Kessales, Gemmeppe-sur-meuse. 1910–12, Chief Engr., Acadia Coal Co., Stellarton, N. S., Canada.
Present position: 1912 to date, Consulting Engr., Brazeau Collieries, Ltd., Toronto.

**Charles Newton Gould,** Oklahoma City, Okla.
Proposed by Anthony F. Lucas, David T. Day, David White.
Born 1868, Lower Salem, Ohio. 1892–99, Southwest Kansas College; B. S. 1899–1900, Univ. of Nebraska, Geology; A. M. 1900–03, Instructor in Geology, Univ. of Okla. 1903–08, Prof. of Geology, Univ. of Okla. 1906, Ph. D., Univ. of Nebr. 1908–11, Director, Oklahoma Geological Survey. 1903–06, Resident Hydrographer U. S. Geological Survey. 1904, Member, Amer. Assn. Ad. Science. 1905, Fellow, Geological Society of America. 1913, Member, International Geological Congress. 1914, Member, Institution Petroleum Technologiste.
Present position: 1911 to date, since resigning directorship of Oklahoma Geological Survey have been employed by various companies and individuals in reporting on properties, chiefly oil and gas.

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**Stephen Harris,** Friesland, N. Queensland, Australia.
Proposed by W. H. Corbould, T. T. Read, Charles F. Rand.
Born 1863, Victoria. 1877–81, Wesley College, Melbourne, Victoria. 1883–85, School of Mines, Ballarat, Vict. Three years course, studying geology, mineralogy, assaying, surveying, metallurgy, mechanical engineering and general mining. 1886, Lead refining, Sunny Corner Silver Mining Co., N. S. W. 1886, Assaying, Pinnacles Silver Mining Co., Broken Hill, N. Z. 1886–87, Assaying New Zealand Smelting Co., Thames, N. S. W. 1887–89, Assaying and Met., Cordillera & Peelwood S. M. Co., Peelwood, N. S. W. 1889–95, Broken Hill, Block 14, S. M. Co., Broken Hill, Kalgoorlie. 1895, Met., Great Boulder G. M. Co., Kalgoorlie, W. Aust. 1895–1900, Genl. Mgr., Hannans Reward G. M. Co. 1900–04, Unattached. 1905–06, Asst. Mgr., Lloyd Copper Mine, Ltd. Burrago, N. S. W. 1906–07, Genl. Mgr., Queen Bee Copper Mine, Cobar, N. S. W. 1908–10, Unattached. 1910–13, Asst. Genl. Mgr., Mt. Elliott Copper Mines, Selwyn, N. Queensland, Aust.
Present position: Resident Mgr., Hampden Cloncurry Copper Mines.

**John Hartley Hastings,** Bergettstown, Pa.
Proposed by John B. Hastings, Waldemar Lindgren, Ralph Nichols.
Born 1887, Ketchum, Idaho. 1906, Grad., East Denver High School. 1906–07, Denver Univ., chemistry, mathematics, and French. 1908, two months, Student's course of assaying, Stephen Rickard's Denver office. 1908, six months, Study of crystallography and microscopic petrography at home with instructor's aid. 1908, Six months study of general geology and field trips adjacent to Denver. 1907, Asst., mine examination in Mexico with J. B. Hastings. 1907, Chainman, railroad survey, Southern Pacific in Mexico. 1908, Miner, Bisbee, Calumet & Arizona Min. Co. 1908, In charge of own custom assay office at Shoshoni, Wyo. 1909, Millman, Wilfley tables, Steptoe Valley Plant, Nevada Con. Copper Co. 1911–14, Chemist and Assayer, Western Chemical Mfg. Co.
Present position: Chief Chemist, American Zinc and Chem. Co., Langeloth, Pa.

**Roswell Hill Johnson,** Pittsburgh, Pa.
Proposed by Anthony F. Lucas, H. B. Meller, David T. Day.
Born 1877, Buffalo, N. Y. 1891–95, Buffalo High School. 1895–96, Brown Univ. 1896–99, Harvard Univ. 1899–1900, Univ. of Chicago; B. S. 1901–03, Univ. of Missouri; M. S. Summer, 1904, Univ. of Cal. 1907–08, Columbia Univ. 1908–12, Consulting Geol., Bartlesville, Okla., in oil and gas.
Present position: 1912 to date, Prof. of Geology and Oil and Gas Production, Univ. of Pittsburgh.

**Guy Manning Kerr,** Garfield, Utah.
Proposed by P. A. Mosman, A. Eilers, Louis D. Huntoon.
Born 1870, Jackson, Mo. Early education in Colo. Springs school and in Cutler Academy and Colo. College. 1892–96, Univ. of Göttingen, Germany; Ph. D. 1890–91, Assayer, lixiviation plant, San Juan de Guadalupe, Mexico, and later at a mine near Morelia, Mexico. 1897–98, Assayer and Chemist, Old Amber Mine and Mill, near Helena, Mont. 1902, Asst. Chemist, Elers Plant, Pueblo, Colo. 1903, Chief Chem., Pueblo Plant, Pueblo, Colo. 1904–05, Asst. Supt.; 1906, Supt.; Durango Plant, Durango, Colo. 1906–14, Supt., Arkansas Valley Plant, Leadville, Colo.
Present position: Acting as Asst. Supt., temporarily at Garfield Smelting Co

**Sterling Sidney Lanier, Jr.,** Nortonville, Ky.
Proposed by George G. Crawford, Frank H. Crockard, Frank D. Rash, F. G. Morris, E. W. Parker.
Born 1888, Birmingham, Ala. Birmingham Grammar Schools. 1906, Grad., High School, Birmingham, Ala. 1906, Univ. of Ala., special course. 1907, Lehigh Univ., course of Min. Engrg. 1910, Grad., Lehigh Univ.; E. M. 1910, Prospecting Engr. and Chemist, Monro Warrior Coal & Coke Co., Birmingham, Ala. 1911, Coal Salesman, Monro Warrior C. & C. Co. 1911–12, Day laborer, Republic Iron & Steel Co., Republic, Ala. 1912, Mine Inspector, R. I. & S. Co., Republic, Ala., and Sayreton, Ala. 1912, Asst. Mgr., Nortonville Coal & Coke Co., Nortonville, Ky.
Present position: 1912 to date, Genl. Mgr., Nortonville Coal Min. Co., Nortonville, Ky.

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**Herbert Houghton Lauer**, Philadelphia, Pa.
Proposed by E. Gybbon Spilsbury, Howard N. Eavenson, G. Aertsen.
Born 1880, Philadelphia, Pa. 1892–1896, Philadelphia Public Schools. 1896–99, Manual Training High School, Phila. 1899–1906, Lehigh Univ.; E. M. 1901–04, Rodman, transitman, Third Asst. Engr. on construction, Belmont Filters and Reservoir, Phila. 1904–06, Lehigh Univ. 1906–07, Engr. and Assayer, Black Mt. Mining Co., Greene Cananea Copper Co., Cananea, Son., Mexico. 1907–08, Chief Engr., Sewanee Fuel & Iron Co., Tenn. 1908–10, Cyanide shift boss and zinc room foreman, Guanajuato Development, Gto., Mexico. 1910–12, U. S. Steel Co., Engr. in mining and metallurgical work, W. Va., Alabama, and Ill. 1912–13, Instructor, Min. Dept., Univ. of Ill.
Present position: 1913 to date, Engr., Midvale Steel Co., Philadelphia, Pa.

**Henry K. McHarg, Jr.**, Roanoke, Va.
Proposed by Walton Ferguson, Floyd E. Cunningham, Stephen E. Puckette, W. P. Peck, J. A. Van Mater and J. E. Johnson, Jr.
Born 1883, New York, N. Y. General education, public and private schools. 1904–12, Blast Furnace Supt., Genl. Supt. Furnaces, Genl. Supt. of Ore Mines, Virginia Iron, Coal & Coke Co.
Present position: Vice-Pres. and Genl. Mgr., Virginia Iron, Coal & Coke Co.

**John William Maller**, Jalisco, Mexico.
Proposed by Robert Peele, J. F. Kemp, Charles P. Berkey.
Born 1887, Rumania. 1903, Grad,. Public School, New York, N. Y. 1903–06, Academic Dept., College of the City of New York. 1906–10, School of Mines, Columbia Univ.; E. M. 1913–14, Post-graduate student, Columbia Univ., Dept. of Geology. 1910–11, Amalgamated Copper Co., Butte, Mont., miner. 1911–13, Engr., shift boss and foreman, New York and Honduras Rosario Min. Co., San Juanito, Honduras.
Present position: Engr., Public Service Commission (temporary).

**Walther Mathesius**, So. Chicago, Ill.
Proposed by H. A. Brassert, P. A. Newton, A. J. Boynton.
Born 1886, Hoerde, Germany. 1893–1904, Grad. Gymnasium at Dortmund. 1905–10, Grad., Technische Hochschule, Berlin, Charlottenburg, department of chemistry and metallurgy; Diplom-Ingenieur. 1907–11, Received degree of Doctor Ingenieur from Kgl. Technische Hochschule in Berlin, Charlottenburg. 1904, Eisenwerk Kraft, Kratzwieck, Blast furnace and coke ovens. 1904, Eisenwerk Joly, Wittenberg, foundry. 1905, Wilhelmshütte, Enlau, Mech. Dept. 1905, Gutehoffnungshütte, Sterkrade, foundry. 1905, Gutehoffnungshütte, Oberhausser, rolling mills. 1907, Eisenhüttenwerk Thale, Thale a. Harz, open hearth dept. 1909–10, Rombacher Hüttenwerke, Rombach, blast furnace dept. 1910–11, Asst. Prof., Dept. for metallurgy of iron, Kgl. Technische Hochschule, Berlin, Charlottenburg. 1911–12, American Steel & Wire Co., Worcester, Mass.
Present position: 1912 to date, Asst. Supt., Blast Furnaces 1–4 and E., South Works, Illinois Steel Co.

**Ernest Martin Merrill**, Beckley, W. Va.
Proposed by John J. Lincoln, E. E. White, Thomas Nichol.[1]
Born 1878, Newark, Ohio. 1896, Grad., Deane Academy, Granville, Ohio. 1902, Grad., Civil Engineering Dept. of Ohio State Univ., Columbus, Ohio. 1900–02, Rodman, Riggs & Sherman, Municipal Engineers, Toledo, Ohio. 1902–03, Draftsman with locating party on Virginian Ry. 1903–05, Chief Engr. and Genl. Mgr., Micajah Coal & Coke Co.
Present position: 1905 to date, General consulting practice, Beckley, W. Va.

**George Shikataro Nishihara**, Madison, Wis.
Proposed by W. H. Emmons, O. W. Wheelwright, C. K. Leith.
Born 1887, Okayma, Japan. 1912, B. S., Univ. of Ariz. Oct.–Dec., 1912, Scholar in Geology, Univ. of Chicago. 1913–14, Scholar in Geology, Univ. of Minn.; M. S. 1914, Member of Sigma Xi, Member of National Geographic Soc. 1910, Assayer, Cerro Prieto Gold Mining Co., Cerro Prieto, Sonora, Mexico. 1912, Geology of Buckman Canyon, Ariz. 1912, Assayer and Geologist, Gundakei Mining Co., Mohawk Mine, Ariz. 1912, Geology of Montezuma district, Maricopa county, Ariz.
Present position: Field geologist, Wisconsin Geological and Natural History Survey.

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**Robert Scott Ord**, Spokane, Wash.
Proposed by Howard N. Eavenson, John J. Lincoln, George S. Patterson, Henry Brooke.
Born 1872, Edinburgh, Scotland. General education. 1878–81, George Watson's College for Boys, Edinburgh, Scotland. 1881–83, Germantown Academy, Germantown, Pa. 1884–86, Grammar school. 1886–88, Duval High School, Jacksonville, Fla. 1903, Wm. C. Atwater & Co., Ltd., coal selling agency, Bluefield, W. Va. 1903–14, Gen. Mgr., Elkhorn Coal & Coke Co., Maybeury, W. Va.
Present position: Genl. Mgr., Corbin Coal & Coke Co., Spokane, Wash. Mines in B. C., Canada.

**George B. Pickett**, Ophir, Colo.
Proposed by Frank Bulkley, S. A. Ionides, D. G. Miller.
Born 1866, Mt. Pleasant, Iowa. 1880–1885, Student at Colo. College, Colo. Springs, in the preparatory and irregular studies in the regular college classes. 1885, Ore-sorter, Belcher Mine, Silverton, Colo. 1886, North Star on King Solomon Mt., Silverton, Colo.; followed by prospecting, leasing and contracting. 1900, Took management of the Carbonero Mine, Ophir, Colo.
Present position: Vice-Pres. and Mgr. of Ophir Gold Mines, Milling and Power Co.

**Ferdinand Ritter**, Castleton, Vt.
Proposed by J. F. McClelland, L. W. Bahney, Arthur F. Taggart.
Born 1888, Chicopee, Mass. 1908–13, Mining Course, Sheffield Scientific School, Yale Univ.; E. M. 1912, summer, Republic Iron Co., Republic, Mich.
Present position: 1913 to date, Supt., Staso Milling Co., Castleton, Vt.

**Alexander George Russell, Jr.**, Tazewell, Va.
Proposed by Howard N. Eavenson, Karl F. Overholt, J. P. Williams, Jr.
Born 1876, Hamilton, Canada. 1881–91, Public school and business course.
Present position: 1902 to date, Engr. and Agent, Faraday Coal & Coke Co.

**Walter M. Saunders**, Providence, R. I.
Proposed by H. M. Boylston, G. A. Reinhardt, Albert Sauveur.
Born 1866, Johnston, R. I. 1881, Public schools of Johnston. 1881-84, Providence High School. 1884–86, Mass. Inst. of Technology. 1888, Brown University. 1889–96, Instructor in Chemistry, Brown Univ.
Present position: 1896 to date, Member of firm, Saunders & Franklin, analytical and consulting chemists, Providence, R. I.

**Edward A. Sayre**, Des Moines, Ia.
Proposed by S. W. Beyer, Edward E. Bugbee, W. B. Cole.
Born 1879, York, Neb. 1906, Grad. Iowa State College, Ames, Iowa; B. S. 1911, Received degree of E. M. from the same institution.
Present position: 1906 to date, General Mgr., Eagle Coal Co., Des Moines, Iowa.

**Quincy A. Shaw**, Boston, Mass.
Proposed by Robert H. Richards, W. E. C. Eustis, H. L. Smyth.
Born 1869, Boston, Mass. 1891, Harvard; A. B. 1891–93, Calumet & Hecla Min. Co., Calumet, Mich. Officer and Director Ahmeek Min. Co., Allouez Min. Co., Centennial Copper Min. Co., Isle Royale Copper Co., Osceola Cons. Min. Co., Superior Copper Co., Tamarack Min. Co., White Pine Copper Co.
Present position: 1894 to date, Pres., Calumet & Hecla Mining Co.

**Alan O. Sommerville**, Arcadia, Pa.
Proposed by Rembrandt Peale, W. C. Snyder, Hugh Archibald, G. F. Dunkle. H. J. Hinterleitner, Edwin M. Chance, H. P. Dowler, T. H. Watkins.
Born 1869, Centre County, Pa. 1874–86, Genl. education, Bellefonte, Pa., Academy. 1886–87, worked in and about the coal mines at Snow Shoe, Pa., for my father, who was operating mines at that place. 1888–92, worked at Winburne, Pa., for Sommerville and Buchanan in mines and store. 1893–96, Managed store of Miners' Supply Co., Philipsburg, Pa. 1897–99, Operated coal mine at Munson, Pa. 1899–1901, Mine foreman, Sommerville and Buchanan, Winburne, Pa. 1906–08, District Supt., Arcadia district, for Beech Creek Coal & Coke Co. and their successors, the Penn. Coal & Coke Corpn.
Present position: Division Supt. of Arcadia, Patton and Winburne districts of Penn. Coal & Coke Corpn.

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**Thomas R. Stockett,** Nanaimo, B. C., Canada.
Proposed by Eli T. Conner, Lewis A. Riley, Lewis Stockett.
Born 1863, Pennsylvania. General and technical education. Member Canadian Mining Institute since 1902. Member since 1902, and Chairman since 1912, of the B. C. Government Board of Mine Examiners. 1879–86, Locust Mountain Coal & Iron Co. 1886–99, Cons. Coal Co., St. Louis, Mo. 1899–1901, Pacific Coast Co., Seattle, Wash. 1901–04, Crow's Nest Pass Coal Co., B. C., Can.
Present position: 1904 to date, Genl. Mgr., Western Fuel Co., San Francisco.

**Benjiman Kendrick Stroud,** San Francisco, Cal.
Proposed by Thomas Cox, A. K. P. Harmon, Jr., M. L. Requa.
Born 1881, Fresno, Cal. 1905, Univ. of Cal., Mining. 1900–08, Monte Cristo Oil Co., San Francisco, Cal. 1908–09, P. J. Berry, St. Francis Hotel, Cal. 1909–10, Pioneer Midway Oil Co., San Francisco, Cal. 1909–10, St. Lawrence Oil Co., San Francisco, Cal. 1910–13, Recovery and Universal Oil Co., San Francisco, Cal. Supt. of all of the former properties.
Present position: Engr., appraisal board of the Independent Oil Producers Assn.

**Charles Walter Tubby,** St. Paul, Minn.
Proposed by Fred J. Leacey, R. R. Seeber, C. W. Macdougal.
Born 1879, Georgetown, Canada. To 1899, Technical high schools and private instructors. For several years filled in vacations in drafting offices and machine shops. 1899, Surveying, Great Northern Ry., Pacific Coast. 1900, Mech. Draftsman, G. N. Ry. office. 1901, Mech. Engr., St. Paul, Minn. 1901, Chief Mech. Draftsman, G. N. Ry. 1903, St. Paul, Minn. 1903, Chief Clerk to Supt. of Motive Power, G. N. Ry. St. Paul, Minn. 1903, Locomotive Coal Testing, Montana Div., G. N. Ry. 1903, Asst. to Mech. Engr., Great N. Ry., St. Paul, Minn. 1904, Mech. Salesman, Chicago Pneumatic Tool Co., St. Paul, Minn. 1906, Representative, International Steam Pump Co. and several other companies building railway and mining machinery.
Present position: 1913 to date, District Sales Mgr., International Steam Pump Co.

**Robert Ray Van Valkenburgh,** Juneau, Alaska.
Proposed by Burt B. Brewster, B. L. Thane, B. B. Nieding.
Born 1889, Le Roy, N. Y. 1908, Completed course in Le Roy High School, Le Roy, N. Y. 1909, Completed one year's course at Rochester Athenium and Mechanics Inst., Rochester, N. Y. 1913, Completed 3 1/2 years' course in Michigan College of Mines; B. S. and E. M. 1914, Michigan College of Mines; B. S. 1912, Six months as timberman, and miner at Franklin Jr. Mines, Demmon, Mich. 1913, Six months as miner at Perseverance Mine, Alaska Gastineau Mining Co., Juneau, Alaska.
Present position: Shift boss, Alaska Gastineau Mining Co., Juneau, Alaska.

**F. de Venancourt,** Paris, France.
Proposed by W. H. Corbould, Charles F. Rand, T. T. Read.
Born 1870, Martinque. 1887, Master of Arts "Diplome" from the Ecole Centrale des Arts et Manufactures, Paris. 1896–99, Copper Mines of "Boles," Lower California. 1899–1901, Boa Kita Gold and Diamond Mining, Brazil. 1901–03, Paras Gold Mines, Colombia (Genl. Mgr.). 1903–05, Mgr., Djebel Ahmar Copper Mines, Genl. Mgr., Paramatta Copper Mines. 1905–09, Smelting works, So. Australia.
Present position: Consulting Engineer since 1909, Asst. Prof. at the Ecole Centrale, Paris (metallurgy of the smaller metals).

**Norman L. Warford,** Anaconda, Mont.
Proposed by Frederick Laist, C. D. Demond, James K. Murphy.
Born 1865, Bridgton, Pa. 1882, Phillipsburg, N. J. (H. S.) 1892–95, I. C. School and Easton, Pa., night schools, and years of practical experience. 1905–08, Engineered, built, and operated the Washington portland cement plant. 1913–14, Engineered, built, and put in operation the coal pulverizing plant for Anaconda Copper Mining Co., Reverberatory Smelting Dept.
Present position: Supt., Pulv. Coal Plant, A. C. M. Co.

**Eugene E. Watton,** Denver, Colo.
Proposed by R. D. George, H. S. Thayer, Richard A. Parker.
Born 1881, St. Louis, Mo. Public schools, no degrees. 1902, Mt. Vernon Cor. Mfg. Co., Mt. Vernon, Ill. 1903–04, Traffic Dept., St. Louis Southwestern Ry., Tyler, Texas. 1904–07, Operating Dept., D. S. S. and A. Ry., Marquette, Mich. 1907–08, Executive Dept., Mich., Kansas, and Topeka Ry., St. Louis, Mo.
Present position: 1908 to date, Treas. and Genl. Mgr., United Oil Co.

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**Walter N. Wetzel**, Sunnyside, Utah.
Proposed by H. N. Eavenson, J. C. Roberts, W. C. Furguson, Alexander Bowie.
Born 1884, Dakota, Ill. 1900–04, Eng. Dept., H. C. Frick Coke Co. 1904–05, Eng. Dept., U. S. Coal & Coke Co. 1905–07, With Hogg & Porter, Engrs., Uniontown, Pa. 1907–08, Eng. Dept., Colo. Fuel & Iron Co. 1908–12, Eng. Dept., Utah Fuel Co. 1912, Supt., Utah Fuel Co.
Present position: Mine Supt.

**John Charles Williams**, Dawson, N. M.
Proposed by T. H. O'Brien, Harry J. Wolf, F. W. Traphagen.
Born 1890, Butte, Mont. 1897–1908, Grammar and High Schools, Denver, Colo. 1908–09, Univ. of Denver. 1909–13, Colorado School of Mines; E. M. 1913–14, General stamp mill and cyanide plant work, Oroya Leonesa, Matagalpa, Nicaragua; Junta G. M. Co., Telluride, Colo.; Liberty Bell, Telluride.
Present position: July, 1914, to date, Chemist, Stag Cañon Fuel Co., Dawson, N. M.

**Shigetake Yoshiwara**, Tokyo, Japan.
Proposed by Tadashiro Inouye, B. Katsura, S. Kano.
Born 1886, Tokyo, Japan. 1910, graduated, technical college, Imperial Univ. of Kioto in Japan; "Kogakushi." 1910–12, studied in Germany, France, and U. S. A.
Present position: 1910 to date, Engr., Kobe Steel Works in Japan and Daikoku Copper Mine, Japan.

## *Associate Members*

**Peter Louis Goddard**, Cloncurry, Queensland, Australia.
Proposed by W. H. Corbould, T. T. Read, Charles F. Rand.
Born 1881, Marseilles, France. 1887–1896, Superior Public School, Young, N. S. W. 1896–99, Private Grammar School, Young, N. S. W. 1902–03, Blast-Furnace, Chilagoe Co., Queensland. 1903–04, Converter Shift Boss, Chilagoe Co., Queensland. 1905–08, Converter Foreman, Mount Molloy Co. 1909, several months general smelter hand at Cobar, Port Kembla, Mt. Lyall, Tasmania, Mt. Morgan, and Great Fitzroy, Queensland.
Present position: 1909 to date, Smelter and converter foreman, Mt. Elliott, Ltd., Selwyn, Queensland.

**James Tenbrook Roberts**, Butte, Mont.
Proposed by Frederick Laist, E. P. Mathewson, C. D. Demond.
Born 1864, Decatur, Ill. 1877–81, Student in Decatur, Ill., High School.
Present position: Acct. and Asst. Secy., Anaconda Copper Min. Co.

**Andrew Constant Vauclain**, Philadelphia, Pa.
Proposed by J. E. Thropp, Jr., E. P. McClure, Karl Nibecker.
Born 1873, Altoona, Pa. 1891, Grad., Altoona High School. Thereafter various night classes and personal studies, during and subsequent to Baldwin apprenticeship. 1891–1912, Baldwin Locomotive works, Phila., as apprentice; 1912–14, in various capacities, chiefly executive and including field engineering this country and in Europe, Asia, and Africa.
Present position: Vice-Pres., Southwark Foundry & Machine Co., in charge of all engineering and construction and manufacturing.

## *Junior Members*

**Peter A. Aline**, Butte, Mont.
Proposed by D. C. Bard, C. H. Bowman, Theodore Simons.
Born 1892, Great Falls, Mont. 1899–1907, Public schools, Great Falls, Mont. 1907–11, Great Falls High School. Worked during the summer months of 1911–12–13 and 1914 at the Boston and Montana Smelter, Great Falls, Mont.
Present position: Senior, Montana State School of Mines.

**Roy Samuel Bonsib**, New York, N. Y.
Proposed by Robert Peele, Charles P. Berkey, J. F. Kemp.
Born 1885, Vincennes, Ind. 1903, Grad., Vincennes High School. 1906, Grad., Madison Academy, Uniontown, Pa. 1910, Grad., Indiana Univ.; A. B.; 1911, A. M.
Present position: Student, Columbia University, School of Mines.

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**Lewis Warner Fogg, Jr.**, Uniontown, Pa.
Proposed by W. R. Crane, C. E. McQuigg, H. D. Pallister.
Born 1892, Dairy, Pa. 1906–10, Uniontown High School. 1910–15, Pennsylvania State College. 1911, Coke Inspector. 1909–10, Shipping Clerk, Tower Hill Coke Co. 1913, Payroll Clerk, Craige County Lumber Co.
Present position: Student, Penn. State College, School of Mines.

**Bertram Grant**, Golden, Colo.
Proposed by F. W. Traphagen, William R. Chedsey, Harry J. Wolf.
Born 1892, Norristown, Pa. 1910, Central Manual Training School, Philadelphia, Pa. 1912, Utah Fuel Co., Castle Gate, Utah. 1913, Barstow Mining Co., Ouray, Colo.
Present position: Student, Colorado School of Mines, Golden, Colo.

**Herbert M. Harbach**, State College, Pa.
Proposed by W. R. Crane, C. E. McQuigg, H. D. Pallister.
Born 1892, Lebanon, Pa. 1911, Grad., Lebanon High School in Scientific Course.
Present position: Student, Penn. State College.

**John Edward Hunt**, Butte, Mont.
Proposed by D. C. Bard, C. H. Bowman, Theodore Simons.
Born 1893, Butte, Mont. 1899–1907, Graded School of Butte. 1907–08, St. Patrick's High School, Butte. 1908–11, Central High School, Butte. 1911–12, Marquette Univ., Milwaukee, Wis. 1912–14, Anaconda Copper Ming. Co., mining and timbering. 1914, Sampler, Timber Butte Milling Co. 1914, Asst. in Eng. Dept. Elm Orlu Mining Co.
Present position: Student, Montana State School of Mines, Butte, Mont.

**Robert M. Hutchison**, State College, Pa.
Proposed by W. R. Crane, C. E. McQuigg, H. D. Pallister.
Born 1890, Harrisburg, Pa. 1909, Grad., Harrisburg High School, in Latin Scientific course. 1909–10, City Engineering Dept., Harrisburg, Pa. 1910–11, Morris County Traction Co., Morristown, N. J. 1911–14, Board of Public Works, Harrisburg, Pa.
Present position: Student, Penn. State College.

**George V. Luerssen**, State College, Pa.
Proposed by W. R. Crane, C. E. McQuigg, H. D. Pallister.
Born 1890, Plainfield, N. J. 1907, Reading High School, graduate in general scientific. 1907–11, Crucible dept. of Carpenter Steel Co., Reading, Pa. 1911–14, Three summers in crucible and open hearth depts. of same company.
Present position: Student, Penn. State College.

**Bertram A. Reber**, Butte, Mont.
Proposed by D. C. Bard, C. H. Bowman, Theodore Simons.
Born 1892, Lindsay, Neb. 1907, Grad., St. Patrick's Parochial School, Butte, Mont. 1911, Grad., Butte High School. 1911–12–13, winters, employed by the Anaconda Copper Min. Co., underground at West Colusa Mine.
Present position: 1911 to 1915, Student at Mont. State School of Mines.

**Albert E. Rhoads**, State College, Pa.
Proposed by W. R. Crane, C. E. McQuigg, H. D. Pallister.
Born 1892, Harrisburg, Pa. 1911, Grad., Central High School of Harrisburg in Latin Scientific course. 1911–12–14, Three summers' experience with Carborundum Co., Niagara Falls, experimental research and analytical work. 1913, Three months' work on furnaces of Frontier Brass Co., Niagara Falls.
Present position: Student, Penn. State College.

**Guy Thomas Woodworth**, Butte, Mont.
Proposed by D. C. Bard, C. H. Bowman, Theodore Simons.
Born 1891, St. Paul, Minn. 1897–1909, Graded schools, Great Falls, Mont.
Present position: 1911 to date, Student, Montana State School of Mines, Butte, Mont.

### *Change of Status, Junior to Member*

**Alan Hay Means**, Boston, Mass.
Proposed by W. Lindgren, H. O. Hofman, Edward E. Bugbee.
Born 1890, Peru, Ill. 1913, Grad. Mass. Inst. of Tech.; S. B. 1914, Geol. Dept., Mass. Inst. of Tech.; M. Sc. 1913, summer, Geological field work, Red Cliff, Colo. 1914, Advanced work in Geology.
Present position: Special geological work at Institute of Technology.

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## CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Oct. 10 to Nov. 10, 1914. This list, together with the list published in *Bulletin* Nos. 88 to 95, April to Nov., 1914, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of Nov. 10, 1914.

ADAMS, HENRY.....Shippan Ave., Shippan Point, Stamford, Conn.
ADAMS, RALPH E.....711 Mills Bldg., El Paso, Texas.
ALDEN, HERBERT W.....21 Edison St., Detroit, Mich.
ALDERSON, VICTOR C.....1043 Emerson St., Denver, Colo.
ALSDORF, FREDERICK C.....1207 W. Third St., Los Angeles, Cal.
AMMON, ROBERT.....American Zinc Co. of Tenn., Mascot, Tenn.
AUSTIN, J. F.....Care Am. Smelting & Refining Co., Leadville, Colo.
BARNES, BLAKESLEE.....Care Arrow Engineering Co., Palmyra, Mo.
BAYER, AUGUST H. VON.....Care Semet-Solvay Co., Lebanon, Pa.
BEALL, ALBERT S. E.....P. O. Box 126, Hollywood, Cal.
BOLLES, MYRICK N.....812 Medary Ave., Brookings, S. D.
BRADLEY, PHILIP R.....Treadwell, Alaska.
BRYANT, ALBERT DALY.....Lock Box 616, Palmerton, Pa.
BUCHANAN, JEROME ROBINSON....Genl. Supt., Care Imperial Reduction Co., Ogelby, Cal.
BUCK, CHARLES A.....Vice-Pres., Bethlehem Steel Co., S. Bethlehem, Pa.
BUCKBY, DE NARD WILSON.....Mill Supt., Stewart Min. Co., Wallace, Idaho.
BUELL, LLOYD T.....359 Warburton Ave., Yonkers, N. Y.
CADMAN, JOHN.....61 Wellington Road, Edgbaston, England.
CARPENTER, ARTHUR HOWE.....19 Hawthorne Ave., Crafton, Pa.
CARTLIDGE, OSCAR.....1363 Whittier Ave., Springfield, Ill.
CARUTHERS, A. WAYNE..Care R. V. Norris, 524 Second Bank Bldg., Wilkes-Barre, Pa.
CHASE, EDWIN E.....1450 Williams St., Denver, Colo.
CHASE, F. D.....Apartado 27, Monterey, Mexico.
CORBUS, A. W.....Los Molinos, Tehama Co., Cal.
COX, THOMAS.....P. O. Box 1, Fellows, Cal.
COXE, CHARLES E.....Care E. M. Morgan, P. O. Box 266, Reading, Pa.
CREMER, FELIX.....Care Metropolitan Exhibit, Los Angeles, Cal.
CREMER, J. H.....2800 Washington Ave., Cleveland, Ohio.
CUSHING, DANIEL.....“The Wampum,” 7216 Lexington Ave., Cleveland, Ohio.
DAY, DAVID T.....1331 F Street, N. W., Washington, D. C.
DEAN, A. L.....P. O. Box 1353, Victoria, B. C., Canada.
DEFTY, WILLIAM E.....618 N. Third Ave., Phœnix, Ariz.
DEGOLYER, E. L.....531 University Boulevard, Norman, Okla.
DOBSON, CHRIS G.....302 Colonial Blk., Butte, Mont.
DUFOURCQ, R. G.....Care Montezuma Lead Co., Santa Barbara, Chih., Mexico.
ERDLETS, J. F. B., JR.....1 London Wall Buildings, London, E. C., England.
EVANS, MILTON R.....236 Gardner St., Plymouth, Pa.
FULTON, CHARLES H.....3046 Euclid Heights Blvd., Cleveland, Ohio.
FULTON, CHESTER A...Care Mina Lo Increible, El Callao, Yuruari, Venezuela, S. A., via Trinidad.
GALLOWAY, A. D. R.....Abosso, Gold Coast Colony, West Africa.
GAMBRILL, G. T.....1419 Eutaw Place, Baltimore, Md.
GARD, IRVING R.....477 Vermont Place, Columbus, Ohio.
GOODRICH, HAROLD B.....450 West 147th St., New York, N. Y.
GRAY, JOHN N. D.....Care Consolidated Arizona Smelting Co., Humboldt, Ariz.
GREBE, EMIL.....Tombstone, Ariz.
GREENE, LEWIS H.....2808 Claremont Boulevard, Berkeley, Cal.
GREENSFELDER, NELSON S.....1109 Summit Ave., Seattle, Wash.
GRIFFIN, F. W.....506 Alaska-Commercial Bldg., San Francisco, Cal.
HAAS, J. C.....Apt. G, The Wellington, Sixth Ave. and Stevens St., Spokane, Wash.
HALL, WILLIAM J....Care Federal Mining & Smelting Co., 32 Broadway, New York, N. Y.
HANSEN, GEORGE T.....609 Kearns Bldg., Salt Lake City, Utah.

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HARDY, J. GORDON....312 City Natl. Bank Bldg., El Paso, Texas.
HARMS, ERNEST....Apartado 93, Torreon, Coah., Mexico.
HAUPT, LEWIS H....Box 65, Palmerton, Pa.
HIBBERT, ARTHUR....Creston, Marple Bridge, Derbyshire, England.
HILDRETH, THOMAS F...Davidson Ore Mining Co., 403 White Bldg., Buffalo, N. Y.
HINCKLEY, E. R....Care D. C. McCormick, Perkins Place, Norwich, Conn.
HOFFMANN, JOHN I....32 Great St. Helens, London, E. C., England.
HOWE, ALBION S....Minas de Matahambre, Prov. Pinar del Rio, Cuba.
IMHOFF, ALEXANDER....Box 122, Murphy, Cal.
KEEN, CORNELIUS D....203 Commercial Natl. Bank Bldg., Shreveport, La.
KELLEY, ARTHUR L....530 S. Grape St., Medford, Ore.
KEMP, L. W....Chuquicamata, via Antofogasta, Chile.
KEPNER, R. B....Velardena, Durango, Mexico.
KROM, S. ARTHUR....609 Madison Ave., Plainfield, N. J.
KURIE, FRANK M....625 I. W. Hellman Bldg., Los Angeles, Cal.
KWANG, KWONG YUNG....Lincheng Mines, Lincheng, via Peking, North China.
LAWRIE, H. N....Atlantic Highlands, N. J.
LE CHATELIER, H. L....13 rue Lesdiquieres, Grenoble, France.
LINDAU, S. PAUL....429 E. Mill St., Liberty, Mo.
LONGYEAR, J. M., JR....Leicester St., Brookline, Mass.
LOUX, CARL H....Pocatello, Idaho.
LOVE, JAMES W....Leonard Hotel, Butte, Mont.
LUCKE, P. K....The Limes, East Molesey, Surrey, England.
MCALLEN, JOHN L....170 Eleventh Ave., Seattle, Wash.
MCBRIDGE, WILBERT G....Oracle, Ariz.
MCDANIEL, ALEXANDER K....50 W. 4th Ave., Denver, Colo.
MCINTOSH, FRANK K....Cook, Minn.
MCLURE, CHARLES D..Care Granite Bi-Metallic Cons. Min. Co., Philipsburg, Mont.
MCNEILL, W. F., Sec'y-Treas., Western Coal Operators Assn., 912 Herald Bldg., Calgary, Alta., Canada.
MCNUTT, V. H....Billings, Mont.
MACDOUGALL, CHARLES W....Millington, N. J.
MAGEE, JAMES F....Sugar Loaf, Boulder Co., Colo.
MARTIN, SAMUEL G....43 Cedar St., New York, N. Y.
MARSH, HARRY W....Moscow, Idaho.
MARSH, ROBERT, JR., Care M. Guggenheim & Sons, 165 Broadway, New York, N. Y.
MARSHALL, N. C....253 College Ave., Houghton, Mich.
MENTZEL, CHARLES....135 Broadway, New York, N. Y.
MITCHELL, T. E., Burma Mines, Ltd., Bawdwin, P. O. Nam Tu, via Manpwe Junction, Northern Shan States, Burma, India.
MOFFETT, CHARLES A....Alabama City, Ala.
MORRIS, RICHARD J....509 Lincoln Drive, Germantown, Philadelphia, Pa.
MORSE, GEORGE H....Huron, Ohio.
MURPHY, E. M....2008 Oneida Place, Spokane, Wash.
NAITO, HISAHIRO....Care Nippon Oil Co., Ltd., Marunouchi, Tokyo, Japan.
NARAVANE, S. G..Asst. Mgr., Choitidih Colliery, P. O. Katrasgarh, E. I. Ry., India.
NORTON, EDWARD G....6023 Vernon Ave., Chicago, Ill.
OLDFIELD, F. W....Mexican Mines Co., Bolanos, Jal., via Zacatecas, Mexico.
PAINTER, ROBERT K....Care General Chemical Co., 25 Broad St., New York, N. Y.
PAYNE, ARTHUR C...Care Oil Fields of Mexico Co., 576 Fifth Ave., New York, N. Y.
PEARSON, WILLIAM R....414 W. 119th St., New York, N. Y.
PERRY, W. A....7 East 56th St., New York, N. Y.
PROCHAZKA, GEORGE A., JR....138 W. 13th St., New York, N. Y.
PROMMEL, H. W. C....739 E. 14th St., Denver, Colo.
PRYER, WILLIAM CRISTY....General Delivery, Deadwood, S. D.
RALEIGH, J. R....Rossville, via Cooktown, Queensland, Australia.
RIBALKIN, M. P....Vassilievsky Ostrov, 6th Line, No. 17, Petrograd, Russia.
RICHMOND, WILLIAM H....Richmond Hill, Scranton, Pa.
ROBINSON, THEODORE W....Room 1620, 208 S. La Salle St., Chicago, Ill.
SAUNDERS, T. SKEWES....“Dos Estrellas,” Apartado 34, El Oro, Mex., Mexico.
SAYRE, MORTIMER F....229 Parkwood Bldg., Schenectady, N. Y.
SCHEBLE, M. C....Vice-Pres. and Gen. Mgr., Bokoshe Smokeless Coal Co., Bokoshe, Okla.
SCHMIDT, WILLIAM C., JR....1056 St. Nicholas Ave., New York, N. Y.
SCHWARZ, CHARLES E....6234 Arundel Place, St. Louis, Mo.

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SHALER, MILLARD K.....2 Bishopsgate, London, E. C., England.
SLOAN, WILLIAM ARTHUR.....212 N. Louise St., Glendale, Cal.
SMITH, CHARLES ALBERT.....3326 N. 17th St., Philadelphia, Pa.
SMITH, HOWARD I.....Urbana, Ill.
SPROAT, A. D.....Care American Consul, Vera Cruz, Mexico.
STERLING, ROBERT.....Goldfield Consolidated Mines Co., Goldfield, Nev.
STERNFELD, THEODORE.....Treas., American Metal Co., 61 Broadway, New York, N. Y.
STEVENS, BLAMEY, Lane Rincon Mines, Inc. (S. A.), Tamascaltepec, Estado de Mexico, Mexico.
STICHT, ROBERT.....Long Beach, Cal.
STUCKEY, L. C.....Mill Close Mine, Darley Dale, Matlock, England.
STÜTZ, ERNEST.....87 Nassau St., New York, N. Y.
SWEENY, HARRY P.....Clementon, N. J.
TACKMANN, HENRY.....253 East 82d St., New York, N. Y.
TARANTOUS, RICHARD.....Care Inspiration Cons. Copper Co., Miami, Ariz.
THOMAS, GEORGE S.....627 Moffet Ave., Joplin, Mo.
THOMAS, W. S.....Care Citizens Coal Co., Salt Lake City, Utah.
TIMMONS, COLIN.....Atlas Mine, Jamestown, Cal.
TRENGOVE, S. R.....2341 Ellsworth St., Berkeley, Cal.
TUSCHKA, OTTO.....Care Smelter No. 2, Monterey, Mexico.
VAN DYKE, B.....821 N. 16th St., Harrisburg, Pa.
VAN LIEW, W. R.....150 Willard Ave., Bloomfield, N. J.
VAN ZWALUWENBURG, A.....la Madrid 15, Mexico City, Mexico.
VIVIAN, HUGH.....Instructed to hold all mail.
WALKER, R. T.....1812 Virginia St., Berkeley, Cal.
WARD, HARRY J.....948 Market St., San Francisco, Cal.
WELLMAN, S. T.....8803 Euclid Ave., Cleveland, Ohio.
WELLS, JAMES S. C.....Care R. deC. Greene, 105 Thomas Ave., Cranford, N. J.
WIERUM, HOWARD F.....Care Chemists' Club, 52 E. 41st St., New York, N. Y.
WILKINSON, JOHN F.....1016 Webster St., Oakland, Cal.
WILLIAMS, RALPH B.....96 High St., Newburyport, Mass.
YOUNG, RALPH W.....12 Prospect Place, New Haven, Conn.
YOUNG, WILLIAM A.....Lewistown, Mont.

## ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name — Last address of Record, from which Mail has been Returned.

AMBROSIUS, J. R.....Guanajuato, Mexico.
ARMAS, MILTIADES TH.....Paris, France.
BODY, J. FRANCIS.....New York, N. Y.
GAMBRILL, G. T., JR.....Birmingham, Ala.
HALL, S. W.....Fredericktown, Mo.
KERR, DAVID GILLESPIE.....Toronto, Ont., Canada.
NEWMAN, BRUNO.....Apartado 90, Aguascalientes, Mexico.
NORRIE, WILLIAM G.....Three Forks, B. C., Canada.
PALMER, CORTLANDT E.....New York, N. Y.
PINKHAM, W. F.....Battle Mountain, Nev.
RAMBO, WILLIAM C. J.....Denver, Colo.
RANDOLPH, B. S.....Tonoloway Quarries, Tonoloway, via Hancock, Md.
RODGERS, JOSEPH H.....Seattle, Wash.
RODRIQUEZ, J. C.....Coahuila, Mexico.
SHEAFE, HARRY J.....El Dorado, Cal.
THOMAS, CHARLES S., JR.....St. Louis, Mo.
WENTWORTH, IRVING H., 245 Belden Ave., Harlandale Addition, San Antonio, Texas.
WHITE, R. T.....Dzansoul, Russia.
WRAIGHT, ERNEST A.....London, S. W., England.
WRIGHT, PERCY E.....Seattle, Wash.

## NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Oct. 10 to Nov. 10, 1914:

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| Date of Election | Name | Date of Decease. |
|---|---|---|
| 1905 | *COLLINS, J. P. | January 21, 1914 |
| 1889 | †HEINZE, F. AUGUSTUS | November 4, 1914 |
| 1911 | *MASTER, NOEL M. | November —, 1913 |
| 1887 | *YORK, JAMES E. | October 4, 1914 |

*Member. †Associate.

## *Biographical Notices*

*Albert Grothe* was born April 4, 1841, in Hagen, Westphalia. His father, Dietrich Grothe, was a professor at the Technical School (Gewerbschule) in that town, until, during the political unheaval of 1848, his revolutionary and democratic views caused his exile from Germany. He settled in the Netherlands as Professor of Technology, first at the University of Utrecht and later at that of Delft.

Albert Grothe was educated in the public school and University of Utrecht, and at an early age entered the Dutch Army, and went out to the Dutch East Indies with his regiment. After serving a couple of years, he was led by the strong natural bent of his mind to abandon a military career and to become connected with railroad construction in the island of Java. In 1869 he returned to Europe, and entered the employ of De Bergue & Co. of London, England, a firm of contracting engineers, by whom he was sent to Russia to superintend the construction of a series of bridges on the St. Petersburg-Moscow Railroad during the winter of 1870–71. On the completion of this work, the same firm had obtained the contract for the erection of the first Tay Bridge, and Mr. Grothe was given the management of the whole work. On the death of Mr. De Bergue, the contract was taken over by Hopkins Gilkes & Co. of Middlesbrough, who retained Mr. Grothe's services until the completion of the undertaking. At the investigation of the terrible disaster which took place two years after the bridge had been turned over to the North British Railway Co., Mr. Grothe was one of the principal witnesses, and it was largely due to his evidence under severe cross-examination that both the engineer, Sir Thomas Bouch, and the contractors were completely exonerated from all blame in the matter.

After some minor harbor construction in the north of Scotland, Mr. Grothe was offered the position of Manager by the Tharsis Copper Co. in the south of Spain, and this definitely turned him from construction-work to mining. He went to Spain late in 1879. During the four years he remained there, he worked out some important improvements in the hydro-metallurgy of copper, particularly in the remarkable process of leaching in *terreros* with subsequent precipitation by metallic iron, which as carried out in southern Spain to-day is unique for the gigantic scale of its operations and the efficiency of its results. After severing his connection with the Tharsis Co. his attention was directed to Mexico, and he was appointed General Manager of the United Mexican Mining Co. of Guanajuato. While there, it is said, he introduced the system of pan-amalgamation, abandoning the old patio process in the company's reduction works.

In 1892, Mr. Grothe became interested in irrigation work in Idaho and prepared to make his home there, becoming an American citizen; but it was not long before he turned to mining again, which inevitably led him back to Mexico in the year 1900. Since that time he has had various

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interests in that country, turning his mind more and more to the metallurgical branch of the profession. In 1907 he formed a partnership with Herbert Fuller Carter, an English engineer and a member of the Institute of Mechanical Engineers of London. Under the firm name of Grothe & Carter, they were prominent as consulting engineers and introducers into Mexico of the most modern and up-to-date metallurgical methods and appliances, among which may be mentioned the Grothe air-agitating tank (described in *Trans.*, xl, 771, 918).

The last four years of storm and stress in Mexico, and the consequent anxiety and nervous strain, undermined his naturally strong physique so that he was not able to withstand the operation which at the end became necessary, and he passed away after a short illness on the 19th of August last, in the seventy-fourth year of his age.

Mr. Grothe to the end was a man young in mind and spirit, thoroughly abreast of the times in his profession, and keenly interested in furthering scientific research and correct engineering principles. He was one of the founders and charter members of the Mexican Institute of Mining and Metallurgy, and was its first President, serving two terms in that capacity and one year as Secretary. He was also, besides being a member of the American Institute of Mining Engineers (which he joined in 1886), a member of the Institution of Civil Engineers of London, England.

*John Barnard Swett Pearse* was born in Philadelphia on Apr. 19, 1842. His father, Oliver Peabody Pearse, a sea captain in the China trade, was drowned while saving a bather at Cape May, June 29, 1848. His mother, whose maiden name was Adelia Coffin Swett, afterward married Dr. Edward Hartshorne, of Philadelphia, and had issue, Joseph Hartshorne, who is now a metallurgical expert at Pottstown, Pa. Mr. Pearse received his early education at a private school under Charles Short. On graduating from Yale College in 1861, he entered Booth and Garrett's chemical laboratory in Philadelphia as a student, where he remained until June, 1863, at which time he took charge of the chemical division in the U. S. army laboratory in Philadelphia, manufacturing various pharmaceutical products for the hospital service, until July, 1865. He then spent over two years in practical studies of mining engineering at the School of Mines at Freiberg, Saxony, and at that at Loeben in Styria, making a specialty of metallurgy, particularly of iron and steel, visiting and working at German and Austrian works and mines. Particularly was he interested in the mechanical devices used, as well as the methods employed in handling different ores from the mine to the finished product.

In January, 1868, he entered the service of the Pennsylvania Steel Co. at Harrisburg as chemist, and in 1870 became General Manager of the company, succeeding A. L. Holley. Here he successfully improved the design and product of the Bessemer steel plant, bringing the production up from 20 to 100 tons a day. In this connection, he made many inventions, for few of which, however, he has received credit. Among these were the slag hole in a continuous melting cupola, and the false plate on the converter bottom, which were universally adopted. He also proposed the use of traveling cranes in steel works, and designed a plant on this basis, but was refused money by his directors for the installation. In the design of plant buildings, he drew designs for riveted steel trusses, but was unable to induce any of the then existing bridge companies to build such designs.

He also made the first steel for rails, hammered blooms being at first

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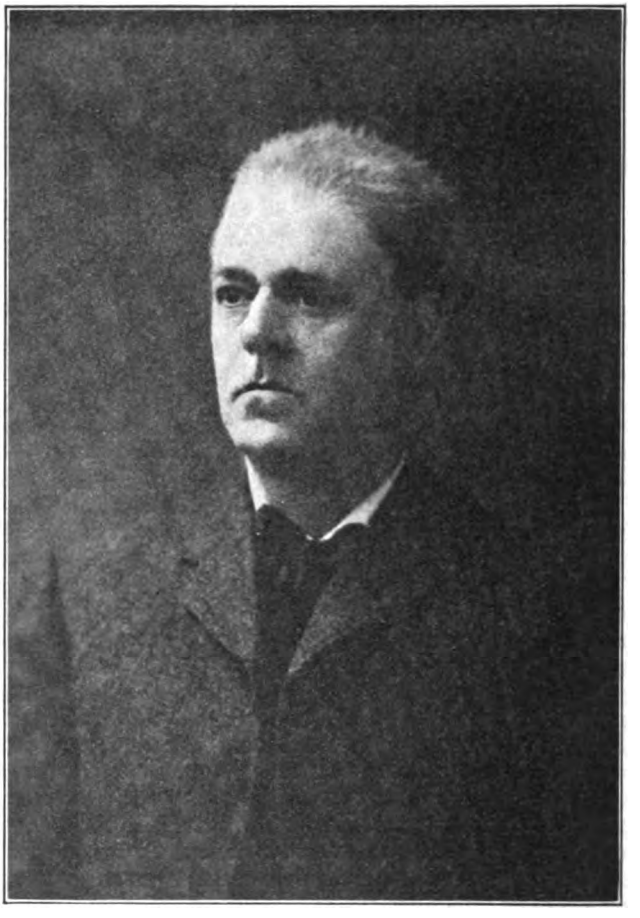
rolled at the Cambria Iron Works and later in rolling mills built by him at Harrisburg. He was probably instrumental in first making Bessemer pig iron from native New Jersey and Pennsylvania ores. The steel rails supplanted the wrought-iron rails then in use, which on certain cross-overs of the Pennsylvania Railroad at that time required daily replacing. The first steel rails lasted over two weeks under similar conditions. He left the Pennsylvania Steel Co. in the early part of 1874.



JOHN BARNARD SWETT PEARSE.

In June, 1874, he was appointed a Commissioner and Secretary of the Second Geological Survey of Pennsylvania, and was largely influential in carrying on its work until he left the State. During the Centennial Exposition in Philadelphia in 1876, he was on a committee in charge of the metallurgical and mining exhibits. In 1876 he went to the South Boston Iron Works as General Manager and there conducted a general foundry and machine shop business, building cast-iron guns and converted guns, as well as projectiles, for the government. He also built the largest lathe of its time for the Watervliet Arsenal and a special boring machine, as well as steam pumps from the designs of E. D. Leavitt for the Calumet &

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Hecla Works. He was especially interested in developing the strength of cast iron, primarily for guns, in which he was eminently successful. He foresaw, however, the superiority of the built-up steel gun now in common use, but was unable to interest his associates. He also invented a breech-block mechanism of superior type for large guns.

In 1883, his health being impaired by overwork, he resigned from the South Boston Iron Works and went with his family to England, where for several years he studied voice production under Emil Behnke, and the Tonic-Sol-Fa system, taking a diploma at the college in South London, and in this way becoming a cultivated musician, with a good knowledge of harmony. After that time he was not active in any professional mining or metallurgical work. On his return to the United States in 1888, he settled in Roxbury, Mass. He then became deeply interested in the violin, and up to the date of his death spent much time investigating the theory of that instrument and the materials and the methods of its construction from a scientific standpoint. This work led to the collection of a large library on woods, varnishes, violins, acoustics, and allied subjects, and to the translation of several hundreds of foreign references in French, German, Dutch, Danish, Latin, Italian, Spanish, and Russian, all of which were written out in long hand, with comments thereon. His summers were passed at his camp in Canada on Lake Memphremagog. In order to understand the operation of motor boats, he attended the Franklin Union, in Boston, at the age of 69, being the oldest pupil, and took a regular course in industrial electricity, covering two years, and received a diploma Mar. 31, 1913.

At all times, his brain was active and interested in scientific subjects. Interested in education, he often commented on the great advantages of the present day student in scientific training. At Yale, in his day science was in its infancy. The teaching of physics and chemistry was elementary and crude, so that practically all his scientific training was derived after graduation. His opinions were pronounced. Once a conclusion was reached, his will never swerved in carrying out his purpose. In his business relations he was honest and just, but those who tried to impose on him usually regretted it.

Although a man of high scientific attainments, and one of the foremost metallurgical experts of his time, he published but little. In 1869 he published *A Treatise on Roll Turning for the Manufacture of Iron*, translated with additions from the German of P. Tunner, which is a standard work even to-day. He also published in 1876 *A Concise History of the Iron Manufacture of the American Colonies up to the Revolution and of Pennsylvania until the Present Time*, and wrote the article on steel and iron for Johnson's Cyclopædia. His mind ran largely to the working out of problems, and he was careless whether the results obtained were imparted to others or not. Although frequently asked by his friends to write the conclusions of his studies or articles for publication, he preferred not to do so, and consequently much of his knowledge and research, both in metallurgical and scientific lines, is lost. He early recognized the significance of various constituents in the manufacture of steel and was years ahead of his time, not only in the metallurgical but in the mechanical details desirable in the operation of steel works. Although a member of the American Institute of Mining Engineers from its foundation in 1872, and at one time Vice-President, he published little except discussions. In January, 1875, he was elected a member of the American Philo-

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sophical Society, and in 1871 became a member of the Iron and Steel Institute.

His election as a member of the American Institute of Mining Engineers took place at the Philadelphia meeting of Feb., 1872; and among his proposers was his friend (and on some professional questions his friendly antagonist), Alexander L. Holley, who had joined the Institute at the Troy meeting in Nov., 1871. At the Philadelphia meeting referred to, Mr. Pearse read a very able and important paper on The Manufacture of Iron and Steel Rails (*Trans.*, i, 162). This paper led to a lively discussion of the merits of hammering vs. rolling in the manufacture of rails, in which Mr. Pearse advocated the former and Mr. Holley the latter. Mechanical and commercial conditions have since settled the practice in favor of rolls, but the force of Mr. Pearse's arguments must still be recognized, if the questions of quality and structure be considered apart from those of rapid, cheap and convenient production.

At later meetings of the Institute he made brilliant contributions to the great discussion of the nature and nomenclature of iron and steel, for which his chemical knowledge eminently qualified him. These discussions are not fully reported in the *Transactions*, but their substance is in the following list:

| Title | Vol. | Page | Year |
|---|---|---|---|
| *Papers* | | | |
| The Manufacture of Iron and Steel Rails | i | 162 | 1872 |
| Improved Bessemer Plant | iv | 149 | 1875 |
| Iron and Carbon, Mechanically and Chemically Considered. | iv | 157 | 1875 |
| *Remarks* | | | |
| Remarks on Iron Chimneys | iv | 109 | 1875 |
| On What is Steel | iv | 147 | 1875 |
| On What Steel is | iv | 338 | 1876 |

After the latest of these contributions, which was orally made at the Washington meeting of Feb., 1876, he does not appear to have taken an active part in the proceedings of the Institute.

He was married on Nov. 1, 1876, to Mary Langdon Williams of Roxbury, Mass., who survives him, as well as a daughter, Alice Williams Pearse, and a son, Langdon Pearse, Division Engineer with the Sanitary District of Chicago. Mr. Pearse died almost instantaneously at his camp at Georgeville, P. Q., Canada, Aug. 24, 1914, from heart disease. He had not anticipated death, as his work was planned for many years.

Of a retiring disposition, making few friends, he was possessed of a very keen analytical mind, joined with a strong body, so that he worked continuously at a problem when interested. His tastes were varied, covering a wide range of subjects; horticulture, natural history, medicine, electricity, music, as well as metallurgy and the investigations of the violin and allied branches. While living in London he made a fine collection of engravings, particularly of the time of Earlom. His mind ran to the collecting not only of books but tools of various kinds. He keenly enjoyed discussing his interests with those friends qualified to do so, but would waste no time on others. He not only possessed the gift of handling men, but was an individual worker of great perseverance and ability.

LANGDON PEARSE.

*William Bleecker Potter* was born Mar. 23, 1846, at Schenectady, N. Y. His father was Rev. Horatio Potter, afterward the Protestant Episcopal

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Bishop of New York City. He was educated at Columbia University, from which he received in 1868 the degree of A.B., in 1868 that of A.M., and in 1869, upon graduation from the School of Mines, that of E.M. After some study abroad, he became assistant in the geological department of his *alma mater* from 1869 to 1871, and was also an assistant in the Ohio State Geological Survey. In 1871, when the chair of mining and metallurgy was established in Washington University, St. Louis, he became its first incumbent; and he remained at the head of that department for 22 years. Upon his resignation in 1893, it was discontinued. During its highly creditable history, not a few noted mining engineers and metallurgists were trained under Professor Potter.

In addition to the arduous duties of this position, he served from 1872 to 1874 as assistant in the Geological Survey of Missouri; from 1874 to 1878 as Engineer of the Pilot Knob Iron Co.; from 1876 to 1878 as consulting metallurgist of the Vulcan Iron & Steel Works; from 1882 to 1893 as Chief Engineer of the Iron Mountain Mining Co.; and from 1888 to 1893 as a member of the Board of the Missouri State Geological Survey, in which he had been an assistant 16 years before.

In 1899, he founded the well-known St. Louis Sampling & Testing Works, of which he was Manager for the rest of his life. He was widely known in the mining districts, especially the lead and zinc districts of Missouri, as a practicing and consulting engineer. In 1887, he was President of the Engineers' Club of St. Louis, and be became also a member of the American Society of Mechanical Engineers, and the Mining and Metallurgical Society of America, and corresponding member of the New York Academy of Science, the Wisconsin Academy of Science, and the National Geographic Society.

Professor Potter was one of the earliest members of the American Institute of Mining Engineers, which he joined in 1871, the first year of its existence. At the St. Louis meetings of May, 1874, and October, 1886, he was a leading spirit in the reception and entertainment of visiting members, and the organization of interesting and instructive professional excursions, as the reports of those meetings in vols. iii and xv of the *Transactions* show. From 1878 to 1880 he was a member of the Council, and in 1888 he was unanimously elected President of the Institute. The following is a list of his contributions to the *Transactions:*

The Character and Composition of the Lignite Coals of Colorado, vol. v, p. 365 (Philadelphia meeting, Oct., 1876).

A Present Need in the Engineering Profession, vol. xvii, p. 380 (Presidential Address at the Buffalo meeting, Oct., 1888).

Some Thoughts Relating to the American Institute of Mining Engineers, vol. xvii, p. 485 (Presidential Address at the New York meeting, Feb., 1889).

Remarks on Coal in Arkansas, vol. iii, p. 33 (St. Louis meeting, May, 1874).

Remarks on Indicative Plants, vol. xv, p. 660 (St. Louis meeting, Oct., 1886).

He died at his residence in St. Louis, July 14, 1914, after only a week's illness, leaving a widow, two sons and two daughters. Their sorrow is shared by a multitude of personal friends and a greater multitude of professional colleagues, who lament the loss of a distinguished and fruitful life, cut short in its prime. For, though Professor Potter was 68 years old when he died, and had behind him a long, active and honorable



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career, his departure must be counted premature, since it deprived his fellows of the treasure of his experience. It is sad to see young buds killed by an early frost, but the loss of a ripening harvest is a greater loss.

R. W. R.

*Charles Arthur Stewart* joined the Institute in 1910, and in the four years of his active connection became not only a contributor to its publications but a vigorous worker in the Spokane Local Section. His death, on Aug. 29, at Westhampton, Long Island, N. Y., deprives our number of one who could ill be spared. Mr. Stewart was born in New York City, Dec. 1, 1885. He was reared in Franklin, Mass., but when in 1902 he was prepared to enter Columbia College, his family moved to New York, which for six years became his home. In college his interest was aroused in geology and even before graduation he decided to follow this branch of science as a teacher. In 1907 he was appointed assistant in mineralogy at Columbia, and a year later, on taking his M. A., instructor in geology at Cornell, where he remained three years, profiting greatly by his association with Professor Ries. In 1906 he had become a candidate for the Ph. D. degree at his *alma mater*, and in 1910 finished his dissertation after a careful study of the Silver Bell mine, a very interesting contact zone in Arizona. The dissertation was published in the *Transactions*, vol. xliii, 240 (1912), and at the 1912 Commencement at Columbia the Ph.D. degree was awarded. In 1911 Mr. Stewart began his work of geological instruction at the University of Idaho, and entered devotedly into the life of the institution. He passed his summers in the field and had accumulated observations on some little-known districts of northern Idaho, on which he was at work when his fatal illness developed. He had also agitated the plan of a State geological survey for Idaho. Dr. Stewart wrote well and frequently. Geology did not alone claim his pen. Papers from him appear in the *Atlantic Monthly* and the *Educational Review*. His life was one of great promise as well as of actual accomplishment. Possessed of high ideals of service and of a warmly affectionate and winning disposition, he was singularly qualified for his chosen calling of teacher of science. In 1913 he married Miss Gertrude Douglas, of New York.

J. F. K.

*Publications of Charles Arthur Stewart*

THE MAGNETITE BELTS OF PUTNAM CO., N. Y. *School of Mines Quarterly*, vol. xxix, pp. 283 to 294 (1908).

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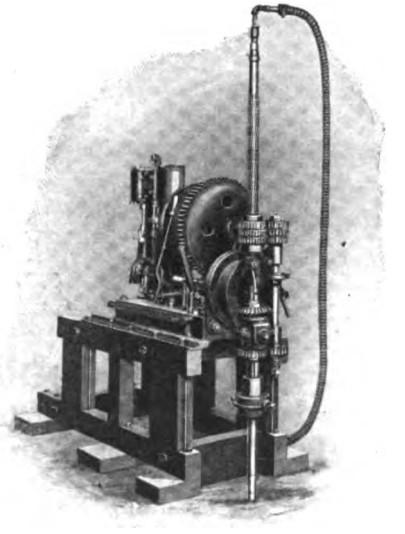

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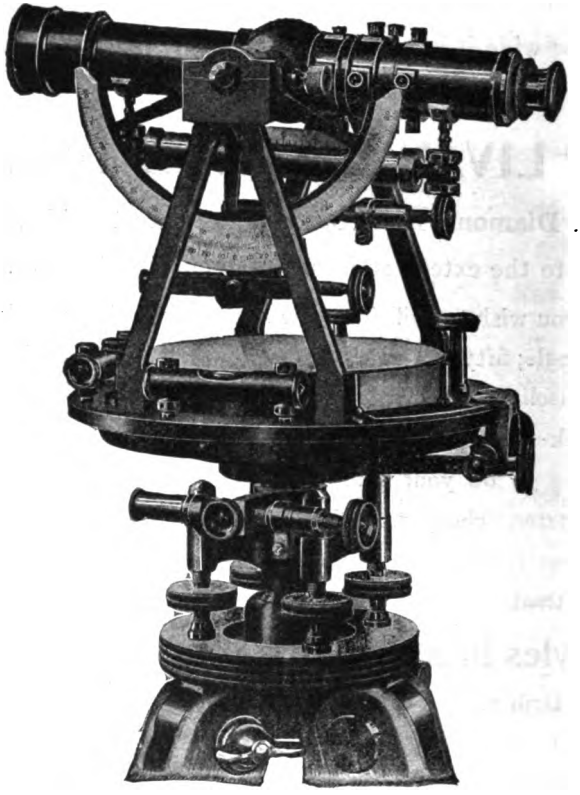

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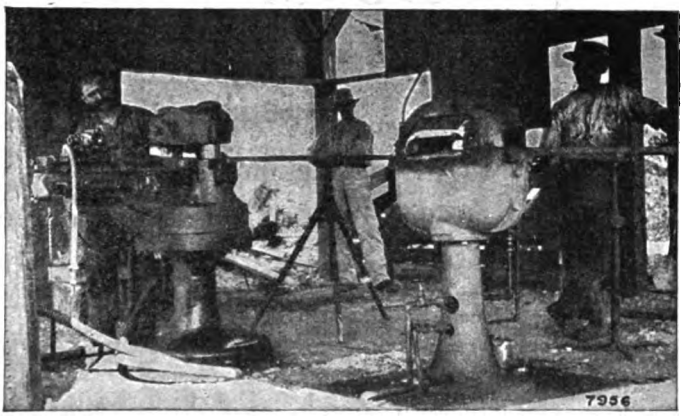


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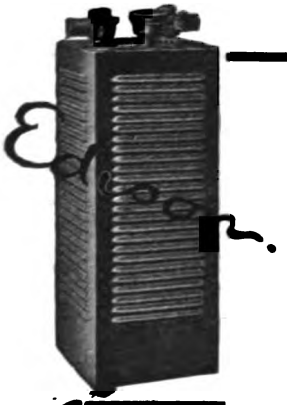

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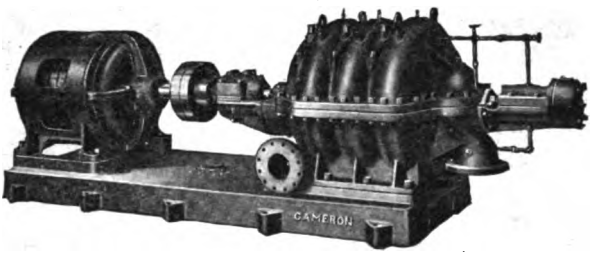


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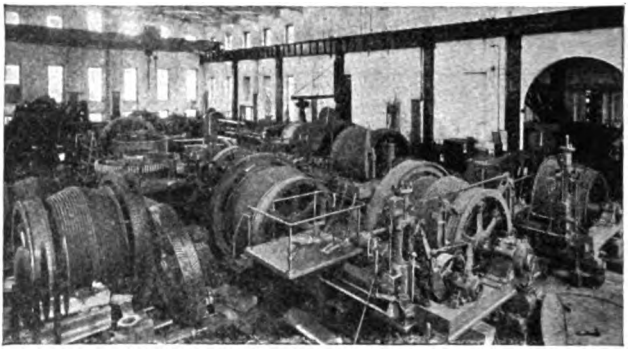

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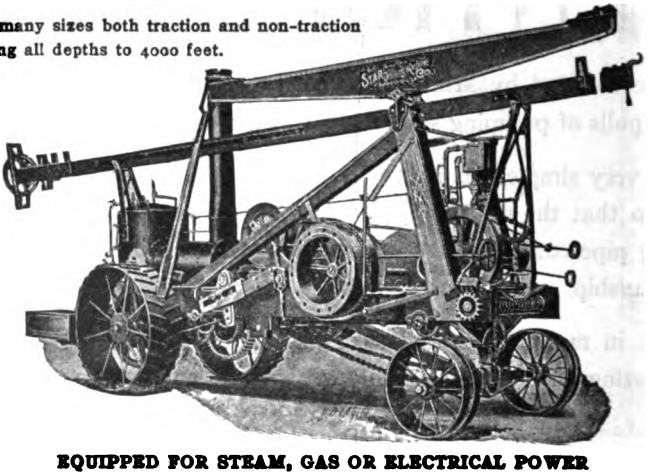

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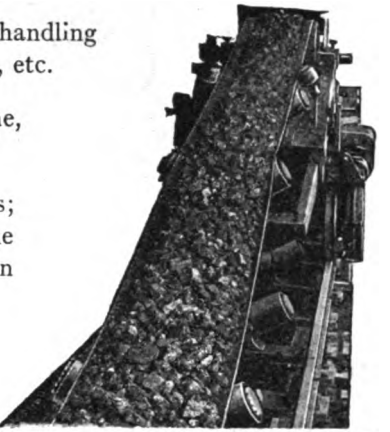

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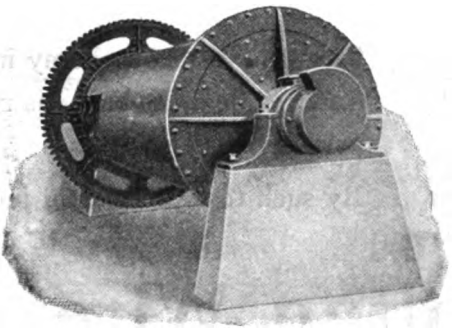

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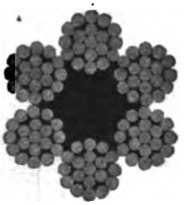

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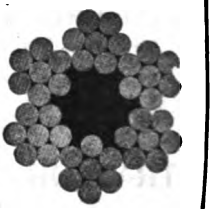

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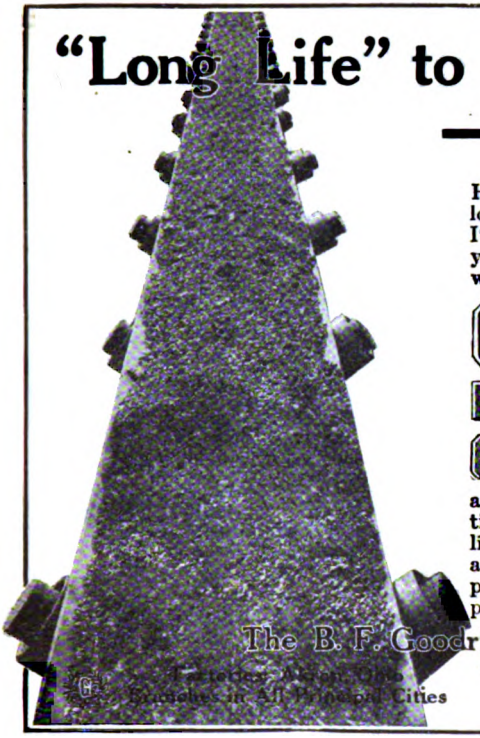

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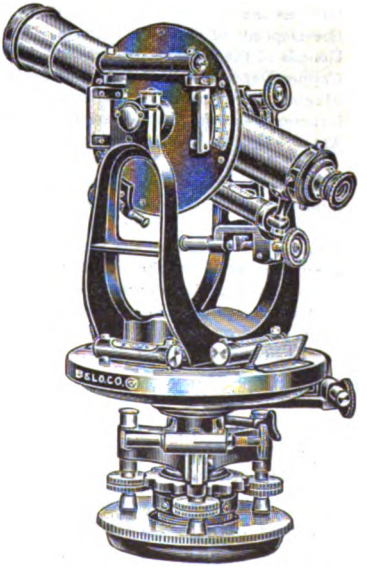


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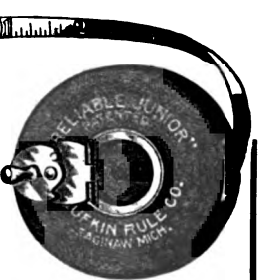


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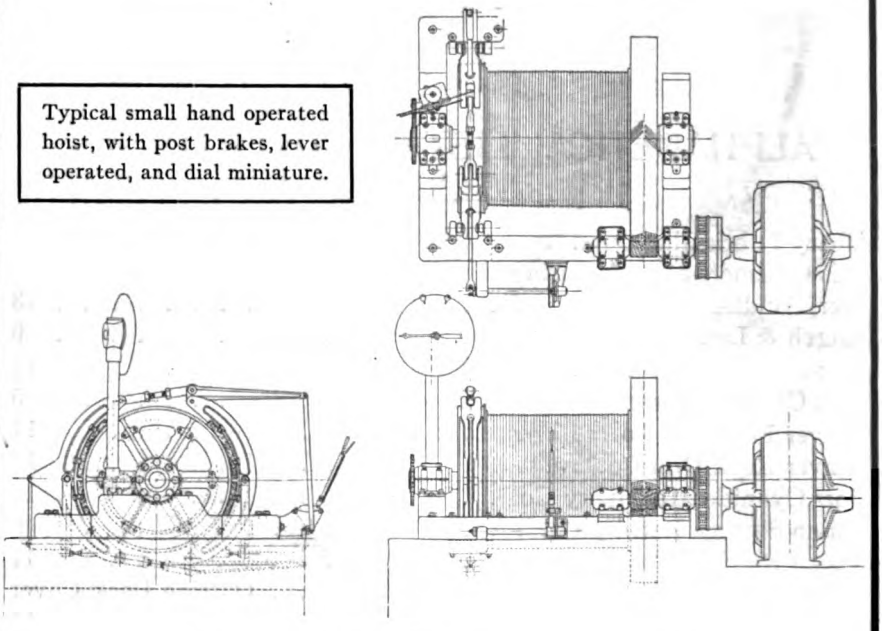
Typical small hand operated hoist, with post brakes, lever operated, and dial miniature.



# Hand Operated Nordberg Electric Hoists

NORDBERG Electric Hoists are built in all sizes from small single drum, hand operated hoists, up to the very largest steam and electric hoists for great depths, loads and speeds.

The above illustration shows a typical small hoist with post brakes, lever operated, and dial miniature. Particular attention is called to the construction of the brakes, which even in the smallest sizes, are of the Post type, operated by a powerful hand lever with quadrant. Also to the gear and pinion at one end of the base, insuring perfect alignment under the most severe service.

**For complete information and detailed drawings of Nordberg Electric Hoists, send for Bulletin 24. If interested in Steam and Air Hoists, send for Bulletin 23.**

NORDBERG MACHINERY

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**Milwaukee, Wisconsin**

**Manufacturers of High Efficiency Corliss Engines; Uniflow Engines; Poppet Valve Engines; Carels-Diesel Engines; Air Compressors; Blowing Engines; Hoisting Engines; Pumping Engines; and other machinery.** 18

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# Every Mine Hoist Condition Analyzed

# Electric Hoisting Equipment Prescribed

THE hoist equipment at many prominent coal and metal mines throughout the world is illustrated and described in the new mine hoist equipment bulletin of the General Electric Company.

All possible mine hoist conditions are analyzed and the correct electric hoisting equipment for each case is described and illustrated.

Complimentary copies of this bulletin, which is just off the press, can be obtained by writing the Publication Bureau, General Electric Company, Schenectady, N. Y., stating where you saw this advertisement.

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